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THE TRAINING OF CHEMICAL ENGINEERS.*

By M. C. WHITAKER.

Much attention is properly being centered upon the education of men in the scientific development and management of agricultural production, and the Government not only gives direct financial support to a large number of schools for such training, but also maintains a Department of Agriculture, the function of which is largely educational.

The annual production of the chemical industries is almost equal in value to the agricultural production of the country, and it is therefore proper to compare the facilities now being provided and the means now being adopted for the education of men to develop and manage the widely diversified units of chemical manufacture with the methods and facilities in other educational fields. Obviously the general educational and technical equipment of men to develop and direct processes or works depending upon the applications of chemistry, and usually involving a knowledge of power and mechanical and electrical machinery, is much greater than the technical equipment required for men in agricultural production. While agricultural education has grown and developed at a rate commensurate with the producing power of that industry, and has been fostered by private, State and Federal funds, education in the equally important and essential field of chemical manufacture has been, in some institutions, entirely ignored, in others only indifferently provided, and in no single case supported and equipped on a basis comparable with the wealth-producing power of the industry.

Efficient production and the economic management of our manufacturing plants are essential features to our commercial development, and it is in this field that the greatest results are to be attained in the conservation of our natural resources. The accomplishment of these results obviously depends upon the application of scientific knowledge to the solution of all problems, both great and small, in the development, the direction and the management of our factories.

The chemical industries very properly look to the schools for men qualified to produce these results, to improve their production and manage their works.

The best evidence that more and better trained men are needed in the chemical industries is the fact that this Institute of Chemical Engineers, composed of men of experience and men now engaged in the direction of great industries, meets twice a year and devotes a large portion of its time to the study of the problem of how to obtain trained men to assist them in the development and administration of their work. Every employer of chemical talent knows that at best the young graduate is only a vague "prospect" and that much has been left undone in his training.

The problem of proper training and correct methods of teaching chemical engineering is one which must be carefully analyzed and studied from its very foundation and is not one to be remedied by a few trivial changes in existing curricula—by dropping this and substituting that. The subject must be opened up and examined in its foundation and if it is found that the present structure is built on sand we should put some "concrete" under it.

Necessarily, the problem must be solved by the co-operation of the teachers with the employer of chemically trained men, and the writer, having served a term of practically ten years in each of the above fields, begs to submit a few observations and conclusions for the consideration of the Institute Committee.

The chemical industries require for their development and management two classes of men of somewhat different training, natural qualifications and range of service.

First: Research chemists who are qualified naturally and by training to originate and to develop in new fields. The training of the research man is more advanced, or supplements that of the chemical engineer, and the research results furnish the foundations for the industries which the chemical engineer organizes and administers. In other words, the research chemist works in the new and undeveloped fields of chemical knowledge and produces results calculated to contribute new knowledge and new arts.

Second: The chemical engineer works in the organization, operation and management of existing or proposed processes with a view to building up a successful manufacturing industry. He uses his classified knowledge of chemistry and the allied engineering branches in developing, perfecting, organizing and administering a plant or a process, for the production of a marketable and useful product at a profit to the investor.

*Paper read before the American Institute of Chemical Engineers.

Such a field as this requires a man of knowledge, originality and resourcefulness. His fundamental training in chemistry, physics, mathematics, etc., must be thorough and must be combined with a natural engineering inclination and an acquired knowledge of engineering methods and appliances. His originality and imaginative capacity seems to me to be essential and a combination result of natural qualifications and training. Resourcefulness, however, comes as a result of experience in application and the ability to adopt a wide range of methods and practices to new conditions. A man with the natural qualifications, supplemented by the proper training, may become a successful chemical engineer.

The success of the research chemist is measured by his scientific achievements, whereas the success of the chemical engineer is measured by his commercial results. While their work bears a certain mutual and cyclic relation it must be apparent that their natural and educational equipment may be entirely different.

The development of specialized training in certain selected lines of manufacture is not a proper function for an engineering school, but should be left to the trade school or to the industries themselves. Specialized research, on the other hand, should be carefully eliminated from the regular engineering training and provided for in graduate courses.

For convenience in discussion, I have divided training subjects for the chemical engineer into three general classes: Fundamental, Associated and Supplementary. No attempt is made to discuss the details of the curriculum, the general purpose being to get broad subdivisions as a starting point. The question of arrangement of subjects into a curriculum and the proper allotment of time for the course depends upon what is finally determined as proper training for the men under discussion.

Fundamental Training.—Chemistry, physics, mathematics, allied subjects.

Associated Training.—Electrical engineering, mechanical engineering, civil engineering, general engineering, business economics.

Supplementary Training.—Study of the applications of the fundamental and associated training in laboratories equipped with the "tools of the trade" and with working plants. Training, by "contact" and by "example," in a laboratory managed as an approved business, in the principles and practices of efficient organization and administration.

FUNDAMENTAL TRAINING.

The training in the fundamental subjects need not be materially changed except perhaps more attention should be given to the laboratory facilities and to the selection of instructors for the fundamental work. Profound scholarship and a record for original research are too often made the sole determining factor in the selection of teachers for these subjects. Research qualifications are of secondary importance in the selection of men to give the fundamental training

to our chemical engineers. Such teachers might better be chosen from men physically, temperamentally and educationally qualified to present their subjects clearly, logically and enthusiastically to the student, and to present them in such a way as to quicken and hold his interest. Too often our undergraduates become muddled, discouraged and even give up a subject because of its dry, disconnected presentation or because the instructor "talks over their heads" or lacks the personality and the power to interest them. Research instructors are often permitted to emphasize their "hobbies" or teach their special experiences without regard to the purpose of the course and its relation to the fundamental training. Our instructors should always be chosen to fit the course requirements and should not be permitted to divert the work into special fields not contemplated in the basic formulation.

The training in the fundamental branches should be more thorough and carefully proportioned than at present, keeping in mind that the chemical engineer's knowledge must be classified, complete and accurate in the broad principles and not statistical, encyclopedic, or of the class of deep profundity required for research in new and original fields. The chemical engineer's knowledge is to be used as a basic "tool" of his trade and may be given as such without detracting one iota from its value as mental training.

ASSOCIATED TRAINING.

Associated training is for the purpose of placing in the hands of the chemical engineer a working knowledge of the other engineering branches, so that he may utilize them in the development and management of his particular problem. Courses designed primarily as fundamental for future advanced study in the same field are not likely to be proper courses for this associated training. Our associated courses should be broad and general and need not carry the student beyond the foundational principles of the subjects with a working knowledge of the existing design, construction, materials, operation and use of engineering appliances.

The existing methods of teaching by lectures, demonstrations and laboratory work, combined with a liberal use of text and reference books is probably the most effective way of reaching the fundamental and the associated subjects.

Assuming that the student now has a thorough fundamental foundation in the basic principles of his profession and a working knowledge of the associated engineering branches schools should be equipped for *directed* study of their *applications* to the problems of manufacture and production. This applied study is classed as "supplementary training."

In the education of teachers, in the education of doctors, in the study of agriculture and in the education of all other engineers except chemical engineers, training in the *application* of the respective fundamentals, under the direction and with the assistance of

experienced teachers, is considered essential. Medical schools without hospitals, teachers' colleges without practice schools, schools of agriculture without experimental farms, or engineering schools without laboratories and shops would hardly be dignified by classification as schools. Such institutions invariably invest large proportions of their resources in facilities for demonstrating the proper practical applications of the fundamental theories.

Is there any reason why chemical engineering should not and could not be taught by the tried methods of the normal schools, the medical schools, the agricultural schools and other engineering schools? Only with proper chemical engineering laboratories, equipped with real working models of standard appliances to illustrate the basic applications of the industries, and a proper curriculum arrangement to permit the student to study and use these applications and the principles involved in their application will we be on a basis comparable with the methods and the facilities now existing in other educational fields.

But we can accomplish still more with proper laboratory facilities. We can so construct and administer these laboratories that the student will be surrounded by an "atmosphere" of manufacturing and business efficiency, where he will learn by *example*, by *daily contact*, by "*attrition*," the modern methods and practices of office and works management. Chemical engineers should have this knowledge. It is fundamental to the proper fulfillment of their job. Instances where failure has been converted into success and industries saved by a knowledge of proper administrative practices and the application of business and works efficiency methods are too numerous to mention. Refinements in chemical processes and ingenious mechanical devices are all ineffective and often useless if inefficient organization and management are permitted to absorb the earnings. Successful chemical manufacturing is based upon three general principles: First, chemical and physical facts; second, mechanical applications; third, organization and management.

The first principle is definite and basic and is, therefore, not capable of variation, but the second and third are variable and may be far from perfection. The greatest opportunity for increased profit, therefore, lies in the possibility of mechanical improvements and improvements in the efficiency of the organization. Instruction is provided in a greater or less degree in existing schools in the two first principles mentioned, but the third equally important and fundamental principle is not even undertaken.

The "contact" system which I am urging for adoption as a method for teaching the principles and the applications of chemistry to manufacture, and for imparting a knowledge of the principles and practices of efficiency in organization, management and administration, are the same as the methods which have always obtained, by force of circumstances, in country

schools where the small children in the A B C classes unconsciously absorb geography, history, arithmetic, etc., by hearing, seeing, and being in the "atmosphere" of the more advanced work. This method can be adopted in the training of chemical engineers and can be made infinitely more effective than in the country school.

The contact or "frictional" method would require a laboratory plant larger and more comprehensive than anything yet established and would have to be managed on somewhat different lines than those now obtaining in engineering schools.

If it is proposed to direct the student in the study of the *applications of fundamentals of factories, factory appliances, factory processes and manufacturing business* by the "frictional" method, the laboratory should be equipped, organized and administered as a live manufacturing proposition. There should be ground space enough to permit of the erection of one or more complete operating plants in addition to the general laboratories for the study of great applications such as distillation, evaporation, filtration, wet reactions, high temperature oxidation, electro-chemistry, etc.

The complete plants should be chosen from some of the important chemical processes of great common interest, which at the same time illustrate a wider diversity of chemical principles and appliances and products—for example, the manufacture of gas.

These plants should be designed and built from the foundation to the roof so that the structure itself, materials used, equipment and arrangement, would illustrate the best known practice and permit of the most economical management and operation. Every structural detail should be provided with a view to *educating by example*. Office appliances, office systems and facilities should be provided and the equipment include books, trade journals, patent literature, samples, advertising matter, house organs, etc., of the business. Conference facilities should be provided in each laboratory for lectures, discussions, demonstrations of the chemical, mechanical and business features of the process. All such work should be conducted in the special laboratories, and the student from the time he enters until he leaves should be made to "stumble over" the tools of his art and become "saturated" with the "atmosphere" of the business from the raw material to the market.

The classes should "go the rounds" of the laboratories in convenient sized sections for conferences and for "manning" the work.

In connection with the chemical engineering laboratory there should be a "shop organization" consisting of competent mechanics, helpers and laborers, necessary for the upkeep of the plant, and this organization should be so administered that our students would gain actual experience in dealing with work-men. Office experience, correspondence, dictation, filing and recording, fundamental costing, etc., may all become

familiar knowledge to the student by having these facilities arranged for his use and by incorporating them in his daily "business."

TECHNICAL ADVISERS.

These laboratories of engineering chemistry should be organized with a system of technical advisers to be selected from representative experts in each typical field of industrial chemistry. These advisers should be specially qualified experts or active works' managers of live concerns. Their connection with the laboratory should not be merely nominal, but should carry with it a responsibility for the proper equipment of the special laboratory of the industry which they represent. They should also give some lectures upon their specialty, not altogether from the chemical standpoint, which presumably can be covered by the regular instructors, but from the business, organization, and relative industrial standpoint.

The personal contact and acquaintance between adviser and student is an obvious mutual benefit. The attitude of the leading industries towards such a chemical engineering school would become one of personal interest instead of one of criticism.

The technical advisory system, if properly initiated and administered, will ensure modern equipment, methods, and instruction in chemical engineering and will give, through the lectures, first-hand and accurate general information on the scope and magnitude of the typical industries. Furthermore, it will definitely dispose of the frequent reference to obsolete and antiquated methods and equipment. By relying upon the judgment of these experts and managers of live and paying industries we would avoid the equally dangerous extreme of becoming loaded up with impractical, untried appliances and uncommercial processes.

INDUSTRIAL RESEARCH.

The largest industrial corporations, such as the General Electric Co., the United Gas Improvement Co., the National Electric Lamp Association, etc., have established and are maintaining research laboratories. These laboratories are directed in their chemical and physical research by able and high-priced men and are equipped and maintained at great expense.

Such laboratories are being established because there are no existing equipments, on a suitable scale, for solving the problems necessary for their industrial advancement. The corporations have not undertaken the burden and expense of academic research from choice, but from necessity. Our scientific institutions have utterly failed to provide the facilities for industrial research, and as a consequence are being "elbowed" aside from their natural positions of technical and scientific leadership.

Under existing conditions the small manufacturer, who cannot afford to equip and maintain a research laboratory, has no prospect of solving his industrial problem. Unless the scientific institutions develop modern chemical engineering laboratories, equipped with the tools of the industries, and place their facili-

ties within the range of the small manufacturers, industrial development will be limited to the few concerns whose means will permit the establishment and maintenance of laboratories for private research.

On the other hand, with such laboratories as I have outlined, equipped with every facility for experimentation on a large scale, factory managers, technical laboratories and individuals could be interested in availing themselves of these facilities and in establishing industrial researches for solving industrial problems.

Frequent reference has been made to the cost of such a laboratory of engineering chemistry as has been outlined. We are dealing with a great industry and a great problem and we should therefore exercise our sense of proportion. It is estimated that the minimum *net profit* of chemical manufactures is \$300,000,000 annually. It is also estimated that the combined investment in chemical engineering equipment in all of the educational institutions of the country is not over \$30,000, or one-hundredth of one per cent of the net profit of the industry for one year. A \$30,000 investment or even one hundred times that is an absurdly small equipment to be devoted exclusively to instruction in the application of chemistry for such an enormously profitable industry, not to mention the facilities for research development. If the plan outlined will produce men better qualified to enter this field of industry and men who can "deliver the goods," I have no fear that the question of cost will ever prove an obstacle to the development of such laboratories of chemical engineering.

The world's output of coal for 1909 is estimated at 1,110,000,000 tons, an increase of 42,000,000 tons over 1908. Output increased by 35,000,000 tons in the United States, and 2,300,000 and 2,100,000 tons in Great Britain and Germany, respectively. The United States produced 37 per cent of the world's output, Great Britain nearly 25 and Germany 20 per cent; the three countries together 81 per cent.

The tanning methods employed in certain parts of China, according to a recent consular report, are as follows: The hide is put into a small pond and immersed in a thick mixture of alum and water. Fresh hides are taken out after two days, but if they had been dried under the sun they are kept in the pond for a number of days until soft and flexible. The flexible hide is then placed on a broad bench and slowly but rather skillfully peeled by hand labor into thin sheets, which are expanded by bamboo sticks bent into bows. A number of sticks applied to one sheet keep it expanded like a drum. It is dried by a straw fire in an earthen stove with a conical cover having an opening on top for the emission of smoke. This also gives the leather a yellowish color. The process is completed by exposing it to the sun's heat until thoroughly dried.

THE SUPPLY OF IRON.*

By JAMES F. KEMP.

The plan of preparing an estimate of the world's resources in iron ore has had the cordial support of the Prime Minister of Sweden and it is a pleasure to acknowledge our indebtedness to him. The plan has been ably carried out by the Swedish committee, and special acknowledgments are due to Dr. J. G. Andersson for the able way in which the difficult task of editing has been performed. As a result we have before us a work which will furnish much food for economic speculation and thought. When we try, however, to project ourselves into the future, we soon find that we are not able to speak with the accuracy of mathematics, but we must be content with the attainment of conclusions which, within certain limits, are not unduly remote from the truth. The subject is a complicated one, and may perhaps be best discussed after the mathematical form of assuming, so to speak, an equation which is a function of several variables. By taking one variable at a time and following out the effect of its changes upon the general function, we may, by a series of approximations, in the end establish a solution worthy of confidence.

The future of the iron industry is a function of the following and perhaps of other variables:

1. The increase in annual production which will make greater demands on the reserves.
2. The decrease of some of the present sources of supply and their final exhaustion.
3. The entrance of new sources of supply now perhaps inaccessible.
4. The gradual decline in average content of iron in the ores used.
5. The increase of expense of production with declining yield.
6. Improvements in existing processes which reduce expenses.
7. The supply of fuel; above all, under present circumstances, of the coking coals.
8. The entrance of new methods of smelting, especially those depending on electricity.
9. The substitution of other materials for iron and steel; among which cement is most important.

We, delegates to the congress, who have come across the ocean, have gazed with profound interest upon the collections in the National Museum which so graphically portray the history of the Scandinavian peoples from the Stone Age, through the Bronze Age, into the Iron Age. It is the Iron Age which many people believe to be the one in which we live. This belief is only partly true. We really live in the Age of Steel, not of Iron, and it is very important that we should so regard our own times. Almost all the pig iron produced is used in the manufacture of steel. In the United States, whose output of pig iron the

past year was over 25,000,000 tons, and was more than that of the next two nations combined, approximately 90 per cent of the yield of the furnaces is marketed in the form of steel. This ratio will probably continue until we pass from the Age of Steel into that next age of the human race, the Age of Cement, a period which already looms above the horizon and which bids fair in no unimportant way to prolong the life of our iron ore reserves. For many structural purposes reinforced concrete has already replaced steel and has even been used in the construction of ships.

The report on the iron ore reserves makes it clear that there will be no exhaustion of iron ore of one kind or another for an indefinite period of time. If we consider iron-bearing materials of 30 per cent as ores, there is no lack of a supply for our furnaces. Much apprehension has been felt on this point, and it is reassuring to find our fears in a measure unwarranted.

Iron, as a metal, of course, never can fail, for if we drop 20 or 10 per cent, we find inexhaustible amounts of basic rocks with these percentages, and we are then dealing with raw materials from three to ten times as rich in iron as are the usual copper ores in copper, the next metal to iron in abundance of production.

But the existence of iron ores of even greater yield than 30 to 40 per cent is one thing, and their availability under ordinary conditions of smelting is another. Iron must be produced at low cost if it is to continue to fill its present place in modern civilization, and thus the sources of supply are restricted to comparatively few regions. Almost all the iron and steel of the world is now produced in Western Europe and North America, roughly about 60 per cent in the former and 40 per cent in the latter. In North America the industry is almost all in the eastern part of the country, whether it be in Canada or the United States. If, therefore, we consider the actual facts of production, we necessarily restrict ourselves to a comparatively small part of both Europe and North America. Again, if we consider the population of the world at large, and the population of the countries that produce the iron and steel, the latter is a small fraction.

When, therefore, we speak of the world's reserves of iron ore, if we have in our minds the entire surface of the globe, we are discussing a matter that is largely of academic interest. But if we come down to the business of producing iron and steel as an economic matter, we must eliminate from consideration as affecting the large features of the immediate future all but that small part of the world that lies in Western Europe and Eastern North America.

What, then, is the outlook in these larger producing areas?

Of the United States I can speak with most intimate knowledge, but they are far removed from the

*A contribution to a discussion on "The Iron and Steel Industry of the World," held at Stockholm, August 1-10, 1910.

countries of Europe, and the iron industry has not yet developed international features except in the importation of ore. The larger features of its future can be summarized in a few words. There are two principal and several minor sources of supply; the Lake Superior region and the Alabama region comprise the former; the Appalachian mountains almost all the latter. The Lake Superior region furnishes four-fifths of the American ores, and one state, Minnesota supplies more than one-half. The ores are low in phosphorus and sulphur, relatively high in silica, and are very cheaply mined. The ores formerly yielded 60 per cent and over in iron. Now they average somewhat over 50 per cent. At 40 per cent, which is still comparatively high, considering the world at large, the amount is inexhaustible. The ores are transported from several hundreds to over one thousand miles in order to meet the coke and be smelted to the best advantage.

In Alabama, and to a less degree in neighboring States, the ores formerly yielded from 40 to 50 per cent, but the most important variety has now settled down to about 37 per cent, a yield which will not essentially change for many years to come. The ores thus are comparable with the minettes of Germany and France. The coal, the limestone and the ore are all within a few miles of each other, so that the low cost of production is not equaled elsewhere in America. The ores were silicious in early times, when they were obtained near the surface. In depth they have become basic. They are stratified and can be estimated as accurately as coal seams. The ores are moderately high in phosphorus, much above the acid-bessemer limits. Other sources of supply contribute lesser amounts of ore in the Alabama region, and still additional sources are found along the Appalachian mountains, but the chief producers have been reviewed above. The reserves are sufficient for many years to come.

West of the Mississippi the iron industry is relatively small; one moderately large plant is in operation at Pueblo, in Colorado, and another is just being completed on Puget Sound. The development of Alaskan coal will be an important factor in the future.

The most interesting feature of the next few years will depend upon the development of the new supplies of ores from the northern side of Cuba. Great deposits have been proved by the drill. They lie upon the surface and will be cheaply mined by steam shovels. When calcined and freed of their abundant moisture they will supply an ore above 40 per cent in iron, almost entirely free from phosphorus and sulphur, but containing chromium and nickel oxides. The anticipation is entertained that they will greatly revive the steel industry on the Atlantic seaboard, if they do not also lead to the construction of new and extensive plants.

In America, therefore, so far as ore is concerned, there is no prospect of a serious change for an indefinite period. America has been a relatively new country of great distances. Means of transportation calling for steel have been absolutely essential to development. Steel is now employed also in enormous quantity for structural purposes, since all large buildings—the “skyscrapers”—have a framework of steel; but cement is being more and more utilized and will probably in time affect the consumption of steel in no unimportant way.

Aside from America, in particular, I can only speak of the iron industry in general terms.

In forecasting the future the first disposition is to divide the tons of reserves by the annual production and thus estimate the life of the mines. I am, however, reminded that in the last 50 years the production has increased by leaps and bounds, as Mr. Van Hise made clear in his address at the opening of the Congress. This rapid increase is true not only of the United States, but especially in later years of Germany. An element of uncertainty is thus introduced into the first simple calculation and the life of the reserves is correspondingly shortened.

On the other hand, in the older iron-producing countries the railways are now built; the great public improvements are largely completed, so that for these purposes the supply of iron or steel is not now required. The iron, moreover, already in the metallic state steadily accumulates for reworking. We cannot anticipate the continuance of the geometric rate of increase. We are more likely to attain a fairly stable annual output. Just what this production will be, it is difficult to state, but the supply of fuel of which I shall speak in a moment is an important factor.

Undoubtedly some of the present sources of supply will decrease and finally will be exhausted. Of this feature I am inclined to speak with reserve because most of us have thought that the iron ores of England and the Bilbao ores in Spain were approaching exhaustion, whereas we have learned from the reports to the Congress that new reserves have been opened up by further exploration; the Bilbao ores, for example, can scarcely be regarded as nearing their end, as over sixty millions of tons remain to be utilized. Nevertheless there are doubtless some present producers which before long will cease to ship. For the furnaces thus deprived of their old sources of supply, new ones must be found. Apparently in these relations Sweden, Algeria, Newfoundland, Cuba and Brazil will come more and more to the front. Assuredly there is a future of increasing importance for the exporting countries. Nevertheless as local sources of supply decline we must anticipate tariffs and legislation looking toward the conservation of natural resources; these will operate to postpone exhaustion. There are thus antagonistic influences which must be balanced in any forecast worthy of confidence.

In some important producing areas, especially in the United States, a gradual decline in the average percentage of iron in the ore is inevitable. The decline is most marked in those countries where the richest ore has been naturally the first to be utilized. But the percentages, although lower than those hitherto available, will soon be fairly stable in America, and they have already practically reached this condition in Germany. Changes will then only be apparent over long periods of years. With the lowering of percentages, and an amelioration of the conditions of production, a wider range of ores becomes available, so that lowering is checked. There is thus a delicate and interesting balance maintained by these influences.

With the lowering of the percentage of iron in the ore, the cost of production naturally increases. More barren material passes through the furnace, with decreasing product, and increasing expense for fuel. From this cause alone one might infer a decrease in output. But, on the other hand, improvements in processes of smelting have from time to time appeared and have more than neutralized these effects. In the United States, for example, furnaces have grown enormously in size and capacity. The recent invention of my countryman, James Gayley, has tended greatly to increase the efficiency and the uniformity of operation. Mr. Gayley passes the incoming air of the blast through a series of coils, cooled below the freezing point of water by a very cold solution of some salt such as calcium chloride, so that all the moisture is chilled from the air, which then enters the stoves in a thoroughly dried condition. The accumulated moisture, frozen to the coils, can, when necessary, be melted off by the outgoing heated gases, while the incoming air traverses a second cold chamber. By such an improvement as this or by others likely to be made as time goes by, the effects of decreasing percentages have been offset and again a delicate and interesting balance of opposing forces has been established.

Let us now assume that essentially the methods of smelting which are practiced today will remain in force for a long period of years. The iron ore goes first to a blast furnace in which, in the vast majority of cases, it is reduced by coke to pig iron. Roughly speaking, one ton of coke is required to produce one ton of pig iron. For the one ton of coke about $1\frac{1}{2}$ tons of raw coal are necessary. If our ores yield 50 per cent in iron, then for each two tons of ore $1\frac{1}{2}$ tons of coal are utilized, and that coal must possess superior qualities in order not to crush down under the heavy burden of a modern blast furnace. In so far as the future of the iron industry is concerned, the serious question is not alone the supply of iron ore, but also the supply of coking coal. It will not suffice in these relations to say that this country or that has so many thousands of square miles or kilometers of coal fields, because only a relatively small part of the coal fields yields the right sort of coal. We must know how

much of the latter the world possesses and then how much of this supply is near enough to blast furnaces to be utilized without excessive transportation. The coal supply would seem to me to be the next question which a future International Geological Congress might most appropriately take up. Not until these data are at command can the remoter future of the iron industry be suitably forecast.

But, may we ask, are our present methods of the reduction of iron destined to continue? To this it may be replied that with the present outlook electric smelting alone seems likely to introduce changes, and so far as it can be employed only to the following degree: In the reactions of a blast furnace about two-thirds of the fuel is employed in supplying the necessary heat and one-third in the reduction of the iron oxide. Electrical energy, supplied by water power, may replace the two-thirds of the fuel as a source of heat; the remaining one-third will always be necessary. In countries where there is abundant and practicable water power, electric smelting may thus be feasible. Charcoal may even in some furnaces entirely replace coke, over which, being a purer source of carbon, it has manifest advantages. Poor charcoal such as may be obtained from refuse lumber may perhaps suffice, so that to some extent the demands upon the coking coals may be reduced and the life of the reserves may be prolonged. Obviously in countries such as England, with limited water power, but with great supplies of coking coal and great iron interests, electric smelting can be of small importance except in so far as it may enable other lands, such as Scandinavia, to export to it some supplies of iron, necessarily furnished at sufficiently low cost. Electric smelting is in its infancy as yet, but it is of sufficient promise to deserve mention as a possibly important factor in the future.

The final variable in our function which calls for brief mention relates to the substitution of other materials for iron and steel as time goes by. Of these cement seems to be the one of most serious moment and only in so far as it may supplant structural steel. In the form of reinforced concrete cement is a material which grows in favor and in use. Some steel is of course used in its manufacture, and the cement is thus only in part a substitute, but its potentialities are sufficient to justify its mention. No one can accurately estimate its future influence.

In conclusion I may repeat that the subject is complex. That the future of the iron industry is a function of a number of fairly independent variables is clear. Some of the variables neutralize each other. Some of the variations we cannot fully trace. On the whole, for many years to come, I believe the iron industry will continue without any very fundamental changes.

Pearl shells valued at \$844,670 were exported from the northwest coast of Australia last year.

THE DEVELOPMENT OF THE CHEMIST AS AN ENGINEER.*

By FRED W. ATKINSON,† PH. D.

The comprehensive circular letter No. 1 to the members of the Committee on Chemical Engineering Education of the American Institute of Chemical Engineers presents clearly and concretely a problem of vital importance to the engineering educator as well as to the manufacturing chemist. The subject is, indeed, one to which little explicit attention has been paid. In any attempt to formulate a definite and practicable program for the training of chemical engineers this society will be obliged in a certain sense to beat out its own path. The situation regarding such training is still chaotic and the technical institutions are by no means agreed among themselves as to the most suitable course of education for a chemical engineer. They will welcome heartily a report from this body of practical men which will point out the causes, results and deficiencies of the present method of training and which will suggest courses likely to produce better results.

I note from my examination of the membership cards of this association that a wide diversity of data is displayed under the headings: "Present Occupation and Business Connection," "History and Education," and "Career in Chemical Engineering." I should infer from the facts exhibited on these membership cards that very few of the members of this institute had had any formal training in engineering subjects and that what they had achieved as chemical engineers was due entirely to their own individual development in response to the practical demands made upon them as chemists. Many of them bear the stamp, like many another excellent product, "made in Germany," which implies that they brought to their profession a knowledge of the modern advance in chemistry and an appreciation of investigation. If I as an outsider may presume to make a suggestion it would be that every member of this institute be given an opportunity to express his opinion with reference to the course in chemical engineering which will produce graduates who shall be so trained as to meet the requirements of the chemical industries in America. In any expression of views upon the general subject, it is important to keep in mind that in technical schools it is not feasible to assign more than 15 hours of lecture or recitation, and 15 hours of laboratory work per week for four years, or, at the most, for five years. A larger weekly assignment only results in poor preparation, giving a smaller net return in training and acquisition of knowledge.

It should also be clearly recognized that it may be possible to develop in the technical school or the engineering department of the university qualities or abilities which are very necessary in the factory, and that

a portion of the education or training of the chemical engineer must always be obtained during a period of apprenticeship in chemical works. High proficiency in chemical engineering can never be reached without natural ability and long experience. The chemical engineer forms no exception to the law that experience is an imperative necessity for every human being. A college can no more turn out completely developed and efficient chemical engineers ready for positions involving superintendence and responsibility than it can produce lawyers and doctors expert at the day of their graduation.

After the views of the members have been obtained and the consensus of opinion of this institute has been formulated which will be no easy task, I suggest as perhaps the best method of approaching the educational institutions that the final report with definite recommendations be submitted to the Society for the Promotion of Engineering Education for consideration by its members, most of whom are engaged in college instruction. A joint meeting of the two societies might also be arranged, for if this problem is ever to be solved successfully it will be by bringing the chemical manufacturers and the college instructors in more intimate and more co-operative relations.

As a result of a popular demand by far the larger number of our colleges are attempting in a four-year course to provide the instruction required by the prospective chemical engineer. This is too short a course. The chemical engineer is a combination of the chemist and the engineer, and I believe it is the fundamental mistake of the colleges that they do not insist upon at least five years for the completion of a course in chemical engineering. A five-year requirement is bound to come and should this institute reach the conclusion that the usual period of four years is too brief for an adequate preparation I believe its recommendation to extend the time will have great weight with the educational authorities, and will tend to accelerate the movement in the direction of a more suitable training.

In either a four or a five-year course the proportion of time devoted to chemical subjects should exceed that given to engineering subjects. As a primary requisite the chemical engineer must possess a broad, comprehensive knowledge of chemistry, theoretical and applied. Although I have no doubt but what some of you may disagree with me, yet realizing as I do the practical conditions governing college instruction it is my opinion that this chemical training must necessarily be general and fundamental rather than specialized. Instruction in chemistry should continue through the whole length of the course. The freshman year of such a course will be identical to that offered to prospective chemists and very similar to that pursued by engineering students. English, French or German, mathematics, physics, drawing and general descriptive chemistry are the usual subjects of the first year. We find at the Polytechnic Institute that about 30 per cent of the students are able at the time of entering to pass examinations in trigonometry and higher algebra, and

*Read at the third annual meeting of the American Institute of Chemical Engineers, held at Hotel Astor, New York.

†President of Polytechnic Institute of Brooklyn.

in chemistry. Such students are able at once to take up more advanced work in mathematics, and to begin their qualitative analysis in the freshman year. In the second year, English, modern language and physics, should be continued; some period of history should be studied; and in mathematics, the calculus should be undertaken. Qualitative analysis should perhaps come the first and quantitative analysis the second half year. With the exception of the introduction of the calculus the second year of the five-year chemical engineering course of the Polytechnic differs but little from the second year of its chemical course. In the excellent four-year chemical engineering course of the Massachusetts Institute of Technology, I note that the engineering subjects of mechanism and mechanical engineering drawing are introduced in the second year. The subjects, machine drawing and surveying, appear in the second year of the course printed in circular letter No. 1 to members of the Committee on Chemical Engineering.

This difference of practice in the time of introduction and in the character of engineering subjects becomes greatly accentuated in the third and fourth years of the courses offered by our American educational institutions. It may safely be said that no two institutions have from among an almost bewildering variety of engineering subjects made even approximately the same selections. Local conditions will of course always be one of the determining factors in the shaping of these courses and a free rather than a standardized development of professional courses is the usual American educational experience. Nevertheless, I believe that greater care in the selection of engineering subjects based on actual experience which the members of this institute can furnish is both desirable and possible. The hopefulness of the situation lies in the fact that the members of this Institute have undertaken this study and are in a position to indicate to the college authorities just what changes in the present courses are necessary to bring about the desired results. Chemical engineering needs to be more sharply defined. Its scope is still in a somewhat indeterminate state and as yet its position as one of the professions is not clearly recognized.

The present courses are founded on the assumption that the chemical engineer must, beside being a chemist, also be something of a mechanical engineer, and to a less degree something of an electrical engineer. It is further assumed that the chemical engineer will not be expected to show results so much as an analytical chemist or as a student of special research who spends his time in the laboratory of the manufacturing plant, but as a managing chemist who remains in the laboratory only long enough to get his business bearings. Broad expert knowledge and an inventive ability combined with a natural aptitude for administration and management, are the principal characteristics of the successful chemical engineer. I should like to know if these assumptions are true, for if they are it is at once evident that the training for

such a profession must be broadly fundamental rather than narrowly specialized.

The colleges are pretty well agreed upon what shall be done in the fundamental branches of mathematics and physics. Solid geometry, advanced algebra, trigonometry, descriptive geometry and the calculus are the mathematical subjects taught, and the physics course of two years' duration is mainly devoted to the subjects of mechanics, light, heat and electricity. English is sometimes limited to the work in the preparatory school, but as a rule it is taught for one year, and often for two years, which is none too long a time. It is of the highest importance that the chemical engineer be able to express himself clearly and concisely. Economics is becoming recognized as a desirable and necessary subject for all engineers. The other general subjects to be included such as history, logic and business law, depend upon the length of the course whether it is a four or five years' course. The colleges are further agreed that mechanical engineering drawing should be included and that there should be also thorough fundamental courses in mechanism, applied mechanics and thermodynamics.

There are many points upon which the colleges are not agreed, and a few of the more important of these, I am presenting in the form of questions to the members of this institute.

1. Are both French and German essential to the chemical engineer?
2. Of what should the course in applied or technical chemistry consist? Specify the subjects to be taken and the proficiency to be acquired in each.
3. How much investigation should be required and of what character?
4. Beside mechanism, applied mechanics and thermodynamics what mechanical engineering subjects should be included? Steam engineering? Machine design? Steam power plants? Gas power? Works management?
5. What electrical engineering subjects should be included? Principles of electrical engineering? Dynamo laboratory? Electrical measurements?
6. What civil engineering subjects beside hydraulics should be taken? Surveying? Mechanics of materials? Testing laboratory?
7. What in your opinion is the main cause of the failure of the present courses in chemical engineering to "produce men who are prepared in the best possible way for practical work in the manufacturing establishments of the United States"?

A movement is being made in England toward buying and selling leather by measurement, instead of by weight, and the authorities are being urged to recognize the measuring machine as a standard means of sale on the same basis as is now the rule with weighing machines and to provide for the proper registration and examination of leather-measuring machines.

THE INDIAN MAGNESITE INDUSTRY.*

By H. M. DAINS, F. I. C.

The first idea of the possibility of starting a magnesite industry in India seems to have originated with Dr. Macleod, who experimented with the Salem magnesite and reported on it to the government of India in 1826. He suggested the use of it as a substitute for imported cement. It is only within recent years, however, that work on anything like a large scale has been carried on.

The principal deposits lie in the "Chalk Hills," two miles from the town of Salem in the Madras Presidency, 200 miles west of Madras, and an equal distance from the port of Beypore on the West Coast. These ports are connected by the Madras Railway, which passes within a mile of the deposits. The magnesite covers a superficial area of about 2,000 acres and occurs in abundant irregular veins of unknown depth, which exceeds 60 ft. The veins intersect an ultrabasic intrusion of eruptive rock called dunite, after its prototype, native and the Dun Mountain, New Zealand.

This rock, which is ultra-basic on account of the absence of felspar and quartz, has undergone a series of remarkable mineral changes owing to the non-resistance of the unstable olivine to metamorphic influences. The change resulting in the formation of magnesite may be expressed somewhat as follows:

ANALYSIS OF DUNITE (sq. gr., 3.176).	
	Per Cent.
Silica	39.10
Magnesia	48.26
	87.36
Iron, alumina, manganese, chromium, moisture, etc.	12.64
	100.00
Dunite ¹	
Olivine	Chromite
(containing a little iron)	
Serpentine	
(A number of varieties, e. g., Massive, Retinalite, and Picrolite)	
Magnesite	Silica, etc.
Chalcedony.	
Dunite from Dun Mountain (Von Hochstetter.)	
	Per Cent.
Silica	42.80
Magnesia	47.38
Ferrous oxide.....	9.40
(After the chromite had been removed.)	
Oriental olivine (sp. gr., 3.351) (Duna) :	
	Per Cent.
Silica	39.73
Magnesia	50.13
Ferrous oxide.....	9.19

¹Report on the magnesite areas near Salem, by C. S. Middlemiss. (Geological survey of India.)

Serpentine is found widely disseminated throughout the deposits associated with the magnesite, but for the most part in small quantities. The magnesite varies slightly in appearance and composition. The outcrops are of a grayish color and intensely hard. Under the surface it becomes softer the greater the depth at which it is found, and in some cases where water has had access to the veins it can be almost dug out with a spade, and crumbles readily in the hand.

*From Journal Society Chemical Industry.

Generally the magnesite is almost perfectly white. It has a sp. gr. of 3.0 to 3.06, and a hardness of about 5.

The average mineral is of excellent quality, being exceptionally low in lime, silica, iron oxide, and alumina. It contains about 96-97 per cent of magnesium carbonate, but picked specimens have given as much as 99 per cent.

INDIAN MAGNESITE					
	Blount.	Dains.	Pattison Cargo Sample.	Ferguson -- 1.	2
	Per Cent.	Per Cent	Per Cent.	Per Cent.	Per Cent.
Silica	0.22	0.29	1.17	0.31	1.70
Iron oxide	{ 0.30	0.65	0.14	{ 0.40	0.65
Alumina				{ 0.10	0.10
Magnanese oxide.....	..	0.20	0.06	..	..
Lime	nil	0.83	0.78	..	..
Magnesium oxide.....	47.35	46.42	46.28	..	..
Carbon dioxide.....	51.44	50.71	50.10	..	..
Combined water.....	0.27	0.16	1.30	0.60	traces
Sulphuric acid.....	..	..	0.03	..	..
Phosphoric acid.....	..	..	0.01	..	..
Magnesium carbonate... (98.79)	(97.13)	(96.34)	0.85	97.40	..
Calcium carbonate.....	..	..	0.85	trace	..
Alkalis	..	..	..	..	nil
	99.58	99.26	99.87	100.06	99.85

Other sources of supply where magnesite is worked are Euboea in Greece, Styria, California, Canada, South Africa and Macedonia. The Greek magnesite most nearly approaches the Indian variety.

Magnesite in the raw state is used chiefly for the manufacture of carbon dioxide. In the United States of America, magnesite is very largely used for this purpose, the gas being obtained by calcination in retorts. The residue is sold to makers of refractory bricks by whom it is made up into bricks for basic furnaces. In some works the carbon dioxide is evolved by the action of sulphuric acid.

Calcined magnesite is of great commercial value. The calcined material may be classed according to the temperature at which the calcination takes place, as: (a) Lightly calcined, or caustic magnesia; (b) dead burnt, sintered, or shrunk magnesia. The caustic magnesia is obtained by calcining at a temperature of 800° C., and the process is best carried out in a gas-fired kiln. It is largely used for Sorel or oxychloride cements, fireproof partitions, plaster, artificial stone, steam packing, flooring, grindstones, millstones, emery and polishing wheels, etc. At Salem the caustic magnesia was first obtained by calcination in ordinary lime kilns. These, however, gave an inferior product owing to the contamination with fuel ash. A regenerative gas-fired kiln has now been erected and is working satisfactorily, the material produced being a great improvement, both as regards color and quality, on the old bottle-kiln material.

The standard quality stipulated for by continental consumers at the Leipzig meeting, December, 1908, is 4 per cent, conditions which the Indian magnesite easily fulfills.

Sorel cement is formed by mixing the caustic magnesia with a solution of magnesium chloride of sp. gr.

1.162 to 1.203. This cement is very hard, white, and of great durability, and will carry up to 20 parts of sand for one of magnesia. When used with sawdust as an aggregate it makes a practically noiseless and dustless flooring.

Dead-burnt of shrunk magnesia is obtained by calcining at a temperature of not less than 1,700° C.

At Salem the calcination is done in a Schneider kiln,

	INDIAN CAUSTIC MAGNESIA.					
	From Old Kilns.			Gas-Fired Kiln.		
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Loss of ignition	4.62	1.82	5.76	1.82	2.31	
Silica	1.64	1.72	1.36			
Insoluble residue				1.13	0.54	
Oxide of iron and alumina	1.02	0.91	0.71	0.63	0.41	
Lime	1.53	2.43	2.25	1.06	1.03	
Magnesia	91.03	90.18	90.28	95.80	96.10	
	99.84	100.09	100.39	100.44	100.42	

but it would be preferable to calcine in a gas-fired kiln to avoid contamination with the ash from the fuel. The product thus obtained is very basic, and has a sp. gr. of 3.5. It is free from carbon dioxide, and has practically no tendency to absorb water.

GREEK MAGNESITE.

	(J. Chem. Soc., Jan., 1903.)		
	Per Cent.	Per Cent.	Per Cent.
Magnesia	46.09	45.75	
Carbon dioxide	51.51	49.88	
Magnesium carbonate	97.60	95.63	93.70
Silica	0.38	1.63	2.20
Lime	1.68	1.44	
Alumina	0.15	0.17	0.10
Iron oxides	0.08	1.19	0.07
Calcium carbonate			4.02
Combined water			traces
Carbonate of iron			

The principal uses of dead-burnt magnesia are as a refractory lining for open-hearth furnaces and converters in the steel industry, for rotary kiln linings in Portland cement manufacture, for furnace hearths, crucibles, cupels, and refractory bricks. The most re-

	INDIAN DEAD-BURNT.		Gas-Fired Kiln.
	Schneider Kiln.	Per Cent.	
	Per Cent.	Per Cent.	Per Cent.
Silica	4.92	5.59	4.38
Ferric oxide			
Alumina	2.74	2.20	1.02
Lime	1.92	1.67	1.04
Magnesia	89.53	89.83	93.12
Loss of ignition	0.39	0.34	0.34
	99.50	99.63	100.00

fractory dead-burnt magnesia is obtained from magnesite containing little or no lime, silica, oxide of iron, and alumina. The Indian magnesite is exceptionally low in these deleterious compounds.

The presence of much lime in magnesite bricks used for high temperatures causes them to disintegrate, and in basic steel furnaces is said to cause the phosphorus to pass into the hearth instead of the slag, the hearth thereby becoming "rotten." Silica, oxide of iron, and alumina are objectionable because they lower the fusing point.

For dead-burnt magnesite a high temperature of calcination is required as it is absolutely necessary not only to drive off all the carbon dioxide but to thoroughly shrink the resulting magnesia.

Styrian magnesite is largely used for making refractory bricks, but it contains a high percentage of iron oxide.

	STYRIAN DEAD-BURNT.	
	Per Cent.	Per Cent.
Silica	2.20	3.40
Iron oxide	7.70	
Alumina	1.02	7.60
Lime	4.92	4.05
Magnesia	83.92	84.29
	99.76	99.34

Dead-burnt magnesia has replaced dolomite to a great extent in basic furnaces because it is not hygroscopic and can therefore be kept any length of time without deterioration, whereas owing to the high percentage of caustic lime in dolomite this material rapidly deteriorates on exposure. Dolomite is useless after being once used and, whereas 550 to 1,100 lbs. on an average of this calcined material is required for repairs after each open-hearth heat, with dead-burnt magnesite only 110 to 220 lbs. is required.

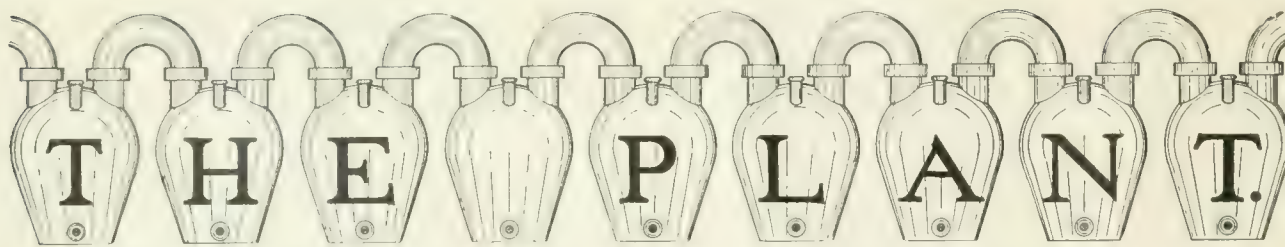
Experiments have been carried out in the calcination of the Indian magnesite on a large scale in electric

	American (Mineral Industry, 1902).				Hungarian.	Silesian.
	(Ferguson.)	Per Cent.	Per Cent.	Per Cent.		
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Magnesia						46-48
Carbon dioxide						46-50
Magnesium carbonate	94.00	93.00	93.66	94.80		
Silica	3.10	2.00	2.76	0.80		4.5-5.25
Lime						0.6-0.7
Alumina	0.80	0.50		1.10		
Iron oxides	0.30	0.07		3.20		1.5
Calcium carbonate	1.98	0.49	2.78	0.10		
Combined water	0.65	0.40				
Carbonate of iron			0.40			

furnaces. In this process the magnesia becomes partly fused, and the resulting product is a crystalline mass of thoroughly shrunk material having a specific gravity of 3.58. The crystals have the same well-defined cubic cleavage as periclase—the natural crystalline magnesium oxide. This crystalline material has been used with much success as a wash and a lining in electric furnaces.

Bricks are usually made in this country by grinding the dead-burnt magnesite in an edge-runner mill, mixing with water in sufficient quantity to make a plastic mass, and molding in an ordinary hand brick press. The bricks after careful drying are burnt in a high temperature kiln. Another process, however, is to grind the crude material and then add water to enable the crude to be molded and burnt. The usual practice on the continent is to mix the ground calcined material with tar, mold, place in a hydraulic press, and after drying burn in a gas-fired kiln. The volatile portions of the tar escape, leaving in the interstices a carbonaceous residue which binds the whole together.

	INDIAN MAGNESITE BRICKS (S. P. GR. 3.58) (WESTMORELAND)	
	(1)	(1)
	Per Cent.	Per Cent.
Silica	2.60	3.15
Iron oxide	1.91	0.53
Alumina		0.23
Lime	1.53	1.61
Magnesia	93.94	94.28
	99.98	99.80



COKE IN FOUNDRY SMELTING.*

The writer has had a part in the efforts of associated foundry men from the beginning of what may be called the new era of the industry. He has listened to many experiences related by veterans and by men newer to the business. All these experiences, including the flowing of enormous quantities of the slag out of the cupola doors on the charging platform and the disappearance of incredible quantities of metal during the melt, have strengthened his belief that troubles laid to the idiosyncrasy or, as some call it, the "innate cussedness" of the cupola, could really be avoided by changing the melting program to suit the fuel that is used.

Normal coke has 50 per cent cell space and has, pound for pound, practically double the volume of anthracite. Hence during the melt the downward movement of the metal within the range of the melting zone must be twice as fast when coke is used as it is with anthracite. Therefore the difference between the first of the metal to melt and the last, with respect to the influences to which they have been exposed, is greater for a bed of coke as compared with a bed of anthracite of like weight. That is, the first metal to melt is likely to be the same on either the coke bed or the bed of anthracite, but the last metal, especially if the charge is very heavy, melts much lower down in the coke-charged cupola and is necessarily subjected to greater oxidizing influences, because the blast low down in the cupola contains more free oxygen. When anthracite is used the part of the bed where the melting is actually done varies least in position.

This advantage is still retained by many foundrymen in the East, where anthracite, as it is still available and no dearer than coke, may be used for the fuel bed and often mixed with coke in the subsequent charges, or even used altogether.

A study of the downward movement of the metal, as just stated, is important, as it suggests the way to make use, in regular casting practice, of practically every variety worthy of the name of foundry coke. Anthracite and coke, being practically identical in composition—that is, containing the same percentage of ash, fixed carbon, sulphur, etc.—differ radically only in physical structure. Thus, if the cell space of a coke is 50 per cent, as mentioned above, the coke has twice the volume of an equal weight of anthracite.

Now, not all cokes have the same cell space, and, moreover, cokes differ in percentage of ash, etc. Hence a given weight of coke varies greatly in volume, and consequently the behavior of charges of iron melted with it also varies. The foundry man when changing his brand of coke makes the mistake of believing that the new kind ought to be handled in the same way as the old. It may be possible to do so if the two cokes have the same characteristics, but if they have not the foundry man is certain to get differences in results, which he usually ascribes to sulphur, high volatile, black ends, and what not. His coke dealer then hears from him. This difficulty is so marked that even when the same brand of coke is bought the year round and trouble results, investigation of the shipments often reveals substitution of coke from different companies located in the same general region. Furnaces have this difficulty to contend with more particularly.

It is therefore highly important that the foundryman get his coke from the same ovens and the same coal mines—in fact, have it as nearly the same as it can possibly be made; if a change is desirable or necessary he should be informed, so that he can adjust his work to the changed conditions.

In this respect the retort-oven coke has a great advantage. After the coals have been contracted for and the proper mixture has been established, the result will continue to be even as long as the coking process is carried out uniformly; and as the highest scientific skill is available at retort ovens, there should be little variation from the standard if the management honestly wishes none.

Another advantage of retort-oven coke is the size and shape of the lumps. A retort oven is narrow, and the coking of a charge proceeds from the walls toward the middle; the result is a mass of coke that, when quenched, breaks into short, thick pieces. The absence of thin fingers is decidedly noticeable. In comparison, the lumps of standard grades of beehive oven coke are generally longer and thinner. Moreover, makers of retort-oven coke have carefully sorted the product before shipment, so that it has contained little breeze and the lumps have been of good shape. The size of the lumps of fuel in a cupola charge has much to do with the progress of the heat, and the absence of thin fingers accounts in a measure for the remarkable success of retort-oven coke when it was first used in foundries. Charges of coke having lumps that are uniform in shape and size give results similar to those obtained by using anthracite, which is marketed in screened sizes, because the uniform sizing of the fuel

*From Bulletin No. 3 of the U. S. Bureau of Mines, entitled "The Coke Industry of the United States as Related to the Foundry." The bulletin was written by Dr. Richard Moldenke, Secretary of the American Foundrymen's Association.

permits an even and regular penetration of the bed by the blast.

In order to adjust the process to the varying character of the coke used for melting in the cupola, it is necessary not only to change the melting conditions, for instance, by using a milder or stronger blast according as the coke is light or heavy, but also to change the relative weight of the metal and coke charges on the bed. A foundryman using heavy charges, though he may not know why, avoids coke with a large percentage of volatile matter. Such coke is necessarily underburned and light and ignites rapidly, spreading the melting through too large a volume. Very light charges will obviate much of this trouble, as the bed of coke will be kept more nearly uniform in height, and the melting will be done at the proper point. A coke in which the volatile matter is too high is virtually a gas coke and is too troublesome for use in the foundry, at least for cupola melting. It will serve well enough for all the other purposes."

Perhaps the custom that is just beginning to be introduced into this country by coke companies of sending out experts to teach foundrymen to use their particular brands of coke may obviate many of these difficulties. The retort-oven coke people were simply forced to adopt this plan, the appearance of their coke being so bad, as compared with the magnificent Connellsville product, that it took great argument to sell it at all. It should be understood that the clean, silvery appearance of beehive-oven coke as compared with the dull black cast of the retort-oven product results principally from the manner of quenching. In the beehive the quenching is done within the oven, and as little water as possible is used. In the retort oven the red-hot charge is forced out and deluged with streams of water as it drops into the receptacles for the coke. Consequently the retort-oven coke contains much more water than the beehive product. The formation of steam in the cells and the condensation and consequent drawing in of the water seem to hold it tight within, and when once water is thus included it remains. Coke exposed to rain will drain off the water to a great extent, but the occluded water of quenching stays. Sellers of retort-oven coke are therefore rather sensitive on the subject of "moisture" in the specifications, though it is hard to understand why they should expect coke prices for water. In practice, coke with a little moisture does no great harm in the cupola, although the bed coke must be dry. There is, of course, a thermal loss in driving moisture off, just as there is a great saving in the blast furnace in the use of artificially dried instead of natural air. Specifications for coke should therefore fix a maximum limit for moisture, with a penalty for any above it, to insure reasonable care in the oven practice, but this upper limit should be such that good retort-oven coke can readily keep within it.

Coke producers should therefore be urged to make thorough melting tests with their coke, so that when

selling it they can advise foundrymen how to get the best results with it. This plan would save enormous trouble, as there are about 6,500 foundries using coke in the United States and Canada, and but a very small percentage know how to make comparative tests of coke and draw conclusions therefrom that will guide them in its proper use. There are, on the other hand, comparatively few coke producers, or selling agents for coke producers, and it would pay these firms well to have a central testing station, or a sort of trouble bureau, for coke tests.

The burning of iron in the cupola process, and in fact every melting process to a greater or less extent, is now becoming better understood. In years gone by no one could be made to believe that such a thing was possible. But anyone having to do with making the iron silicates can realize how very little silica can carry great quantities of iron to make a thin black slag. The process can be watched very nicely in heating and forging shops, where billets of iron and steel are made ready for the hammers. The modern regenerative heating furnace keeps things at an extremely high temperature, and hence a billet once up to heat must be removed and worked quickly, as otherwise it will "waste away" very rapidly. In fact, if the furnace is too hot the outside of the billet will often be ready when the inside is still too cold, so that wasting from the surface takes place and cannot then be avoided.

The surface of the billet becomes oxidized or scales heavily. The scale drops off, and on touching the white-hot sand of the bottom unites with it as a silicate of iron in the form of a thin slag that runs off. Careful operators try to keep this wasting action down to a minimum, and they collect the slag in pots, selling it to blast furnaces. It contains over 60 per cent of iron and is therefore rich, though it requires much limestone for treatment if large quantities are used.

In the bottom of the open hearth furnace pools of iron often remain after the heat is drained out. These pools oxidize rapidly, burning away with a fine display of sparks, the oxidized material uniting with the sand bottom and making a temporarily dark spot. This action is observed especially in the malleable process with the open hearth furnace. In the Bessemer process the burning of iron is so obvious that one has only to stand near the blow to be spattered with pellets rich in iron. In making pig iron one has only to note that the same size of furnace that makes a given quantity of good "honest" pig iron for the foundryman is made to yield very much more pig iron when forced. Iron made thus must be oxidized slightly and produces castings weaker than they ought to be. Even in the cupola the scintillations can be seen as the drops of metal find their way to the bottom. The exact degree to which the oxidization thus brought about affects the product depends on the rapidity of the stream downward, the conditions of the blast as regards unconsumed oxygen at the point where the iron is melted, and the consequent condition of the metal itself. If the metal is oxi-

dized even slightly, its melting point is raised, and it will drop more sluggishly and receive more damage from the blast farther down. Hence the importance of nicely balancing melting conditions by proper attention to the coke and charging.

In the coke-melting tests at St. Louis another matter carefully watched was the quality of the metal cast. In every test that showed high melting loss the castings were observed to be full of pin holes near the surface, and in some tests the metal was like a big sponge. Though the silicon necessarily ran low with repeated remelting, enough remained to keep the body of the metal gray; but whenever the iron was burned the fins on all the castings were white.

Long experience in the foundry business has taught the writer that whatever cause may be put forward for such defects in castings as pin holes, excessive shrinkage, draws, and cracks, and for others due to the metal and not to the molding, practically all of them can be traced back to burned iron. Whenever steps have been taken to correct the cupola practice, these troubles have disappeared. Pin holes are due, not to the inability of the air to get out of the molds fast enough, but to the yielding up of dissolved gases in the iron at the moment of set. The first thing that happens in the mold, if the iron poured is not much overheated, is the setting of a thin skin of metal against the sand, and hence these gases in going upward and outward to escape strike the thin shell that has set and stay there. When the casting is put on the planer and the skin is removed the pin holes make their appearance and the casting is condemned. Such an outcome is particularly annoying with respect to fine varieties of rolls, on which an unblemished surface is absolutely necessary. When the metal is badly burned, as, for instance, in low-silicon heats for malleable castings, the formation of gas is so strong that the casting is honey-combed, not with pin holes as in the high-silicon iron, but with blowholes half an inch across, and the interior surface is black from oxidation.

As a further difficulty, if the metal is oxidized even very slightly its melting point is raised considerably, so that it readily freezes or, as the foundryman has it, loses its "life," and the ladles "skull." As a consequence, in pouring the gates freeze up and the castings are short poured. If the casting has been poured, the molten metal may be cut off so quickly that interior shrinkages are more than likely to occur, as no feeding can take place.

Finally, burned metal is very weak. It seems as if a coating of oxide gets between the several crystals, so that their tenacious grip on one another is lessened. Hence casting strains find the metal unprepared to resist and a crack is the result.

A little oxygen can do an immense amount of damage. The very worst burned iron ever seen by the writer contained only a few hundredths of 1 per cent. The evils of high sulphur are as nothing compared to it. Hence only a very small amount of aluminum or

titanium is necessary to make the correction, more than 0.1 per cent seldom being required for addition to the ladle to produce the desired deoxidation.

All the facts outlined above show how important the subject is and why the ferro alloys are becoming such a factor in casting practice. However, prevention of this burning or oxidation is better than its correction afterward by means of alloys of iron with titanium, aluminum, manganese, and silicon.

Unfortunately, we are still in the dark regarding the actual processes in the cupola. Until means are provided to take accurately temperatures as well as samples of the gases formed at every point within the active part of the working end of the cupola we shall have only theories. It is to be hoped that some day an experimental cupola will be erected and specially arranged for such tests, so that deductions based upon determined facts may be made. In the absence of the necessary data only conjecture is possible; nevertheless, the processes described in the following paragraphs seem actually to take place.

Inasmuch as only the top of the coke bed is effective for proper nonoxidizing melting if it was made right at first, the question arises why it is the prevailing practice to make the first lot of metal several times as heavy as those succeeding. The obvious reason is plausible enough at first glance—namely, that a lot of metal ought to go on top of a lot of coke in the bed. It is forgotten how little of that big bed of coke really does the work of melting.

The damage caused by the big first charge of iron may best be shown by figures. Let us assume that every 1,000 lbs. of iron in melting uses up one inch of the coke bed. If the first big charge of metal is 6,000 lbs., 6 ins. of the coke will have been burned away when the metal is melted. If iron is really damaged by oxidation, the last 3,000 lbs. of that charge is not quite so good as the first 3,000. Now, the first intermediate charge of coke comes down. Does that restore the 6 ins. of the bed? By no means. It is merely enough to melt the next and much smaller charge of metal. Let us say it restores the bed 3 ins. and disposes of a succeeding charge of 3,000 pounds of metal. The bed, restored only half way up to its original position in the melting zone, burns down 3 ins. again in melting the second charge, nearly all of that iron being melted at a point where it is affected by unconsumed oxygen in the blast. This restoration and melting is repeated until the end of the heat, and the practical effect of the big first charge is to give an inferior grade of iron for the entire heat, unless extra coke between the charges brings the bed up to proper level.

Many foundrymen have recognized this point, and now run their cupolas with charges of metal alike from the first to last; and the writer has persistently advocated this method as the first step in solving the problem of using all kinds of foundry coke successfully.

But if there is any advantage in keeping the fluctuations of the bed uniform, there is additional benefit in making this fluctuation as small as possible; hence the writer's second recommendation is to make the charges so small that the layers of coke are only thick enough to cover the layers of metal, unless definite knowledge regarding the range of the melting zone of a cupola permits heavier charges.

Another very important point remains for discussion. The bed must be of the right height to start the melting properly; otherwise all subsequent care is useless. Many foundrymen have adopted the method outlined above of melting by small charges, most of them successfully from the start, but others have not succeeded until attention was called to the height of their bed. Practically the only way to know when this height is right is to observe the time between "blast on" and "first iron." Every foundryman knows that when this time is about seven to ten, preferably eight, minutes, he gets his best results. So in determining the height of bed, if iron comes at a time within these limits, it is safe to continue to charge as in that heat. If the iron comes, say, in 15 minutes, the bed is too thick; if it comes in four or five minutes, the bed is too thin. Thus this matter can be attended to at first. In a new cupola the bed can purposely be made too thick in the first heat and reduced subsequently until the proper time of "first iron" is reached. This "first iron" means the first metal running continuously just previous to "stopping up," and not the few random drops that often come in five minutes. Where in practice the tap hole is closed when the blast is put on, it had better be left open, for a few days, until the action of the cupola in this respect is known.

In making the estimate of the correct time from "blast on" to "first iron," it was assumed that the charging was done on a fuel bed well burned through, with all the wood consumed. The writer has noted instances in which about a third of the bed coke was put on and burned through. Then the remaining two-thirds was added, the cupola charged, and the blast turned on. In such instances the first few minutes of the observed time to "first iron" were taken up by the fuel bed burning through and did not form any part of the melting time. Iron in eight minutes was really iron in four or five minutes, consequently the results were disappointing.

Too heavy a bed has nearly as bad an effect as one too thin. The thin bed, as has already been explained, has just the same effect as a very heavy first charge. The entire process of melting throughout the heat takes place too low down and in a more or less oxidizing atmosphere. If the intermediate charges of coke used were not as a rule slightly heavier than necessary, thus causing the fuel bed gradually to rise, a low bed together with a heavy first charge of metal would give serious trouble throughout the heat. As it is, these conditions may account for much of the badness of the "first iron," commonly laid altogether to sulphur.

On the other hand, too thick a bed unduly delays the metal. The bed has to burn away, and while it burns some of the stock melts slowly too high in the cupola and hence is exposed too long to the influence of the cupola gases. The melting is therefore slow, and the iron may be slightly burned and is apt to be cold. Slow iron nearly always means inferior iron. The evils of too heavy initial charges are perhaps most noticeable in air-furnace and open hearth practice, where a slow heat is an abomination.

If the bed is too thick, a heavy first charge of metal keeps the succeeding charges of iron from melting too high in the cupola and reduces the danger from burned iron. The writer, indeed, has observed a number of instances of excellent results obtained, with a thick bed and heavy first charge, with very small intermediate charges. Such practice does not seem to hold good, however, as some of the "first iron" will be dull. Better have a proper bed of uniform charges.

It does not follow that burned iron always results from very heavy charges. When the conditions become such that the molten iron is attacked by oxygen, the iron may be protected by the manganese in the charge. Silicon gives similar but less protection. The foundryman merely has to remember that when the metal is slightly too low in the melting zone of the cupola the same conditions obtain, though in far smaller degree, as in the Bessemer process, in which manganese, silicon and carbon, and with them more or less iron, are burned out. What the foundryman should look to is that his cupola does not become a sort of converter during any part of the melt.

In conclusion, it may be said that the logical method of cupola charging after the bed is in would be to do away altogether with separate metal and coke charges, and yet keep the proper ratio of iron to coke. This ratio can be maintained by weighing out metal and coke in separate piles on the platform and charging a small quantity at a time from each pile, after the manner of proportioning sand and facing materials. The claim has been made that this method, the one used by our foundrymen forefathers, means slower melting, but careful trial has shown that such is not the case. However, the discomfort and cost of the method will preclude its use.

The writer has advanced another suggestion, that to reduce the cost of charging a cupola to a minimum the charges should be made up in the yard. They should be carefully laid in special charging buckets with swinging bottom doors to drop the charges exactly as laid. If the cupola is cut off at the platform, one man in the yard with a reversible hoisting engine and revolving jib crane can handle the charges. This plan has been tried at a large foundry with excellent results. A suitable hood and stack above the cut-off cupola will carry off escaping gases.

The probable processes of a cupola heat have now been considered, and the lessons derivable therefrom if the assumptions are correctly made. In this description of a cupola heat, the foundryman may find

suggested a method by which he can make use of practically any kind of coke. If the coke is heavy—almost like anthracite—he can use fairly heavy charges, as the fluctuations of the melting zone will be slight. If the available coke is light, he absolutely must take small charges, the smaller the better, for large charges would result in too strong combustion and perhaps in a red-hot coke near the charging doors. The limit for lightness is set only by coke so light that combustion takes place too rapidly and the iron cannot take up the heat fast enough. This limit is quickly found in practice. The methods of making all varieties of coke, and particularly those made in retort ovens, are now fairly well settled, and hence foundry coke is made a specialty at only those plants that can obtain the right kind of coal for the purpose; that is, these coke makers have coals of various kinds available, which when ground and mixed give the desired result.

As retort-oven coke, it is to be hoped, will be the foundry fuel of the future, the coke makers, by careful study of details, will probably bring about a very uniform product, with so slight variations in composition and physical structure that a method of operation in the cupola such as that outlined above will give uniform satisfaction to the user. In the meantime, however, we have to do with existing conditions. We have coke with low and with high sulphur, coke with low and with high ash, and occasional coke so soft as to be very near the danger limit. By the very nature of the process of making it coke cannot be uniform in structure, and only careful selection from the ovens as they are discharged gives the foundryman what he is paying for. In brisk times this selection is likely to be made less conscientiously, coke forks being discarded or their prongs made closer. Many coke tests have shown conclusively that much can be done to improve a coke by adapting the process of making it to the requirements. This matter will be treated more fully in another bulletin of the Bureau of Mines.

The Chilean government has made extensive investigations for petroleum near Punta Arenas; some oil was found, and the surroundings seem to indicate presence of large quantities. Sufficient gas was found escaping in one place to burn on ignition.

A consular report states that a company of American capitalists expects to commence the exploitation of the deposits of phosphate rock and manganese on the island of Margarita and the other adjacent islands, including Orchilla and Los Roques in Venezuela. The representative of the company, who has boring machines with which to explore the phosphate deposits for the purpose of learning their extent, expects soon to begin the work of extraction and to export at least 2,000 tons of this product monthly to the United States. The output of magnesite on Margarita Island will also be greatly enlarged.

WATER-JET EXHAUSTERS.

These instruments work by a water-jet and create nearly absolute vacuum. The water pressure may be small, a head of 15 ft. is sufficient, but the capacity as regards volume of gases moved, increases at high pressure.

They are used to create vacuum in stills when dis-

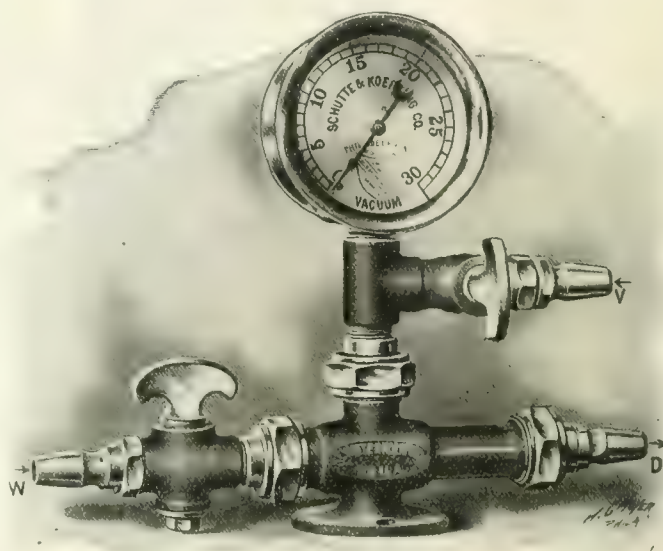


Fig. 1.

tilling under vacuum and are found superior to vacuum piston pumps where gases have to be handled which attack metals; an additional advantageous feature is the simultaneous condensation of vapors. The advantages claimed are: 1, Reliability. 2, Cheapness. 3, Simple Manipulation.

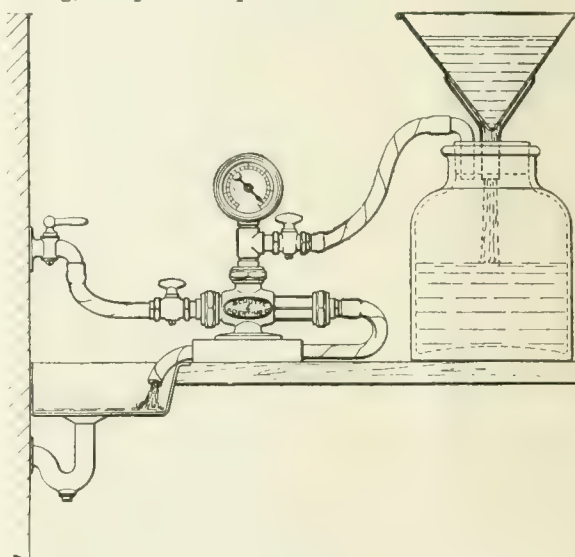


Fig. 2.

In Fig. 1, D is the discharge, V the vacuum and W the pressure water.

The water-jet vacuum pump, Fig. 2, is actuated by a water-jet which should have a pressure of 10 lbs. or more; this is obtained by connection to the city water or other convenient source, or from a tank elevated 20 ft. above the working floor. In the latter case the

water can be discharged into a tank below and pumped over again. In manufacturing processes where small volumes have to be handled, these instruments are very satisfactory; also where the creation and maintenance of a vacuum to assist filtration, percolation, evaporation, distillation and condensation, either by



Fig. 3.

removal of air or the condensation of vapors is desired.

The water-jet air compressors, Fig. 3, are used for soldering purposes, forges, laboratories and the like. The illustration shows one pattern of this apparatus made for producing small quantities of compressed air, such as are necessary for the above operations. The upper part of the reservoir C is provided with a water-jet air compressor and water enters this by the opening E. The air is sucked in through the small holes L at the side. Separation of water from the air takes place in the reservoir, the water being discharged from a syphon pipe. The air is discharged at P.

By use of this apparatus pressures of some feet water column, varying according to the water pressure available can be obtained.

For soldering work the apparatus is made to give a pressure of 4 inches of water with a water pressure of 45 lbs. (the usual city water pressure).

The water-jet exhauster shown on Fig. 4 is used extensively for the following applications: Priming centrifugal pumps; creating a vacuum on stills; assisting filtration, for creosoting tanks.

The pressure water enters at W, and air is drawn in at V. D is the discharge.

According to advance reports of the United States Geological Survey the production of talc and soapstone in the United States in 1909 was 130,338 tons. The average price of the rough was \$2.90 per ton; ground, \$7.54; sawed into slabs, \$18.67; and manufactured articles, \$22.19.

EFFICIENCY IN FUEL COMBUSTION.*

By H. M. WILSON.†

In reporting for the Geological Survey on investigations concerning the use of fuel in locomotive practice, Professor Goss stated that in the average locomotive furnace 20 per cent of the total fuel supplied to the locomotives performs no function in moving the train forward. In fact but about 40,000,000 out of 90,000,000 tons consumed in locomotives in 1906 represented heat transmitted from the furnaces to water to be vaporized. Adding to this the losses by radiation, in transmitting the steam power through the engine to the axles, adhesion to the rails, etc., it is estimated that of the latent heat value in the coal probably not over 5 per cent does useful work in moving the trains forward, the remaining 95 per cent being wasted. On the other hand, about 25 years ago the fuel waste in power production in the average stationary plant in the United States showed a loss in useful work of about 96 per cent and useful work performed of about 4 per cent. Probably the best performance at that time in stationary steaming practice was between 5 and 8 per cent. Today such power plants as those of the Chicago Edison Co., the Inter-

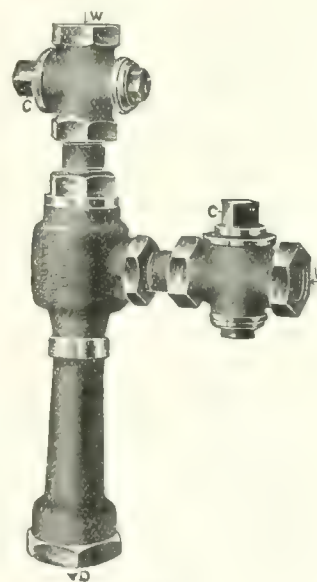


Fig. 4.

borough Rapid Transit Co. and others show useful work performed by 10 per cent of the latent fuel value, and an average for all stationary plants in the country of 6 per cent. Think what means a saving of the latent energy in coal of only 5 per cent, when only this percentage does useful work. This would mean an increase of 100 per cent in work performed. With an annual production of 500,000,000 tons this, alone, represents a saving of 250,000,000 tons to get the same amount of power. This at an average cost of \$1

*From an address delivered December 22, 1910, before the Railway Club of Pittsburgh.

†Chief Engineer, Pittsburgh Testing Station, U. S. Bureau of Mines.

per ton is equivalent to \$250,000,000 saved through more efficient conversion of coal into steam power.

A quarter of a century ago there was not a producer gas plant in operation in the United States. Today there are estimated to be 500 such plants, some of unit capacity of 5,000 or 6,000 hp. and producing, in all, nearly 120,000 hp. It is well known that producer gas can be manufactured from coals of such low grades as to be valueless for steaming purposes, thus saving that much higher grade coal for other uses. It is also well known that, on the average, the producer gas plant today converts into useful work from 10 to 16 per cent of the latent energy of the fuel, the average being over 13 per cent.

Much of the coal discarded to the culm bank in the anthracite regions has been recently proven a valuable source of fuel, not only in the gas producer, but for use in the boiler furnace by conversion into hard nodules called briquettes. By briquetting the waste anthracite culm, and the slack left at bituminous mines, a further source of fuel saving has been developed. So, also, within half a generation, further enormous waste through the fuel gases of the blast furnace, the coking oven, and similar appliances have been proven unnecessary and avoidable.

Finally, whereas until recently it was customary in mining to leave an average of 50 per cent of the available coal in the ground, a loss to all future time, and to discard all other valuable coal prior to the more general introduction of washing and sorting methods as now practiced, it is conceivable that with better practice as evidenced in Europe, and with more favorable commercial and transportation conditions these mines wastes may be reduced to 25 per cent or less. This would represent a saving to the future of one-quarter of the present annual production of 500,000,000 tons.

For the above reasons the Congress of the United States, and through it the Secretary of the Interior, have directed that the recently created Bureau of Mines shall devote its first energies to an investigation of the problems connected with the mining, preparation and marketing of coal, and its conversion to commercial use. With the results of the preliminary investigations before them, and with the removal of the uncertainty surrounding the work of the Geological Survey, due to the hand-to-mouth method of making annual appropriations, it is now possible for the officers of the Bureau of Mines to plan broadly and deeply for the future.

This investigative fuel testing work by the federal government is not to be confused with that variety of testing conducted by commercial institutions, railways, and power plants. These have as their primary object the testing or development of the machine. The investigations of the federal government have as their primary object the better utilization of the fuel resources of the country. The Bureau of Mines tests are, therefore, not competitive or investigative tests of the various makes of power producer, but they have

in view the ascertainment of the best existing type of commercial appliances in which to burn with highest efficiency each separate variety of fuel. They are also planned with a view to learning something of the fundamental principles underlying fuel combustion in various appliances, in the hope that thereby inventors and manufacturers may develop improvements in them.

The Bureau of Mines has another function to perform in connection with its fuel tests which has led to much misunderstanding of its purposes, and some unwarranted criticism. Used as a testing laboratory by the several hundred purchasing officers charged with the procurement of fuel for the power plants in hundreds of public buildings scattered from Maine to California, in the power plants of the Isthmian Canal Commission, the steamships of the Panama Railway, and the heating plants of the various military posts and Indian agencies, the Bureau of Mines is charged with the duty of protecting the government to the extent of ascertaining that the quality of coal furnished by contractors is the highest procurable under the specifications. Of over 400,000 tons of coal shipped annually to the Isthmus of Panama, and 100,000 tons consumed in Panama railway steamships, specifications calling for an unusually high grade fuel have been adopted by the government purchasing officers, as would be done by those of any commercial institution which aimed at economic business management. It would be unprofitable to transport from the mines to the isthmus the vast tonnage represented in an unnecessarily high ash or moisture content. On the other hand, the Bureau of Mines has been instrumental in the cases of many public buildings throughout the United States, and in the 40 odd buildings in Washington, in showing how, by comparatively minor changes in existing equipment or replacement thereof, or different methods of manipulation therein, it is possible to burn advantageously lower grade coal, bituminous coal or buckwheat anthracite, in place of high grade expensive steaming coal or furnace anthracite previously in use. In one public building in Washington, alone, it is estimated that a saving of \$15,000 was made by change of furnace equipment, whereby buckwheat was substituted for furnace anthracite. In buildings there and elsewhere the use of bituminous coal has recently been introduced through the agency of this bureau, thus encouraging, so far as possible, the use by the government of the cheaper low grade and comparatively waste fuels in place of the more expensive and scarce high-grade fuels.

It is well known, perhaps, to many of you that when coal is contracted for and sold under some trade name such method of sales gives little opportunity for the purchaser to determine an inferior delivery in case the vendor is short of the coal contracted for. Moreover, the purchaser would have little recourse in the courts, especially if much of the coal has been consumed. The Bureau of Mines, and its predecessor, the

Geological Survey, have consistently advocated the purchase of coal on a specification which fixed the heat value of the fuel offered. Such an arrangement is eminently fair both to seller and purchaser; the chemical analysis determines the number of heat units, or British thermal units, in the average sample, and accordingly as the delivery is above or below, premiums or deductions are made in settlement. No question need be raised as to the rejection of the delivery for the purchaser will pay a less price for an inferior quality. In other words, he will get what he pays for and the vendor will be paid for what he sells.

Today the United States government, which, through its many agents, is the largest single purchaser of coal, buying annually an amount estimated considerably in excess of 6,000,000 tons, is procuring 3,000,000 tons of this on specifications fixing the number of British thermal units. The remainder of its fuel purchases, chiefly those for the navy, have been practically reduced to an equivalent basis through the tests and mine data furnished the navy purchasing officers. Nevertheless, this feature of the work of the Bureau of Mines which, as previously stated, has brought some criticism of its apparent encouragement of the use of only high-grade coal, is the least of that bureau's operations in connection with fuel investigations.

Many hundred steaming investigations have been made by the Bureau of Mines. These tests have been made on horizontal return tubular and on water tube boilers of standard make, used in the daily power production at the Testing Station here in Pittsburgh. They have covered such a wide range of fuels that they may be characterized as a combustion survey of the coal of the United States. These coals are burned in furnaces in which the baffling is changed from time to time. They are hand-fired or mechanically stoked, with under-feed, over-feed or chain grate stokers. The draft may be natural, forced or induced, or a combination of the latter. It has thus been possible to determine in a preliminary way what type of furnaces, how baffled, how fired, how stoked, and with what draft, each coal should be burned in order that it may produce the highest efficiency, and emit the least smoke. Similar investigations on some hundred coal purchases in different parts of the United States for use at army posts have been conducted on three types and sizes of heating boilers; a small one for heating a residence building or officers' quarters, a larger one for heating a medium sized hospital or medical hall, and a good-sized boiler for heating a large barrack building. In the latter connection a series of tests is now in progress to ascertain the most efficient and most smokeless method of using the different coals in a horizontal return tubular hand-fired boiler of medium capacity—50 horsepower.

In each test a mining engineer of the bureau samples the coal as mined and as loaded in the cars; samples are taken when unloaded at the testing station and as fired. A bulletin is in course of preparation

setting forth the result of the proximate analyses, calorimetric determinations, and ultimate analyses of nearly 5,000 mine samples, of which the origin and history is authenticated. With this information it is possible for the government representatives and it should be for those of railway and commercial institutions, to predict the character of the coal which can be furnished them from any mine in the United States. Through the medium of the ultimate analyses it has been possible to detect that coal specified by the name of a colliery or coal field, for instance as Pocahontas No. 3, did not come from that seam, but from some other seam carrying a less high-grade coal, though drawn from the same shaft and through the same tippie.

In the course of these tests feed water is weighed and analyzed, clinker and ash are weighed and analyzed, and fuel analyses and temperature measurements in the furnaces and in the flues are taken. These data furnish an invaluable index to the best method of utilizing each coal and for pointing the way for future investigations along these lines.

The gas producer investigations, the briquetting and coking investigations have been accompanied by similar care in fuel sampling, and in chemical and physical analyses and measurements.

With fixed appropriations and the permanency of the work better assured, the Bureau of Mines has, since its organization in July last procured the equipment and commenced investigations of a more fundamental nature based upon these preliminary tests. An exaggerated furnace, named a long combustion chamber, has been perfected and put in operation. This is nearly 45 ft. in length from the fire door to the boiler tubes and approximately 3 ft. in rectangular inside cross-section. The inner lining is of fire-brick, separated by an insulated space from the outer portion of common brick. At every 5 ft. of its length are peep holes and work holes where the appearance of the flames may be observed, and the temperatures measured and gas samples taken from any part of the cross-section. In all, 36 simultaneous gas samples may be taken in different parts of the furnace.

In this appliance coals will be burned under fixed conditions of firing and draft, and it is believed that many of the fundamentals of fuel combustion will here be better developed. It should be possible to determine just what length of furnace is necessary to procure the most efficient mixture and highest temperature in the fuel gases from any coal, and that necessary for the emission of the least smoke. These first investigations will be followed by others in which different arrangements of baffling or different cross-sections will be given the combustion chamber. The present cross-section is not unlike that of a locomotive, and the results should, therefore, be of value to those interested in locomotive practice.

Having investigated producer gas generators of commercial types, the Bureau of Mines has just completed the installation of an experimental producer of

moderate size which may be operated either as apressure or as a suction producer. It is so arranged that gas samples and temperatures can be taken at various points in the fuel bed, the formation of clinker be studied, and the problems concerning the efficiency of the fire-brick lining, and the several processes which take place within the producer can be thoroughly investigated. The preliminary tests in this apparatus will have to do with the effect of depth of fuel bed, different ratios of air to steam, and with different rates of combustion with various types of fuels.

In the producer gas tests heretofore run no difficulty has been experienced in making runs of a week's duration on any fuel, including the most volatile bituminous coal, lignite and peat. Comparisons of the several hundred tests run on coals from different parts of the country in the steam boiler and the gas producer, are most interesting. The gas producer operated in the best way with West Virginia coal produced 3.3 times as much efficiency as did the steam boiler of the best type operated on the same fuel in the best way in test practice. The lowest grade of lignite developed, in the gas producer, about the same efficiency as did the best steaming coal under the boiler.

The preliminary briquetting tests have proven the value of this manufactured product as a substitute for run of mine coal under certain conditions. The coal is ground and mixed with a small percentage of binder, usually five or six per cent of pitch. It is then heated and briquetted under a pressure of one or two thousand pounds per square inch. These coal briquettes burn from the outside, giving off the volatile matter slowly and coking as the combustion proceeds until the entire briquette is consumed. They burn much as does furnace anthracite. A number of carloads of lignite each from Texas, Colorado and Dakota have been successfully briquetted at the Pittsburgh Testing Station of the Bureau of Mines. For this purpose the lignite is disintegrated, then dried to free it of excessive moisture, and is then briquetted under great pressure, about 20,000 lbs. per square inch, the resulting briquettes being as firm and hard as anthracite. They are especially well adapted for combustion in locomotive or in house heating furnaces and ranges. This fuel, like slack coal, may be had at the mines at about \$1 per ton; adding \$1.50 for manufacture and binder, a product is obtained at \$2.50 per ton at the mine, which compares favorably with high-grade bituminous or anthracite coal. Since the government and the railways own some million of acres of lignite in the West, this will undoubtedly be an economical source of coal supply when its manufacture on a commercial scale is perfected. Many of you will recall the conditions under which, a few years ago, a coal famine occurred in mid-winter in North Dakota. This was due, in large measure, to the reliance of that region on the slack coals of neighboring states, which could not be carried long in storage and called for daily deliveries. Heavy snow

drifts impeded transportation and the resulting suffering ensued. Briquetted lignite from local sources can be carried in storage throughout the winter months.

The damage which smoke inflicts each year in the United States has been estimated to amount to several hundred million dollars annually, the destruction of merchandise, the defacement of buildings, the injury to animal and plant life, and the greatly increased labor and cost of housekeeping. The Bureau of Mines has given much attention to this question through the medium of its combustion tests of coal. The tests made indicate clearly that the production of smoke in stationary power plants is unnecessary when proper equipment and operation are provided; that, though bituminous coal will produce more or less smoke in many types of furnace, that practically every coal can be burned smokelessly and, therefore, with increased efficiency in some existing type of furnace.

Smoke reduction is largely a question of recognizing the fact that there are very many kinds of coal and that each requires a different combustion chamber, rate of draft, grate area, and method of stoking, to produce from it the most efficient and smokeless operation. For railroads the problem of smoke abatement is a much more difficult and complicated one than for stationary power plants, as it is also in certain furnaces and smelting operations. Fortunately we are coming to realize that the emission of smoke is not the badge of prosperity it has in the past been thought, but is, on the contrary, a sign of inefficient or ill-directed management. In the practical solution of the smoke problem a fatal error to be avoided is the enactment of a stringent ordinance. Such measures elicit only antagonism. The real campaign must be one of education.

The following information concerning a wild cocoon silk reeling mill, established by the South Manchuria Railway at Fushimidai, Japan, is from a local newspaper: "Everything at the Fushimidai silk reeling mill is, to a certain degree, in an experimental stage, and it will take eight months more to complete its first working year before its status can be assured. The great difficulty has been the training of the unskilled hands, and for their encouragement a group has been receiving wages for actual work done. Should this prove satisfactory, the system will be applied to the whole force. The present wage scale for the 140 working girls, of whom 80 live in the quarters provided by the company and 60 with their friends in the town, begins with 20 sen (10 cents) for apprentices and goes up to 50 sen (25 cents) for the best workers per day. The mill will be placed on a self-supporting basis when, with 200 reeling hands, it can turn out 100 kin (133 pounds) of wild cocoon silk per day, which is considered feasible, and quite within their reach when the hands have acquired sufficient skill."

THE HEAT-TREATMENT OF BRASS.*

By G. D. BENGOUGH, M. A., AND O. F. HUDSON, M. Sc.

OBJECTS.

The objects of the present investigation were:

1. To study the effects of heat-treatment on rolled and drawn 70:30 brass of varying size of section and of somewhat varying character, as made by different manufacturers.
2. To throw some light on the question of "burning." For the purposes of the present paper a sample of brass is considered to be burnt, when its ductility, as measured by the percentage elongation, has been lowered by heat-treatment.

HISTORICAL.

The most important researches on similar subjects have been carried out by Charpy, Grard, and Cubillo. Charpy's work,† which consisted of an investigation into the constitution, structure, and properties of alloys of copper and zinc, included a number of experiments on the effect of annealing on 70:30 brass. His results are given in Table I, and may be compared with those obtained by the authors. His conclusions on the question of burning were as follows: (1) 70:30 brass made from pure copper and pure zinc was not burnt after annealing at 900° C. (2) Ordinary 70:30 brass (0.15 per cent tin and 0.2 per cent lead) is burnt by heating to 800° C. His results, however, were not very definite, and he only examined the question from the point of view of temperature. The authors, on the other hand, have endeavored to ascertain by what means the temperature has effected the deterioration in properties—whether by volatilization of zinc, by excessive crystalline growth, or by mere loss of cohesion between the molecules at high temperatures, in which case permanent deterioration may result from careless handling. The influence of time has also been taken into account.

Grard's‡ experiments were mainly directed to ascertaining in a quantitative manner the effect of work on α -brass, and the exact temperature at which that effect was removed. With a constant annealing time of fifty minutes (using testing bars of 52 sq. mm. and 60 sq. mm. in cross section) the elastic limit was found to fall off very slightly up to 200° C., then more rapidly to 300° C., and beyond that temperature more slowly to 825° C., when it was very small. The breaking load was practically unaffected below 275° C., but between 275° C. and 300° C. it fell off rapidly, and then more slowly to 900° C. No effect was produced on the ductility as measured by elongation by annealing temperatures up to 250° C. From 250° C. to 300° C. there was a rapid increase in the elongation, and from that temperature a regular increase up to 285°

C., after which the elongation decreased as the annealing temperature became higher. Grard also investigated the effect of varying time of annealing for the lower temperatures, and found that at 300° C. one hour was required to produce any noticeable change in the strength or ductility, while at 400° C. annealing for ten minutes was sufficient to produce a marked effect. For temperatures above about 500° C. to about 650° C. annealing for more than one hour had little further effect on the strength or ductility. Grard also examined the microstructures of the annealed bars, but his work throws little new light on the question of burning.

Cubillo§ sets forth his experience as a manufacturer of brass for cartridge and other purposes, but gave no details of experimental results. His paper deals with effect rather than cause.

TABLE I (A).—70:30 BRASS.*

Annealing Temperature, Degs. Cent.	Maximum Stress Tons per Sq. In.
0	31.4
200	32.5
280	29.5
420	21.6
500	21.6
560	19.0
600	17.4
650	17.4
730	18.6
780	18.2
800	18.2
850	18.2

*Charpy's Results.

TABLE I (B).—70:30 BRASS, WITH 0.15 PER CENT TIN AND 0.2 PER CENT LEAD.*

Annealing Temperature, Degs. Cent.	Maximum Stress, Tons per Sq. In.	Elongation, Per Cent.
0	39.4	3.8
540	20.3	55.0
585	20.1	57.3
620	19.0	60.8
700	18.6	64.8
730	19.0	62.0
860	17.5	57.0
930	16.8	56.5

*Charpy's Results.

MATERIAL USED IN THE PRESENT RESEARCH.

A $\frac{3}{4}$ -in. round bar of ordinary high-grade 70:30 brass was supplied in the hard-rolled condition by a well-known firm. This bar was cut into test pieces, lettered G, and may be taken as representing the ordinary 70:30 brass of commerce. Its analysis was as follows:

	Per Cent.
Copper	71.0
Tin	0.02
Lead	0.25
Iron	0.2

To compare this, another bar of $\frac{3}{4}$ -in. round hard-rolled 70:30 brass was supplied by another firm. This bar was cut into test-pieces, lettered HB. Its analysis was as follows:

	Per Cent.
Copper	69.4
Tin	Trace
Lead	0.005
Iron	0.15

It will be noticed that this alloy is of unusual purity. The same firm also supplied to the authors a third bar,

*From a paper read at Glasgow meeting, Institute of Metals, Sept. 21-23, 1910, and reprinted in *The Metal Industry*.

†Bulletin de la Société d'Encouragement l'Etude des Alliages, p. 1.

‡Revue de Métallurgie, 1909; Mémoires, p. 1069.

§Engineering, October 27, 1905.

B, of similar material, having the following analysis:

	Per Cent.
Copper	69.7
Tin	trace
Lead	trace
Iron	0.2

This bar was 1-in. round, and was in a softer state than the others. It was mainly used for some experiments on the effect of prolonged annealing at high and other temperatures.

All the material was such as is supplied by the manufacturers in the ordinary way of business to their customers.

SCHEME OF WORK.

The bars as received from the makers were cut to 9-in. lengths, turned into test-bars, and heated in electrically-heated tube furnaces, the temperatures of which were ascertained and controlled by thermoelectric pyrometers. After annealing, the bars were withdrawn at a dull red heat, cooled in air, and tested in the ordinary way. Before testing, a piece

TABLE II.—MECHANICAL TESTS OF BARS HB.

Mark.	Treatment, Time, and Temperature of Annealing.	Yield Point, Tons per Sq. In.	Maximum Stress, Tons per Sq. In.	Elongation, Per Cent. on 2 In.	Contraction of Area, Per Cent.	Surface of Bar After Test.
HB	As rolled	31.5	31.7	19.6	65.0	Smooth
HB14	½ hour, 320° C.	27.4	32.9	19.5	65.5	"
HB10	½ hour, 405° C.	11.3	25.1	55.0	75.5	"
HB1	½ hour, 500° C.	12.4	23.2	41.0	72.0	"
HB2	½ hour, 570° C.	9.6	23.4	60.0	77.0	"
HB3	½ hour, 630° C.	6.5	21.0	65.0	75.0	"
HB12	½ hour, 657° C.	5.3	20.95	68.0	81.0	"
HB4	½ hour, 660° C.	6.4	20.5	76.5	80.0	"
HB5	½ hour, 680° C.	5.4	20.2	70.0	77.0	{ Very slightly roughened.
HB6	½ hour, 702° C.	5.1	20.0	76.0	77.0	"
HB11	½ hour, 754° C.	4.8	20.3	72.0	80.0	{ Slightly roughened.
HB7	½ hour, 765° C.	4.1	19.8	79.0	70.0	"
HB8	½ hour, 808° C.	3.4	18.4	81.0	75.0	Roughened.
HB13	½ hour, 847° C.	4.0	18.8	77.0	73.0	"
HB9	½ hour, 900° C.	3.3	17.7	83.5	70.0	"

†Broke outside gauge marks.

was cut from each test-bar for microscopic examination. With a view to ascertaining the influence of size of section, a hard-drawn ⅛-in. diameter wire of brass similar to bar G was annealed at the same time as the test-bars. Test-pieces of this wire were also heated in an electric resistance furnace to temperatures between 300° C. and 600° C., and tested while at these temperatures. Various subsidiary sets of experiments were also made, particularly on the question of "burnt brass."

EFFECT OF ANNEALING TEMPERATURE.

The results of the ordinary tensile tests on bars and wires annealed for half-an-hour are given in Tables II to IV. From these results the following facts are evident:

MAXIMUM STRESS.

1. In all cases the maximum stress increases with rise in annealing temperature up to about 300° C. It then falls off rapidly to 500° C. The HB bar evidently had received more work than the G bar or the wire.

and, in consequence, there is much greater numerical loss of strength between these temperatures. Nevertheless, for temperatures between 500° C. and 800° C., the strength of the HB test-bars is in all cases superior to that of the G bars, though the superiority gradually diminishes as the temperature rises, and has disappeared at 850° C. As regards the wire, the strength falls off uniformly from 500° C. to 900° C., and is, for all temperatures, less than that even of the G bars, the difference increasing as the temperature rises. It will be seen later that prolonging the time of annealing has the effect of lowering the strength of a bar; it would also seem from the above that decreasing the section has a similar effect. The

TABLE III.—MECHANICAL TESTS, BARS G, HALF-HOUR ANNEALING.

Treatment, Time, and Temperature of Annealing.	Yield Point, Tons per Sq. In.	Maximum Stress, Tons per Sq. In.	Elongation, Pct. on 2 In.	Contraction of Area Per Cent.	Surface of Bar After Tests.
G As rolled.	21.5	25.0	35.0	65.0	Smooth.
G15 ½ hour, 285° C.	20.9	25.6	35.0	66.0	"
G16 ½ hour, 350° C.	18.4	25.2	38.5	53.0	{ Slightly roughened.
G17 ½ hour, 400° C.	16.3	24.6	40.0	53.5	Smooth.
G18 ½ hour, 500° C.	5.8	20.9	67.5	74.5	{ Slightly roughened.
G21 ½ hour, 725° C.	4.3	18.5	80.0	75.0	Roughened.
G26 ½ hour, 920°-925° C.	3.7	18.2	78.0	63.0	Cracked.

TABLE IV.—MECHANICAL TESTS OF WIRES.

Mark.	Treatment, Time, and Temperature of Annealing.	Yield Point, Tons per Sq. in.	Maximum Stress, Tons per Sq. in.	Elongation, per Cent. on 2 In.
W0	As drawn	23.1	25.3	15.5
W0	As drawn	22.5	25.0	19.0
W15	½ hour, 285° C.	22.5	28.5	20.0
W14	½ hour, 320° C.	19.9	25.3	25.0
W16	½ hour, 350° C.	18.2	26.2	30.5
W10	½ hour, 405° C.	8.5	20.7	45.0
W8	½ hour, 500° C.	5.2	18.1	54.0
W9	½ hour, 570° C.	6.6	18.4	55.0
W4	½ hour, 630° C.	4.3	17.1	58.0
W12	½ hour, 657° C.	2.8	16.4	55.0
W1	½ hour, 660° C.	3.8	16.7	*53.0
W7	½ hour, 680° C.	3.2	16.5	58.00
W3	½ hour, 702° C.	3.1	16.4	63.0
W11	½ hour, 754° C.	3.8	15.8	63.0
W2	½ hour, 765° C.	1.9	14.7	61.0
W6	½ hour, 808° C.	3.1	15.3	59.0
W13	½ hour, 847° C.	3.8	15.5	59.0
W5	½ hour, 900° C.	4.1	14.3	53.0
W26	½ hour, 925° C.	...	12.9	25.0
W25	2 hours, 910° C.	...	10.5	18.0

*This bar showed signs of melting.

corollary at first sight would seem to be to proportion the time of annealing to the section of the bar, and so to avoid burning thick sections by annealing at higher temperatures. It will be seen later, however, that time is at least as important a factor in burning as temperature. It does not seem, therefore, that any decided advantage can be claimed for the method suggested over the rival one of proportioning the temperature to the section. As a matter of fact, both plans are used in works, some managers favoring one and some the other.

YIELD POINT.

2. The yield points given in these tables are the stresses required to give the first noticeable elongation, but, except in the case of the bars as hard-rolled,

and in a few other cases, the yield point is not very sharply defined.

ELASTIC LIMIT.

3. It was thought that it would be interesting to make a determination of the true elastic limit in the case of one of the alloys used, and to ascertain whether any definite load existed which could be strictly regarded as a sharply defined yield point; also, if it existed, what relation this had to the elastic limit. This determination was made on the B alloy, using a bar with screwed ends, 16 ins. long and 12 ins. between shoulders. The elastic limit is at 8.35 tons per square inch. The yield point cannot be strictly defined, but the extension which occurs at 12.5 tons would probably be the first which would be easily recognized in ordinary testing. It was thought that it would be interesting to make a few experiments as to the effects of a period of rest, and of very low annealing temperatures on the elastic limit. The results obtained are shown in Table V. It will be seen (1) that a prolonged period

TABLE V.—DETERMINATION OF ELASTIC LIMIT, BAR B. DIAMETER OF TEST PIECE, 0.781 INCH.

No. of Loading.	Treatment.	Elastic Limit, Yield Point,	
		Tons per Sq. In.	Tons per Sq. In.
1	As rolled.....	8.35	12.5
2		8.35	14.0
3	12 days after No. 2.....	8.35	15.2
4	1 hour, at 90° C.....	8.35	15.4
5	1 hour, at 210° C.....	5.0	7.0

of rest raises the yield point; (2) that annealing for 1 hour at 90° C. has no effect whatever; and (3) that annealing for 1 hour at 210° C. has very considerably lowered both elastic limit and yield point.

ELONGATION.

4. In all cases the elongation is not much affected until an annealing temperature of 300° C. is passed. Just above this temperature a very rapid increase occurs in the case of the wires. In the case of the bars the rapid increase sets in at about 350-400° C. Here again an increase in size of section seems to require a longer time of annealing to produce similar results. The maximum elongation is reached with an annealing temperature of 750° C. for the wires, and with approximately the same temperature for the bars G. In both these cases a diminution seems to occur with annealing temperatures above 800° C. For the HB bars, on the other hand, the elongation appears to increase regularly up to a temperature of 900° C.

INFLUENCE OF TIME IN ANNEALING.

The above remarks apply only to bars annealed for half an hour. The tests given in Table IX were made for the purpose of comparison with the tests referred to.

In the case of bars B, i. e., of the brass free from tin and lead, annealing for one week at 440° or seven hours at 665° C. has practically no effect. Annealing for twenty-four hours at 750° C. causes a lowering of the maximum stress, but the elongation of 81 per cent on 2 ins. indicates that the alloy is not injured by this treatment. Similarly it will be noticed that annealing for twenty-four hours at 810° C., although causing

a still further lowering of the maximum stress, has not produced any effect that indicates "burning." Bar B9, which was annealed for only six hours at 840-850° C., shows a marked lowering of the elongation and contraction of area, and has suffered by this treatment a distinct deterioration. Lastly, an annealing for seven hours at 900-910° C. has resulted in a further very noticeable weakening of the material, accompanied by a pronounced loss of ductility. A comparison of the mechanical properties of bars B17 and B9 with those of

TABLE VI.—MECHANICAL TESTS, G BARS ANNEALED IN REDUCING TEMPERATURES.

Mark.	Treatment, Time, and Temperature of Annealing	Yield Point, Tons per Sq. In.	Maximum Stress, Tons per Sq. In.	Elongation, Per cent on 2 In.	Contraction of Area, Per Cent.	Surface of Bar After Tests
G0	3 hours, 700° C.	5.9	21.5	68	66	Smooth.
G1	3 hours, 825° C.	3.7	18.6	80	70	} Slightly roughened.
G4	7 hours, 850° C.	..	17.1	62	..	
G3	3 hours, 900° C.	3.5	16.8	58	..	} Badly cracked
G2*	2½ hours, 900-930° C.	3.7	15.3	41	32.7	

bars HB9 and HB13, of similar material, annealed for only half an hour at 900° C. points very clearly to the important effect of time in annealing at very high temperatures. It need hardly, however, be said that such very drastic treatment as bar B17 has received is very unlikely to occur in practice. These experiments, therefore, cannot be claimed to have ascertained the cause of the phenomenon of burning, which is said to occur occasionally even in the best regulated works.

COMPARATIVE EFFECTS OF OXIDIZING AND REDUCING GASES IN THE ANNEALING FURNACE.

It was thought possible that the presence of reducing gases in the annealing furnace might, at unduly

TABLE VII.—MECHANICAL TESTS, BARS G ANNEALED AT HIGH TEMPERATURES IN AN OXIDIZING ATMOSPHERE.

Mark.	Treatment, Time, and Temperature of Annealing	Yield Point, Tons per Sq. In.	Maximum Stress, Tons per Sq. In.	Elongation, Per cent on 2 In.	Contraction of Area, Per Cent.	Surface of Bar After Tests
G25	2 hours, 900-925° C.	4.5	18.0	67.0	70.6	} Roughened. Cracked.
G27	1 hour, 910° C.	3.6	17.25	71.0	51.3	
G28	3 hours, 910°-920° C.	3.4	16.4	62.5	49.7	} Badly cracked. Cracked.
G29	3 hours, 910°-920° C.	3.0	16.9	64.5	47.0	

high temperatures, cause a reduction of small amounts of oxide or slag contained in the metal, and thus affect its mechanical properties, especially in the case of the G bars, which contained a certain amount of impurities. Accordingly bars G0, G1, G2, G3, G4 were annealed in an atmosphere of coal gas. The results are given in Table VI. It will be seen that annealing for three hours at 825° C. has in no way injured the alloy, but that seven hours at 850° C. has produced a distinct deterioration, as shown by the elongation. Three hours at 900° C. has also injured the alloy, and two and one-half hours at 920-930° has altogether ruined it. Comparing these with the similar tests (Table VII) carried out in an oxidizing atmosphere, it seems probable that

the presence of reducing gases is at any rate not a predominant factor in the production of burnt brass, since annealing in an oxidizing atmosphere produces very similar effects.

The authors wish to express their acknowledgments to Mr. F. H. Hummel, M. Sc., for his assistance and interest in many of the tensile tests detailed above.

LOSS OF ZINC BY VOLATILIZATION.

It has been suggested* that loss of zinc by volatilization may be one cause of the phenomenon of burning. The following experiments were therefore made to test this: Turnings were taken at various depths from the exterior of certain of the bars after they had been annealed at the higher temperatures, and estimations of the percentage of copper made on each sample taken. The results are recorded in Table VIII. It will be seen that the exterior layers, to a depth of 1-64 in., showed an increase in the copper (i. e., a loss of zinc) of 1.5 per cent after being annealed for half an hour at 850-900° C. The loss of zinc decreases as the distance from the surface increases, and at a depth of about 1-16 in. the loss of zinc by volatilization is almost

TABLE VIII.—LOSS OF ZINC BY VOLATILIZATION.

Bar.	Treatment.	Distance from Surface.	Percentage of Copper.
G2	Annealed for 2½ hours at 920°-930° C. in a reducing atmosphere	0-1/64 in.	76.7
		1/64-1/32 in.	75.3
		1/32-3/64 in.	72.3
		3/64-1/16 in.	71.5
		1/16-5/64 in.	71.2
G27a	Annealed for 1 hour at 930° C. in an oxidizing atmosphere	0-1/64 in.	75.7
		1/64-1/32 in.	74.4
		1/16-3/32 in.	70.6
HB9	Annealed for ½ hour at 900° C.	0-1/64 in.	70.8
		1/64-1/32 in.	70.0
		1/32-3/32 in.	69.7
		3/32-1/64 in.	69.4
HB6	Annealed for ½ hour at 700° C.	0-1/64 in.	69.9
		1/64-1/32 in.	69.7
		1/32-3/64 in.	69.8
		3/64-1/64 in.	69.5

negligible. For a bar, such as HB6, which was annealed for half an hour at 700° C., the loss is small, and amounts to only 0.5 per cent on the exterior layer of 1-64 in. For a longer annealing time, and for bars that were known to be distinctly burnt, the loss of zinc was considerably higher, amounting in one case (G2) to 5.5 per cent. It will be seen that this loss is approximately five times that lost by HB9, which was annealed at about the same temperature for half an hour. But even in this case the loss of zinc by volatilization almost ceases to be appreciable at a very short distance from the surface of the bar, and consequently the ratio of zinc lost to total zinc originally present must be small. Moreover, the zinc does not appear to be removed by violent evolution, but by gentle diffusion and evaporation from the surface. It seems to the authors probable that the removal of zinc is not a primary cause of burning, but merely an accompaniment of that phenomenon. To a very limited extent it occurs also at ordinary annealing temperatures.

*C. H. Desch, *Metallography*, p. 387.

TESTS AT HIGH TEMPERATURES.

It is thought that it would be interesting to carry out a series of tests on one of the alloys at various temperatures, so as to afford a comparison with those of the similar alloys which have been merely annealed at a high temperature and afterwards tested cold.

The method used for carrying out these tests was as follows: A length of wire was clamped at the top end and heated in a vertical electric resistance furnace. From the bottom end of the wire a stirrup was hung on which could be placed a series of standard weights. It was found that the speed of loading had a considerable effect on the maximum stress, especially at the higher temperatures. The figures for this property correspond to the load required to break the wire in from 15 to 20 minutes. It was found that any attempt to mark the test-wires with punch marks, or fine scratches made with a steel or diamond point, localized the fracture, particularly at temperatures above 400°. To get over this difficulty fine graphite marks were made on the wires at distances apart of 1 in.; these were used for determining the elongation in the ordinary way. This plan was found to be satisfactory up to a temperature of 500° C. At temperatures in the neighborhood of 600° C. some difficulty was experienced in recognizing these marks owing to the large amount of oxidation of the alloy. The figures for the wires tested at 590° C. may however, be said to be reliable, since four results agreeing fairly closely have been obtained. No temperatures higher than 600° C. have so far been used in these tests, owing to the difficulty mentioned. The results are given in Table IX.

TABLE IX. MECHANICAL TESTS—BARS B.

Mark.	Treatment, Time and Temperature of Annealing.	Maximum Stress, Tons Per Sq. In.	Elongation, Per Cent on 2 in.	Contraction of Area, Per Cent.	Surface of Bar After Test.
B5	As rolled	22.4	65.0	78	Smooth.
B6	"	22.25	57.0	76	"
B3	1 week, 444° C.	22.2	62.5	70	"
B4	1 week, 440° C.	22.6	70.3	72	"
B1	7 hours, 665° C.*	21.85	70.5	77	Smooth.
B2	7 hours, 665° C.†	22.25	69.5	76	"
B7	24 hours, 750° C.	19.6	81.0	..	{ Slightly roughened.
B8	24 hours, 750° C.	19.5	81.0	..	"
B12	½ hour, 860° C.	17.7	87.5	..	Roughened.
B16	24 hours, 810° C.	17.3	18.5	75	"
B9	6 hours, 840°-850° C.	17.3	70.5	68	Cracked.
B17	7 hours, 900°-910° C.	15.6	45.5	54	{ Badly cracked

*Yield point 7 tons per square inch.

†Broke outside gauge marks.

MICROSCOPIC WORK.

The microstructure of the bars after annealing and before testing was examined, and typical structures are illustrated by photographs reproduced in Plates I to III. The general effect of annealing on the crystalline structure of 70:30 brass is now a matter of common knowledge, but there are one or two points that may perhaps be noted here with advantage.

1. Annealed Structures.—The first effect of annealing on a rolled or otherwise strained sample of brass is to bring any altered and crushed material into a condition suitable for subsequent recrystallization. As the temperature rises, recrystallization sets in from a large number of new centers, the result being that after annealing for half an hour at 500° C. the structure appears very distinctly finer than in the original bar as rolled. From this point crystal growth proceeds rapidly.

2. Over-Annealed Structures.—The result of annealing at an unduly high temperature is a very rapid crystalline growth, which causes the coarse structure characteristic of over-annealed brass. After a certain time this very rapid growth of crystals probably slows down, and size of crystals approaches a maximum value.* Possibly there is a definite size of crystal characteristic of each temperature, which is not affected by time once equilibrium has been reached.

3. The Microstructure of Burnt Brass.—As far as the authors have been able to ascertain, the structure of burnt brass, in the case of the material used for this work, closely resembles over-annealed brass, with a possible difference that the lead segregates in the crystal boundaries. They have found no sign of the pitted structure, which is generally said to be characteristic of burnt brass, except with very deep etching.

4. Effect of Lead.—The presence of lead is indicated by small, more or less rounded, particles scattered through the crystals. When the brass containing lead has been annealed at a very high temperature, there is a tendency for the lead to segregate between the crystals, and this effect seems to increase with the time of annealing.

HIGH SPEED TOOL STEEL: METHODS OF MANUFACTURE AND HEAT TREATMENT.†

By WM. VANC. BRANDT.

The making of high speed or self-hardening tool steel differs in many respects from the making of ordinary steel. An important difference is that in making ordinary open hearth steel, for example, the charge is placed in the open hearth furnace in direct contact with the gases and flame, whereas in making tool steel the charge has no contact with the gases. In making tool steel the charge, consisting of carefully selected scrap, mostly old tools, with the addition of enough low carbon scrap to keep the carbon down within the specified limits, is placed in a small crucible. In the making of one well-known brand a black powder containing about 98 per cent tungsten is then added, so much of this powder being added to each

pound of scrap. The chrome is then added in a ferro-alloy state and the crucible tightly closed and placed in a pit or furnace. The furnaces may be made to hold as many pots as desired; a four-pot furnace is not uncommon. Hard coal is then packed around these crucibles and hot air blown through the coal, producing gas. The gas passes up and around the crucibles, until the charge is thoroughly melted. If vanadium is to be added it is put in at this point and the metal allowed to boil for about 10 min. longer.

While the metal is boiling down the molds are put in shape to receive the charge. As they will naturally be colder than the molten metal they must be thoroughly heated in order that the metal may not become chilled, which would result in a bad ingot. As a further precaution the molds are covered on the inside with lampblack, an ordinary lamp being used for this purpose. Lampblack or soot acts also as a nonconductor and keeps the metal from coming in direct contact with the inner surface of the mold. The molds used are made in sections and may easily be knocked apart, leaving the clear ingot. They vary in capacity, a common size being 3¼ in. section, with height of 10 ins.

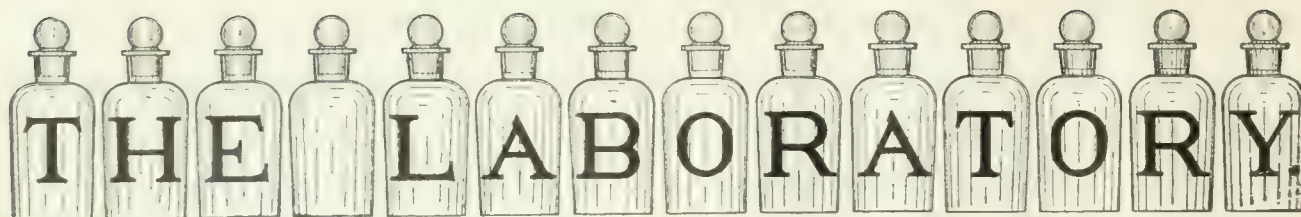
When all is ready the molds are set in place, the crucibles lifted out of the furnace and the metal poured in the mold. As soon as the ingot is sufficiently cooled the mold is knocked apart and the ingot covered with coke dust and allowed to cool gradually. This takes from three to four hours. After the ingots are cooled they are placed in an annealing furnace and annealed at about 900 degrees C., this being followed by cooling in the air. They are then forged down into bars and the bars again annealed at a higher temperature, about 1100 degrees C. After another cooling in air the steel is ready to be forged into various tools.

Great pains must be taken in forging the tool, for it is here that a great many errors creep in which result in defective steel. The tool should be forged at a good white heat and care must be taken not to upset the bar. After the forging the piece should be heated to a very high heat. It has been found that the best results are obtained when the temperature is within a few degrees of the melting point, or until the nose of the tool is almost ready to run. Various methods of cooling are employed, as, for example, the use of an air blast or quenching in oil.

After this treatment the tool is ready to be ground and great care is necessary here as in the other processes. The tool should not be ground too fast. If a dry emery is used it should be run slowly and the tool should not be forced too hard against it. It is better to use a wet emery or the ordinary grindstone. By forcing the tool too hard and grinding it too fast small hair lines appear in the structure, which later result in fracture. After grinding the tool is ready for use.

*Reference may be made here to the work of Ewing and Rosenhain on crystal growth after annealing, *Metallographist*, 1902, p. 81.

† From *The Iron Age*.



VOLUMETRIC SULPHUR DETERMINATION.

By ROBERT E. BRADLEY.*

In most cases where rapid, accurate results are wanted on liquids carrying small amounts of sulphur, the Andrews method cannot be improved upon. This consists in adding a dilute hydrochloric acid solution of barium chromate to the liquid containing not over 2 per cent of sulphur. This precipitates all sulphur as barium sulphate, and sets free one equivalent of chromic acid. An excess of ammonia is now added, giving ammonium chromate, and precipitating all excess of barium chromate, since this is insoluble in alkaline solutions. The barium chromate and sulphate are filtered off, and the filtrate is acidified with concentrated hydrochloric acid, potassium iodide added, and the liberated iodine titrated with thiosulphate.

In the case of chlorinated brine as in electrolytic plants using a mercury cell, this is quite inapplicable unless all oxidizing matters such as chlorine and chlorates are first removed, since they would be carried through and liberate iodine from the acidified potassium iodide.

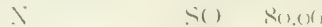
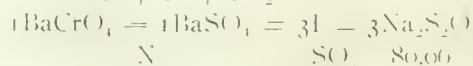
Attempts to remove this chlorine and chlorate by boiling initially with concentrated hydrochloric acid were impractical on account of the necessity of long boiling and great difficulty in determining when the strong hydrochloric acid fumes no longer carried chlorine in escaping. Here, on account of the relatively small amount of sulphur present, a comparatively small amount of oxidizing material left would induce a large error. Attempts to reduce all oxidizing matter by acid and pure zinc were hopeless on account of the length of time required and the uncertainty of completion.

In this case the only method (excepting the gravimetric) capable of giving good results in a very short time was the following:

Dissolve two to four grams pure barium dichromate in one liter of normal hydrochloric acid. Determine the strength of this by taking a known volume, adding strong hydrochloric acid, potassium iodide, and titrating the liberated iodine with tenth normal thiosulphate. It will be necessary to allow this solution to stand about three minutes to allow the liberation of iodine to be completed, since the reaction is not instantaneous.

To a known volume of chlorinated brine add a known volume of barium chromate solution estimated to be in excess, and warm to throw down all barium

sulphate. Then add a slight excess of ammonium hydroxide to precipitate all the excess of barium chromate. Filter off the barium sulphate and chromate and wash with dilute ammonium hydroxide. The filtrate, containing the chromic acid as ammonium chromate, and all the oxidizing matter, is discarded. The precipitate of barium chromate is dissolved on the filter by dilute, warm hydrochloric acid, and the filter well washed with the same. This solution is then made strongly acid with hydrochloric acid, potassium iodide added, and then liberated iodine titrated by tenth normal this sulphate. This result will be evidently lower than the former by the amount of barium chromate removed as barium sulphate and chromic acid.



$$\therefore 1,000 \text{ cc. } - \frac{\text{Na}_2\text{S}_2\text{O}_3}{10} = \frac{\text{SO}_3}{30} = \frac{\text{I}}{30} = 2.667\text{gSO}_3$$

i. e., ten grams of SO_3 in the amount taken for analysis will be equal to the difference in the number of

$\frac{\text{N}}{10}$ cc. of — thiosulphate used multiplied by 0.02667.

Primitive methods are employed by the Chinese in manufacturing sugar. During the harvesting of the sugar cane a number of small rough sheds are erected for the manufacture of the sugar. Outside each shed there is a set of pressing apparatus, driven by two oxen or buffaloes. Inside are three or four large iron pans, placed over a shallow ditch, which constitute the boilers. The pressing apparatus is also simple and inexpensive. It consists of two granite rollers or wheels, fixed perpendicularly to a wooden frame. These wheels are turned by a long lever, one end of which is fixed to the upper axle of one of the wheels, and the other connected with the yokes of buffaloes. The animals are driven round in a circle, and small bundles of sugar cane fed between the revolving rollers. The juice flows into a large earthenware receptacle beneath the wooden frame. Any person with ordinary intelligence can work the pressing apparatus, and it is generally managed by the farmer and his family, but a little skill is necessary for boiling the juice, and this is usually done by an expert. After the juice, previously mixed with a little lime and tung oil (wood oil), is boiled to the required consistency, it is removed from the boilers by a large ladle and put in shallow bamboo pans.

*Winchester, Mass.

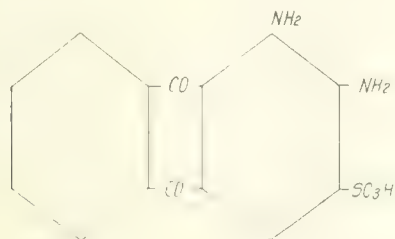
A NEW REACTION FOR COPPER.*

Doctor Rudolpf Uhlenhuth.

There are a great number of methods for the qualitative and quantitative determinations of copper. The simplest way is to introduce a piece of clean iron wire into the weakly acid solution and await the deposition of the copper after a few hours.

With solutions which may be filtered the color and precipitation methods may be used. The precipitation method is used when the solution contains much copper and together with the electrolytic method is often used for quantitative determinations. There are also many colorimetric methods for dilute solutions. Since a scale of colorimetric samples can easily be made it follows that this method can also be used quantitatively.

A new, very sharp reaction has been found through the observation that a solution of 1, 2-Diamidoanthrachinon-3-sulphonic acid in dilute alkali takes on an intense blue color with copper salts and that this color is easily distinguishable even at high dilution.



Especially so if one uses an alkaline solution of the diamidoanthrachinon sulphonic acid as a comparison.

The solution is prepared by dissolving 0.5 gms. of 1, 2 diamidoanthrachinon sulphonic acid in 500 cms. of water with the addition of 40 ccms. of con. sodium hydroxide solution of 40° Bé.

It is possible to detect copper with this solution even though all others fail. The D. solution is allowed to run into copper solution and the blue color is instantly formed.

The blue color is easily noticeable with 0.0000019 gm. copper in 1 ccm. of solution (= 1.9:1000000), the limit is 0.00000019 gms. in 1 ccm. of solution (= 1.9:10000000).

This reaction should be of use in the inspection of foods, also in pharmaceutical and physiological chemistry. The reaction seems to be characteristic, since other metallic salts do not give this reaction. The testing of the acid was carried out in the Farbwerke of Meister Lucius und Brüning in Höchst a M.

In the Province of Santa Fe, Argentine, there are 2,951 industrial establishments, 2,426 being in the hands of foreigners. The sugar mills have an annual output of nearly \$13,000,000 United States gold, the 43 flour mills nearly \$8,000,000.

A MODIFICATION OF ESCKA'S METHOD FOR SULPHUR DETERMINATION IN COAL AND COKE.

BY CHAS. R. M'CABE.*

In employing Escka's method for the determination of sulphur in coal and coke the writer has adopted a modification which saves time without sacrificing accuracy.

After burning the weighed portion of the sample mixed with the Escka mixture, the mass is dissolved in hydrochloric acid. About 10 cc. of 1:1 acid for every gram of Escka mixture used suffices for rapid and complete solution. The solution is nearly neutralized with ammonia, one-half cubic centimeter of hydrofluoric acid added, and the sulphates precipitated with barium chloride in the usual way. Results accurate.

The saving of time is due not merely to the fact that the sulphates are more quickly obtained in solution for precipitation, but also to the fact that satisfactory precipitation is more easily effected. This is owing to the circumstance that the solution is heavier and may be kept more concentrated than is the case when the usual plan is employed.

A SENSITIVE REACTION FOR GLUE.†

By EUGENE, SCHMIDT, ENGINEERING CHEMIST.

If ammonium molybdate solution, as is used in the determination of phosphorous is added to a water solution of glue a white amorphous precipitate is formed. This precipitate is characteristic and a closer study of this reaction gave me interesting results.

If a solution of simply ammonium molybdate is added, no precipitation occurs; but if a few drops of nitric acid are added the precipitate comes down; the precipitate is more intense, falls quicker and the supernatant liquid is practically clear. The use of neutral ammonium molybdate solution with the subsequent addition of acid has, as will be seen from the tests, certain advantages.

QUALITATIVE TRIALS.

In the following tests a solution of 1 gn. of glue in 500 ccms. water and a solution of 3 gms. ammonium molybdate in 250 ccms. of water were used.

(1) Glue solution and ammonium molybdate. No precipitate.

(2) Glue solution and ammonium molybdate. Boiled 3 minutes. Gives a very weak turbidity.

(3) Glue sol. and amm. molybdate. 2 drops dilute nitric acid. Immediate white precipitate. When heated it gradually dissolves. After cooling a strong turbidity remains.

*Laboratory of the Lima Locomotive & Machine Co., Lima, O.

†Translated from the *Chemiker Zeitung* for the Chemical Engineer.

*Translated from the *Chemiker Zeitung* for the Chemical Engineer.

(4) Glue solution + ammonium molybdate. Boiled 3 minutes and then added dil. nitric acid. Gives a strong cloudiness, but no flocculent precipitate.

(5) If an excess dilute HNO_3 is present neither the ppt. or turbidity are produced.

(6) Glue solution + ammonium molybdate solution boiled 3 minutes gives no ppt. and only slight cloudiness when an excess of HNO_3 is added.

In the action of ammonium molybdate and glue the following must be taken into consideration. If dilute HNO_3 is slowly added to a glue solution, a ppt. is formed which does not entirely dissolve on addition of an excess of acid, but passes over into a cloudiness. If the acid is added rapidly the precipitate is formed, but dissolves immediately, and the solution remains clear.

(7) In place of HNO_3 dilute H_2SO_4 or HCl may be used.

(8) Dilute acetic acid gives a turbidity which becomes stronger on heating.

(9) Tartaric acid gives a turbidity which remains unchanged by boiling.

(10) Dilute oxalic acid gives a precipitate which disappears on boiling, but reappears on cooling.

(11) If dilute HNO_3 is added to glue, and molybdate solutions and the precipitate allowed to stand for some time, the ppt. will dissolve in the excess HNO_3 only after strong heating or shaking and a slight cloudiness remains.

The above precipitate dissolves readily in con. HNO_3 , and in con. HCl , less in con. H_2SO_4 it barely dissolves in 80% CH_3COOH and the last solution will not become clear, even after extended heating.

(12) The normally produced precipitant and the supernatant liquid acquire a bluish-green color after a time.

The above reactions characterize the new glue reaction.

The reaction is not only intensive and characteristic, but is also exceedingly sensitive as the following tests will show.

(1) 1 ccm. glue=.002 gms. glue+5 ccm. mol. sol.+1 drop dilute HNO_3 gives an intense ppt.

(2) .5 ccm. glue sol.=.001 gm. glue+5 ccm. water+2 ccm. mol. sol.+1 drop HNO_3 give an intensive ppt.

(3) .25 ccm. glue sol.=.0005 gm. glue+5 ccm. water+1 ccm. mol. sol.+1 drop HNO_3 give a strong turbidity. After one hour a ppt. formed on the bottom of the test tube.

(4) 1 drop glue sol.=.00007 gms. glue+2.5 ccm. water+1 ccm. mol. sol.+1 drop HNO_3 gives a very plain turbidity. After one hour a ppt. forms.

(5) .00001 gms. glue gives a distinct turbidity if less glue is present, the turbidity appears only after a time.

One sees therefor, that this reaction is far ahead of

others in sensitiveness and intensity. It is also characteristic for glue alone, since trials with other substances showed that only con. solutions of gum arabic, linseed, and white of egg give a weak turbidity with ammonium molybdate and HNO_3 which can easily be distinguished from the glue precipitate.

Gum arabic gives only a weak turbidity which becomes more intense on heating. Linseed acts the same. The new reaction is good for the examination of sized goods and of sizings.

It is sufficient to boil the sample with water, to concentrate the extract on the water bath, and then determine the glue in the manner described. In order to work with only one solution the following is recommended: 3 gms. ammonium molybdate in 250 ccm. of water and 25 ccm. HNO_3 of sp. g. 120.

QUANTITATIVE TESTS.

For these tests a solution of 1 gm. of good quality Gelatine glue in 250 ccm. water was used. The viscosity of the glue was 1,693. 25 ccm. of the solution were heated in a heater with 200 ccm. water. 50 ccm. ammonium molybdate sol. and 5 ccm. nitric acid were then added and well stirred. After 15 minutes the ppt. had settled. The clear liquid was run through a rapid filter paper which had been dried at 100° and weighed, and a ppt. washed with water to which had been added a few drops of nitric acid. The ppt. is then brought on to the filter and washed until the washings show no residue on evaporation in platinum.

The ppt. and paper are then dried at 80° , then 100° to constant weight and weighed.

No.	Gms. glue used.	Gms. glue found.	Found %	Difference %
1.....	0.1	.1058	105.8	5.8
2.....	0.1	.1046	104.6	4.6
3.....	0.1	.1055	105.5	5.5
4.....	0.1	.1039	103.9	3.9
5.....	0.1	.1053	105.3	5.3
6.....	0.1	.1002	100.2	0.2

The precipitate is bluish green after drying and sticks to the paper. When burnt in a crucible and the residue heated it sublimes in shining yellow crystals. I intend to extend these tests on the value of this method for quantitative determinations since I believe that by this manner an accurate and simple method may be found.

During the five months ended August 31, 1910, the cotton yarn spun by the mills of India amounted to 272,973,079 lbs., and the amount of goods woven during the same period was 414,515,014 lbs., as compared with 283,300,929 lbs. of yarn and 375,599,626 lbs. of woven goods in 1909, and 268,408,062 lbs. of yarn and 320,797,405 lbs. of woven goods in 1908. Several mills closed down during 1909-10 and others shortened time in consequence of high prices of cotton and low price of the manufactured product. The hand-loom industry, however, was more active and prosperous than ever.

METALLURGY.

ELECTROLYTIC LEAD REFINING PRACTICE OF THE CONSOLIDATED MINING & SMELTER CO. AT ITS SMELTER AT TRAIL, B. C.

The smelter and lead refinery of the Consolidated Mining & Smelting Co. of Canada, Ltd., is located at Trail in the West Kootenay District of British Columbia. Practically all of the lead ores and most of the other custom smelting ores of the interior of the province are treated at the Trail plant. For the year ending June 30, 1910, the ore smelted averaged 9,500 tons per week of all classes, of which 1,100 tons were lead and dry ores or concentrates. In the September, 1910, issue of *Mines and Minerals*, Mr. J. M. Turnbull, Mining Engineer of the company, gives an excellent description of the Trail smelter and lead refinery. The following is an abstract of his account of the lead refining practiced at Trail. The Betts fluosilicate process is used. There are 240 vats, measuring 3 by 8 ft. by 3½ ft. deep, made of coast fir and lined with asphalt. They are arranged in six double rows in a series of cascades and the electrolyte is kept in circulation by means of pumps. The dynamos generate 3,500 amperes at 80 volts, and the vats are operated with a current density of 16 amperes per square foot of cathode area, the voltage averaging 0.32 per vat. Each vat contains 20 anodes which weigh 370 lbs. each. The anodes are cast with shoulder lugs which rest on copper bars on the sides of the vats. They are spaced 4½ ins. apart. Each vat contains 21 cathodes made of lead sheets 1-16 in. thick, which are hung upon copper bars measuring ½ by ¾ in. by bending the top part of the sheets once round the bars. The cathodes are made slightly wider and deeper than the anodes in order to prevent short-circuiting.

The electrolyte is lead fluosilicate with an excess of fluosilic acid. It contains 12 per cent of acid and from 5 to 6 per cent lead. The acid is made by mixing fluorspar, silica, and sulphuric acid in a cast-iron pan, and the fumes are condensed in towers, with water spray, which is used over and over again until its strength reaches 30 per cent acid. This is freed from sulphuric acid by the addition of lead dross. It is found advisable to add from ½ to 1 lb. of glue to the electrolyte for every ton of lead; this helps the deposit on the cathode to be coherent and so prevents short-circuiting.

The refining process takes about 8 days after which the remaining parts of the anodes amounting to 15 per cent are sent back to the melting pot. Most of the gold and silver slime adheres to the anode scrap and it is removed by washing and scraping in a special

tank. Some of the slime settles at the bottom of the vats and is removed about once a month. The cathodes are washed and melted and cast into pigs of a size to suit the market. For instance those sold to China weigh 190 lbs. to suit the coolie transport. An average analysis of the pigs shows 99.99 per cent lead, with minute amounts of zinc, silver, iron, copper, tin, and antimony, but no trace of arsenic or bismuth.

The slime is carried in copper cars to the treatment plant, where it is first washed with hot water to remove the electrolyte. The wash water is drawn off and evaporated by steam coils to recover the acid. After filtering, the slime is placed in cars and dried in the furnace flue, and when dry is melted in a water-jacketed reverberatory which is lined with magnesite brick. The impurities are here removed by oxidation, and the metallic fume is caught in cooling flues and returned to the blast furnaces. The slime averages 35 per cent silver, 0.035 per cent gold, 25 per cent antimony, 20 per cent arsenic, 8 per cent copper, with some iron, bismuth, and silica, and traces of tellurium and selenium. The doré metal is parted in sulphuric acid and the silver sulphate is run into steam-heated vats where the silver is precipitated as a slime by means of copper bars. The silver slime and the residual gold slime are melted and cast into bars, the silver being 999 fine and the gold 995 fine. The remaining solution contains copper sulphate which is recovered and sold as bluestone.

When commercial conditions warrant it, the antimony is recovered from the slime by a process devised by A. J. McNab, the superintendent of the lead plant. By this method the slime is boiled with sodium polysulphide, and the solution electrolyzed between lead anodes and steel cathodes both in sheets. The deposit on the cathode is a dense hard antimony containing 2 per cent arsenic, which can be removed by melting with alkaline fluxes. The remaining slime is roasted in a muffle furnace and afterwards treated with 10 per cent sulphuric acid which extracts 90 per cent of the copper and 10 to 75 per cent of the silver. This solution is heated in the presence of copper and the silver precipitated as before. The residual slime is then treated in the reverberatory.

The discovery that alcohol can be extracted from the fruit of the carob tree has caused this article to be looked upon as much more valuable than formerly. This tree is a leguminous evergreen found in Spain, Italy, and the Levant, and its fruit, commonly called the "carob bean," is about 1 in. wide and from 6 to 8 ins. long.

THE ELEMENT OF CHANCE IN THE SAMPLING OF ORES.*

By LEWIS T. WRIGHT.

In the sampling of an ore a portion called the "sample" is taken to represent the whole bulk. To perform this operation properly it is necessary to observe such precautions as will defeat unfair methods and control avoidable disturbing influences. If all the particles of ore were homogeneous or alike in grade there would be no need for precaution in the taking of any portion as representative of the whole. Any one portion or "sample" would be like any other portion and equally and truly representative. But ores are more heterogeneous than homogeneous, and therefore in the taking of any representative fraction special precautions and apparatus are required to avoid a bias that may be either accidental and mechanical or intentional and personal. When these precautions are entirely successful in their aim the element of impartial chance is free to exhibit its capricious vagaries undisturbed by the influences of carelessness, trickery or partiality. Chance, which is the operation of a cause or causes that we cannot predict or control, in the long run is impartial, sometimes leaning to one side, sometimes to the other, and by taking a sufficient number of results or chances the deviation of their sum from the true average can be made to balance within a small range of error. All that can be done in the matter of sampling is to mix and to take a portion. Mixing produces unevenness; an even mixture of different things can only be obtained by counting, sorting and ranging them.

Perhaps the simplest way of demonstrating this unevenness, or deviation from the average, of a sample of mixed things is to take a sample of four cards from a shuffled pack. It may be a matter of indifference whether the four cards be taken consecutively from any part of the pack or separately from four different places. That has no influence on the average result. The average sample of one of each kind comes but about once in 40 times. The samples can deviate from the average up to four of one kind or a deviation from the true average of 300 per cent. On making the sample ten times larger it will be found that the percentage of deviation, though large, is distinctly reduced. In a number of trials the greatest deviation from the true average of 10 of one kind was 18, or an error of 80 per cent. Thus by making the sample 10 times larger the maximum error or deviation was reduced $3\frac{3}{4}$ times. On making up samples that should have contained 100 of a kind a sample with only 76 was found, or an error of 24 per cent. Here the maximum error was reduced 3 times by increasing the size of the sample 10 times. A well-known formula much used by scientists for the calculation of the mean error of the mean of a num-

ber of observations which is based upon probability, implies that the error is reduced according to the square root of the number of observations. Thus by increasing the number of observations 10 times the error would be reduced 3.17 times. To reduce the error of 300 per cent possible with the above samples that should contain one of a thing to a possible error of 1 per cent—that is, to reduce it 300 times—we would have to take a sample of a size that should contain on an average 300 by 300, or 90,000 of that thing. It is easy to satisfy one's self that samples which on an average should contain 10,000 things may deviate 1 per cent from the true average. This exemplifies the limits of the unevenness of impartial mixing of heterogeneous things. Impartial mixing is the basis of sampling.

This is the principle of the "safe" weight of the sample. It is a question relating to numbers. We do not know the number of particles of mineral or of gangue or part mineral part gangue in our samples.

The question of what is a "safe" weight of sample has to be studied empirically or by trial, and the best way of doing so is to study extreme cases. In the following brief summary of investigations each individual assay, unless otherwise stated, was made on a portion of the ore sample ground to 100 mesh or finer, commonly called the "pulp," and weighing one-half of an "assay ton." The assay ton, or A. T., for short, weighs 29,166.66 milligrammes, and each milligramme of gold obtained per assay ton represents one ounce of gold per ton of ore. The results are stated in ounces per ton, generally to the second, and more rarely to the third, place of decimals. The balance used in weighing the buttons is sensitive to 0.005 mg., which corresponds to 0.005 oz. gold per ton, and the buttons are weighed to the nearest 0.005 mg. It is usual to make the fusions and assays on portions of $\frac{1}{2}$ A. T., consequently the results reported in ounces per ton are correct so far as weighing is concerned to the nearest 0.01 oz. gold per ton, and deviations of more than 0.01 are due to other causes than avoidable errors in weighing. In commercial practice it is often agreed between the buyer and seller that differences of 0.03 oz. per ton or less shall not be disputed or submitted to umpirage.

In order to throw light upon the extent of differences arising in good assaying practice a mixture of sand and limestone of 100-mesh fineness was soaked in a dilute solution of gold chloride to compose an artificial gold ore of about 2 oz. per ton and of homogeneous content. The mixture was dried and calcined with the intention of coating each particle with a film of gold so that the particles should be as nearly equal in gold content as art could arrange. If the intention was successful then the vagaries of chance would be reduced to a minimum, and each $\frac{1}{2}$ A. T. would contain well within the sensitiveness of the assay balance the same quantity of gold, and the visible

*From the Mining Magazine.

deviations between one assay and another would be due to error in assaying.

In the first set of six assays there were five of 1.92 oz. and one of 1.94. Here there was an extreme difference of 0.02 oz. per ton. In a second set of four assays two were 1.92, one of 1.94 and one of 1.91. Here there was an extreme difference of 0.03. In another series 10 results of 1.915 were obtained with one of 1.895, the extreme difference being 0.02 oz. per ton. Combining the 24 observations, the mean is found to be 1.918, with a mean error of + 0.0017 and an extreme difference between observations of 0.045. It would thus seem that with a homogeneous ore a difference as high as 0.045 oz. per ton, or a difference higher than the limits of good weighing can arise, due presumably to deviations in the assay operations other than those of weighing, between the highest and lowest results of a number of $\frac{1}{2}$ A. T. assays. The extreme differences between individual assays of $\frac{1}{2}$ A. T. found in the ordinary practice of sampling gold ores are very much greater than 0.005 oz. per ton.

If we want to study the magnitude of the element of chance as it affects the sampling of ores we cannot do better than to choose an extreme case, such as may be found among ores containing free gold, where the value of the mineral is high and the difference of specific gravity between it and the gangue is wide. If all the particles of gold in such an ore were free and distinct from the particles of the gangue and all the particles both of gold and gangue were of one size and that size were equal to 100 mesh, and the ore were of a grade of 1 oz. gold per ton, then in one assay ton there would be approximately 4,000,000 particles of quartz or gangue and 21 particles of gold. Of course, no one would expect to be able to take out from a bulk of such ore of 100-mesh fineness various small portions that would always contain 21 particles of gold and no more or no less for every 4,000,000 particles of gangue. We are not wizards. We would naturally and instinctively expect considerable variations from the average number of 21 particles of gold per 4,000,000 particles of quartz. A difference of one particle would make an error of 5 per cent. We could reasonably expect any number of particles, say, from 10 to 30 of gold, rather than exactly 21, neither more nor less.

In most ores that have been crushed the valuable ingredient is chiefly in the fine, and it is therefore likely that the average particle of mineral is smaller than the average particle of gangue. In other words, it is probable that in our samples there are more particles of mineral than there would be if the average size of the particle of mineral were equal to the average size of the particle of gangue. For this reason in our hypothetical ore, with all its gold free, there would probably be many more than 21 particles of gold for every 4,000,000 particles of quartz, because their average size would be smaller than

the average size of the whole ore, but, relatively to those of quartz, the number would be extremely small and the chances of deviation from the average number in any one small sample would still be very great.

In this article I want to insist upon the element of chance, which makes it impossible for us to get an absolutely true average sample from a mixture of different things. We can only get a sample the deviation of which from the true average is negligible by taking a sample containing a sufficiently large number of the particles or things to be estimated. If, for instance, we had an average, as in the case imagined, of 21 particles of gold in a sample, we would expect to get deviations of relatively great magnitude in different samples. By taking our sample so that on an average there were 210 particles, the percentage of deviation would be less, and so forth. In a general way it is supposed that the percentage maximum deviation of a sample from the true average would be reduced 3.17 times by taking a sample 10 times greater.

To show the importance of these differences the first two assays of many "pulp" of different gold ores containing free gold were noted. The average difference between the two assays of each of 24 *consecutive* samples of an ore marked (B) was 0.043 and included differences ranging from nothing up to 0.23, which was the maximum difference. Thirty-six sets of 2 assays of each of 36 *consecutive* samples of an ore marked (A) gave an extreme difference of 0.33 and an average difference of 0.05. Sixty-four sets of 2 assays each of 64 *consecutive* samples of (C) gave an extreme difference of 0.6 and an average difference of 0.068.

A gold ore known among assayers as being "spotty" (that is, giving wide differences when assayed) in 12 assays gave an extreme difference of 0.345 between two $\frac{1}{2}$ A. T. assays, or a difference of 0.60 oz. on a total assay value of 2.93 oz. per ton, a difference of 23 per cent.

The evil effect of such large differences between individual assays can be neutralized by making a larger number of assays and taking the average of them. In the parlance of ore sampling, $\frac{1}{2}$ A. T. is not a "safe" sample of such ores.

Before discussing the question of weight as affecting the "safety" of a sample of such 100-mesh pulp or one that with another of the same weight might only differ within the limits of good assaying, for example, 0.045, the mean of which would be 0.0225, it is well here to refer to a series of experiments made with the "pulp" as delivered to the assayer. This pulp is commonly ground so that all of it will pass through a 100-mesh screen, and a great many experiments were made with such pulp, along the following lines: First, could a method of mixing the pulp be discovered that would render the distribution of the gold more even than is indicated by ordi-

nary practice? Could these extreme differences between $\frac{1}{2}$ A. T. samples be reduced?

Second, and conversely, could the pulp be so handled by the assayer, on the rolling cloth or paper, within the limits of fair mixing or selection of the sample so as to increase these differences?

Both of these lines of investigation failed to show that the extreme differences could be favorably or unfavorably influenced. The pulp was divided in every conceivable way. The $\frac{1}{2}$ A. T. sample was taken in small portions from over 100 different parts of the bulk or as one portion from one part of the bulk. A stream of the pulp was cut a number of times by a miniature time-sampler and the $\frac{1}{2}$ A. T. was composed of these cuts. The pulp was rolled in various ways, thought likely to be conducive to the segregation of its mineral or to its perfect distribution, though it is difficult to conceive how segregation could be induced by the slow rolling of a material the particles of which tend to cling to and lock into each other.

I shall not enter into the details of experiments made upon the finer grinding of the samples. The conclusion reached by those engaged in the work was that finer grinding was of no assistance in reducing the differences, and that as a size for the assayer 100 mesh or even less was practically as safe as 200 mesh. The results were always the same. It was as impossible to combat the element of chance as it would be in the case of different colored balls taken blindfold from a bag or of cards dealt from a fairly shuffled pack.

There is a well-known formula used by physicists and chemists for the purpose of calculating the mean error, plus or minus, of the mean of a number of observations. It is:

$$F = \frac{\sum d^2}{n(n-1)}$$

in which

F=mean error of the mean of all the observations.

d=deviation of each measurement from the mean.

$\sum d^2$ =sum of the squares of the deviation.

The satisfaction, if any, we get from this formula, is to learn that we can only reduce the mean error of the mean in the proportion of the square root of the number of observations. If we want to reduce the mean error of the mean to one-tenth of its magnitude we must take 100 times as many observations.

In the trials of pure chance, to which I have referred, and wherein the true average is known, the maximum error found was reduced somewhat less than in the proportion of the square root of the number of observations. In the case of our pulp samples, unless we assay the whole bulk, we are at the disadvantage of not knowing what the true average is. Some portion of the differences noted must be due to differences in assaying, but this latter source of difference has, relatively to the other, been shown to be small.

Given an ore that with samples of $\frac{1}{2}$ A. T. exhibits a difference between the highest and lowest of 0.6, if we wanted to take samples sufficiently large to exhibit a difference no higher than 0.06, we should (neglecting the possible but smaller error due to assaying details) have to make the weight assayed as high as 50 A. T. If by convention it were agreed that a "safe" weight of sample should be one that did not differ from another sample of the same weight more than 0.06, we should have to make that weight for such an ore 50 A. T. This is 10 times as heavy as the sample usually distributed by the sampling works to the seller, buyer and umpire. 50 A. T. would be the smallest safe division we could make of such a pulp.

Every time a sample is divided, and in the sampling operation it is divided many times, there may be an error, which will be important according to the number of particles of the sample and the character of the ore.

Having determined for an ore or class of ores the safe weight for a sample reduced to 100 mesh, can we then from this knowledge alone determine the safe sample-weight of any coarser size?

H. A. Vezin, who is identified with this question of the safe weight, having found that of the pyritic ores of Gilpin county, Colorado, one ounce or 1 A. T. was a safe weight for a sample that had passed a 20-mesh screen, for all other sizes larger multiplied this weight by a number that represented the number of times the volume of the largest sized particle was greater than the volume of a 20-mesh particle—that is, by a number proportional to the cubes of the sizes of the largest particles. Vezin's statement that the safe weights of another larger size than 20-mesh could be obtained by multiplying the ascertained safe weight of the 20-mesh size by the number of times the volume of the larger size is greater than the volume of the smaller size could only be founded upon the assumption that the ratio of the number of particles of the richer mineral and the poorer gangue was the same in all the stages of the crushing of the ore from coarse to pulp, and that during the grinding the mineral and the gangue were crushed at the same rate.

We know that the fine ore in any stage of crushing contains more of the valuable mineral than the coarse. The valuable component is chiefly in the fine ore. When the pieces of ore are 2 or 3 inches in diameter we do not find pieces of gold that are 2 or 3 inches in diameter, nor do we notice similar conditions in other smaller sizes. The particles of the richest mineral, namely, gold, in ores containing free gold, are in size obviously smaller than the other particles of the ore.

In the following experiment (Table I) a gold ore was sized and the various sizes were assayed with results as stated.

TABLE I.

ORE E			ORE F			ORE C		
Size in mesh	Percentage by weight	Assay Oz. per ton	Percentage by Weight	Assay Oz. per ton	Size in mesh	Percentage by weight	Assay Oz. per ton	
On 8	69.88	1.72	62.41	2.02				
8 to 16	11.36	3.41	13.90	2.60				
16 to 30	6.70	4.29	9.33	3.74				
30 to 60	3.67	4.88	5.54	4.45				
60 to 120	2.75	6.30	2.25	6.34	60 to 80	15.8	0.44	
Through 120	5.69	8.81	6.57	8.63	80 to 100	16.	0.49	
					100 to 120	10.2	0.69	
					120 to 150	4.6	0.90	
					150 to 200	9.6	1.25	
					Through 200	12.1	2.73	
					Loss (dust)	1.4		
Calculated average		2.75		2.93				1.56

Vezin's principle that a definite ratio between the weight of the sample and the size of the largest particles shall be preserved through the whole of the crushing operation is intelligible if the relation between the number of particles of the mineral and of the gangue remains constant, otherwise it would be purely arbitrary. If it has any merit it errs on the safe side because in the coarser sizes the proportion of the valuable mineral particles is, owing to their fineness, much greater than in the finer sizes.

His table, which applies particularly to the pyritic ores of Gilpin county, Colorado, is thus given in Tofman's "The Metallurgy of Lead:"

Diameter of largest particle in inches—

1/25 1/12 1/6 5/16 5/8 1 1 1/4 2 1/2

Minimum weight of sample in pounds—

1/16 1/2 4 32 256 2048 16,348

This table starts with a sample of 20 mesh or 1/25 of an inch, for which an assay weight of sample of 1/16 lb., or about 1 A. T., was taken.

It is difficult to think of any method other than that of actual trial for approximately deriving the safe weight of all sizes from the safe weight of one size as determined by trial. To construct a curve to serve for any size we want three points or the known safe weight of three sufficiently different sizes, at least, and the curve then would be true only for that ore or class of ore. For the ores discussed in this paper I had the opportunity of making a similar comparison between the contents of two samples of the same ore taken at 1/8 in. and at 100 mesh, the latter being the size to which it is reduced before it passes into the hands of the assayer. In the sampling of these ores two cuts of from 50 to 100 lbs. each were taken when the ore was of a maximum size of 1/8 in. According to the standard practice obtaining in the United States this weight of sample is considered

safe for that size. The volume of 1/8 in. is 11,737 times that of a particle of 100-mesh size. According to the practice recommended by Vezin, if 50 A. T., or 3.2 lbs., were the safe sample at 100 mesh, then at 1/8 in. we would have to take 373,584 lbs. as a safe sample. In Table II is given the maximum and average differences noted between pairs of samples of different ores at 100-mesh and 1/8 in. maximum size.

It will be noted that generally the 50 to 100-lb. sample at 1/8 in. is safer than 1/2 A. T. at 100-mesh, both in respect of maximum and average difference. In this case the difference may be the result of the accumulation of deviations arising in different divisions of the sample, because in reducing the bulk of a sample at 1/8 in. size to a few pounds of 100-mesh pulp it has to be crushed and divided successively into halves. This division is made six times. At each division a difference, large or small, is undoubtedly originated, and these differences may not always be neutralized by the next division, and thus the differences may be cumulative and not truly representative of the difference between the two 100-mesh portions taken at the 1/8 in. stage. This is the difficulty that pursues the ore-sampler at every step and is particularly noticeable with special ores. If one-half of the difference be attributable to an original difference at the time of dividing the 1/8-in. sample, then for the ore (C) with a difference of 0.395, the weight of a sample to show no greater difference than 0.06 would be about 815 lbs. If the whole of the difference is attributable to the same cause, then the weight would be 3,260 lbs., a quantity immensely smaller than required by the old-fashioned rule of the cubes of the largest particles.

It is the practice in Western American sampling plants to distribute the samples of 100-mesh pulp in 5-oz. packages to the shipper, buyer, and, when re-

TABLE II.

Ore	Difference between pairs of 1/2 A. T. assays		Difference between pairs of 3 A. T. assays including the equation of the two assayers		Assayers	Difference between pairs of 50-100-lb. sample weights at 1/8 in.	
	Maximum	Average	Maximum	Average		Maximum	Average
B	0.23	0.043	0.124	0.045	V and W	0.195	0.034
			0.164	0.040	V and X		
			0.19	0.049	Y and Z	0.395	0.060
C	0.60	0.068	0.19	0.050	Y and W		

quired, the umpire. If a number of results obtained by different assayers are arranged in horizontal lines, each line containing the results found for the same ore, and if each assayer be given a separate column, we can compare both the work of the assayers among each other and also the natural unevenness of the different portions of the same 100-mesh pulp as divided and submitted to the different assay offices. On adding the columns in corresponding groups of 10 or 20 horizontal lines we obtain the total results of the different assayers on corresponding samples. These totals generally harmonize, except for certain differences that are usually one-sided.

Thus, for example, on a number of results assayer W will be higher or lower than some other assayer by an amount that for each sample seems trifling though in the aggregate it may be quite important. This one-sided difference in the totals is distinct and characteristic. It may be called the equation of that particular assay office and represents the sum total of the methods and persons involved in the work, and it may be large enough to be serious. If we examine the items in the horizontal lines we note differences sometimes larger than is at all usual between the work of the best assayers, but these differences are not all in one direction; they are capricious, erratic and impartial. These differences are mainly due to the natural unevenness of the ore.

To illustrate this unevenness of pulp, the assay values of different portions of 5-oz. taken from the same bulk of 100-mesh pulp, of which 3 A. T. were used in each of the results reported by the different assay offices, were compared and set forth in Tables III and IV. They represent the average and maximum differences found between different assays, and for the purpose of intelligent comparison I give also the differences noted between pairs of $\frac{1}{2}$ A. T. assays made in the same assay office and also those differences noted between duplicate 50 to 100-lb. samples taken at the size of $\frac{1}{8}$ in., all for the same lots of ores B and C, and from a large number of consecutive samplings.

If we turn back to the assays of an ore all of which was reduced to less than 60 mesh, given according to screen sizes, we note that for such an ore when assaying 1.56 oz. the worst that art could do by screening, when dividing such an ore at that stage into two halves by weight, would be to get one-half of an assay value of 0.3 oz. and the other of 1.26 oz., or a difference between two samples of 0.96 oz. The perfectly mixed pulp at 100 mesh of similar ore yields differences between two $\frac{1}{2}$ A. T. samples of 0.60 oz. That was the work of chance.

In order to see what would happen when $1\frac{1}{2}$ -lb. lots of ore of different sizes were divided over a single edge into two portions without any special care to avoid size separation the following results were obtained. One side of the dividing edge is called "East"

and the other "West." The samples were poured over the dividing edge from a shovel.

The number of experiments of this kind is far too small to determine maxima, yet it is instructive. There need be no surprise at the difference of 0.52 oz. at 60 mesh. A very little experience with what are called "spotty" ores will convince anyone that with the best assaying larger differences than 0.52 can arise between $\frac{1}{2}$ A. T. samples, crushed to 100 mesh.

The more we ponder on the assumption implied in the Vezin rule the more we are at a loss to understand its inception. It seems opposed to the instincts of a practical mind to imagine that this proportionality of

TABLE III.

Ore	Average assay, oz. per ton	Number of pairs	Difference between pairs of ½ A. T.			Difference between pairs of 50 to 100 lb. lots of ⅛ inch.		
			Maximum	Average	Minimum average	Maximum	Average	Maximum average
A	1.48	36	0.33	0.050	6.6	0.340	0.04	8.5
B	0.85	24	0.23	0.043	5.3	0.195	0.034	5.7
C	2.10	64	0.60	0.068	9	0.395	0.060	6.6

TABLE IV — AVERAGE ASSAY, — OUNCES GOLD PER TON.

Size.	East.	West.	Difference.
$\frac{1}{2}$ to $\frac{3}{4}$ inch.....	1.18	1.64	0.46
$\frac{1}{2}$ inch.....	1.13	1.06	0.07
	1.14	1.13	0.01
	0.985	1.12	0.135
$\frac{1}{4}$ inch.....	1.10	0.99	0.11
	1.12	0.99	0.13
	0.875	0.93	0.055
$\frac{1}{8}$ inch.....	1.32	1.58	0.26
20 mesh.....	1.48	1.435	0.045
	0.97	1.05	0.08
	1.05	1.03	0.02
	0.855	0.825	0.03
40 mesh.....	1.45	1.495	0.045
	0.99	1.00	0.01
	1.07	0.98	0.09
	0.81	0.795	0.015
60 mesh.....	1.36	1.88	0.52
	0.93	0.97	0.04
	0.98	1.00	0.02
	0.79	0.83	0.04
100 mesh.....	1.37	1.36	0.01
	1.03	0.98	0.05
	0.96	1.01	0.05
	0.82	0.795	0.025

the numbers of the particles of different things is going to be preserved through all the crushing of the ore from coarse to fine in all its stages. The general observation that the pieces of valuable mineral are generally smaller than the gangue should have discredited the idea that a fixed ratio between the weight of a sample and its largest particle should remain constant through all the stages of crushing.

To raise the matter of ore-sampling to a perfect art, the science of the unevenness of mixed things must be applied to its operations, and, this being recognized, means should be taken so that the sampling operation can be checked within itself by the duplication of the sample at any of its many divisions.

THE OPPORTUNITIES OF A METALLURGICAL CHEMIST.*

Although the day when all foundry work was carried on by "rule of thumb" is over, there are many foundries today that do not employ a chemist, and, if you were to ask why they did not have one, they would invariably reply that they did not need one. Now, the statement that a chemist is not needed in a foundry is not true, for the simple reason that there are times when heats go wrong, no matter how careful and watchful the melter may be, and a batch of metal is consigned to the scrap heap when an analysis of it would instantly show where the trouble lay and a means for remedying the same so that the metal could all be saved.

The real reason why the foundry has no chemist is that the amount of work done would not be sufficient to warrant the expense connected with the equipment and maintenance of a chemical laboratory, and it is to meet the requirements of small concerns where that is the case, that we have in all our large cities well equipped laboratories that look after the chemical work of several of these industries.

WHAT THE CHEMIST SHOULD KNOW.

In order to be of most use to any foundry the chemist connected with it must be in absolute control of the weighing, mixing and melting, for the slightest variations in weight often lead to faulty castings and other gross errors. He must have an accurate knowledge of the alloying properties of the different metals and be able to produce alloys for any desired purpose. He must know what impurities are allowed and the percentages, and he must know a way to eliminate these impurities if such a thing is possible.

Fluxes he must be acquainted with, their melting points and other physical properties. It might not be amiss at this point to name a few of the more important fluxes and state their general properties.

VARIOUS KINDS OF FLUXES.

A flux, according to our latest chemical dictionaries, is any substance or mixture used to assist the fusion of mineral substances. This assistance may be brought about in different ways, e. g., the flux may act by combining chemically with some part of the substance under treatment, thus breaking up a compound that could not easily be separated into its constituents. To illustrate we can take the metallurgy of copper. Where low-grade sulphide ores are used in the blast furnace, iron pyrites (sulphide of iron) is used as the flux; this furnishes the sulphur to combine with the copper of the ore to make the copper matte, and the iron combines with the silica in the ore, forming a slag.

The flux may also act by keeping the fire gases away from a metal or alloy and thus prevent oxidation, or, in some cases, absorption of gases; this is the principal use to which the flux is put in the foundry.

Of the fluxes in use may be mentioned the following: Borax, glass, lime, fluorspar, soda ash, cryolite and

salt. Of these, borax is by far the most valuable on account of its power to form highly fluid combinations when fused with silica and most metallic oxides. As ordinary borax contains about 47 per cent of water, it has to be melted and the water driven off by heat. The result of the fusion is known as borax glass. As borax is somewhat expensive when used alone, it may be used mixed with an equal amount of powdered glass. Although of high melting point, window or plate glass free from lead but high in lime, is to be preferred.

Lime is used in large metallurgical operations as the flux for silica, clay, etc., as, for instance, in the blast furnace in the smelting of iron ore, lead ores, etc. The source of the lime is limestone and should be as pure as possible.

Fluorspar is used as a flux for sulphates, silicates, etc., forming a very fusible slag.

Soda ash is a powerful desulphurizer; it also forms the most fusible slag with silicates.

Common salt is used very extensively to prevent access of air to a melt, and can be used either alone or mixed with other fluxes.

Cryolite is a valuable flux for aluminum because it has the property of dissolving large portions of alumina in it.

In soldering, anything that will dissolve away the oxides of the metals to be soldered, leaving clean surfaces, may be considered as a good flux. Zinc chloride is probably used in more soldering fluxes than any other substance, as it has an exceedingly low melting point and is an excellent cleaner of most metal surfaces—both points of advantage in soldering.

Silica sand is used by all blacksmiths as a flux in welding, as it combines with the iron scale, leaving two bright surfaces for the weld. I might go on almost indefinitely in this manner, as for almost every metallurgical operation—be it large or small—a flux of some nature is needed to separate the impurities from the metal.

IMPORTANCE OF TEMPERATURE.

In any melting operation it is essential not to overheat the metal, and it is the chemist's duty to determine accurately, by means of a pyrometer—of which there are many reliable makes on the market—the temperature best suited for the pouring of the melt, and then to make charts giving the weights of the various metals in an alloy and the temperature at which it should be poured. Overheating entails a loss in two ways—loss of fuel and loss of metal due to oxidation—and relying on the eye for the determination of temperature is very poor practice. I give here in degrees Fahrenheit the melting points of several of the more important non-ferrous metals:

Copper	2,008 degs.	Tin	 450 degs.
Manganese	...2,204 degs.	Antimony	...1,166 degs.
Nickel	2,615 degs.	Bismuth	 516 degs.
Aluminum	...1,215 degs.	Gold	1,940 degs.
Zinc	 786 degs.	Silver	1,762 degs.
Lead	 620 degs.	Platinum	3,187 degs.

It is needless to state the well-known law that alloys melt at a lower temperature than their constituents.

*From The Metal Industry.

THE BRIQUETTING OF IRON ORES.*

By CHEVALIER C. DE SCHWARZ.†

The question of briquetting iron ore is one which becomes of increasing importance every year, not only owing to the fact that iron ore in general, and lump ore in particular, are becoming more and more scarce, at least in Europe, but also because of the severe conditions imposed by modern blast furnaces; not only in regard to the chemical composition, but also to the physical conditions of the ore.

The use of improved and more effective explosives in the iron mines, as well as the fact that iron ores have to be transported over long distances and frequently reloaded before they reach their destination, tend to increase the percentage of small and dust ore, to the disadvantage of blast furnace working which—on account of the greater height of the furnaces themselves, the high pressure of blast, and the use of blast furnace gases for gas engines—require an ore which is not too fine nor too dusty.

The blast, being of comparatively high pressure in modern blast furnaces, will naturally blow out a considerable portion of the fine ore, which will mix with the escaping blast furnace gases, causing well-known inconveniences. This is the more objectionable, as the blast furnace gases are now, almost everywhere, utilized for driving gas engines, for which purpose the gas must be almost entirely free from dust.

The fine ore, during its downward passage in the blast furnace, advances quicker than the lump ore, charged at the same time; consequently it is not exposed long enough to the beneficial reducing influence of the carbon monoxide gas arising from below. It therefore leaves the zone of reduction in the blast furnace imperfectly freed from oxygen, and, as a consequence, clogs together as soon as it reaches the zone of carbonization, causing scaffolding, and even explosions.

A considerable number of methods for briquetting iron ore have been brought out, but none as yet which can be applied generally with success. A few of the methods introduced into practice have proved successful, but only for certain kinds of ore, or in local circumstances and conditions. Some methods which have yielded favorable results with a particular kind of ore have entirely failed with other ores. Experience has also shown that the method to be adopted must be modified to suit the nature of the ore, not alone with reference to its physical condition, but more especially with regard to its chemical composition. Most of the different methods that have been tried for the briquetting of fine-grained ore have proved to be too expensive, not alone with reference to initial outlay and cost of manufacture, but also, indirectly, on account of the great reduction of

the contents of iron in the ore which occurs owing to too great an addition of binding material.

The conditions for successful briquetting are:

1. The iron ore briquettes must have a certain resistance against mechanical influence. They must resist a pressure of not less than 2,000 lbs. per square inch, and, when dropped from a height of 10 ft. on a cast iron plate, they must not fall into dust, although they may break into pieces.

2. They must resist heat. Heated to 900° C. they may commence to sinter, but they must not disintegrate into small fragments.

3. They should be capable of being placed in water for a certain time without softening.

4. They must resist the influence of steam at 150° C. without crumbling.

5. They must possess a certain amount of porosity in order to allow the carbon-monoxide in the blast furnace to penetrate the interior of the briquette and to exercise its beneficial reducing influence. In order to test the briquettes for porosity, they are, after being dried, placed for about 25 minutes in water, during which time they ought to absorb not less than 12½ to 16 per cent of water, according to the nature of the ore.

6. The binding medium, if any is used, should not contain noxious substances (sulphur, arsenic) to such an extent as to be injurious to the quality of the pig iron produced.

7. The cost of producing briquettes should not exceed the difference in the prices between lump ore and fine ore.

The different systems of briquetting ore can be divided into those in which a binding medium is used and those in which no binding medium is used. The former may again be divided into methods employing inorganic and organic binding materials, respectively.

In most of the cases, where no binding medium is used, the briquettes have to be heated at considerable expense, but there are exceptional cases where no heating is necessary notwithstanding that no binding medium is employed. This case presents itself where the iron ore contains a certain percentage of clay, because in such cases it contains in itself a binding medium. Clay iron ore and bean ore are instances.

At Kertsch, in Russia, where bean ore is briquetted, the latter is mixed with 8 per cent of water and made into briquettes under a pressure of about 5,600 lbs. per square inch. At Ilsede, in Germany, where clay iron ore is used, it is mixed with the waste of the iron ore washing plant and with other powdery waste products coming from the rolling-mills and containing sufficient iron. The mixture, which is not allowed to contain more than 6 per cent of water, is heated to about 75° C. and pressed into briquettes, the pressure being about 4,000 lbs. per square inch. In both the above cases no heating of the briquettes is necessary, and no binding medium is used.

The results obtained at Kertsch and at Ilsede nat-

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urally suggest the addition of common clay to other iron ores (containing no clay) as a binding medium in order to make briquettes by means of a cheap medium, and without going to the expense of heating the briquettes. However, it has been found that—in order to achieve the desired result—it is necessary to add a comparatively large percentage of clay to the ore, thus indirectly reducing its contents of iron to too great an extent. Common clay contains, moreover, a high percentage of silica, requiring extra additions of limestone. The practice of using clay as a medium has therefore been abandoned, and it is only employed in cases where ferruginous clay is at disposal.

In some cases it has been sought to avoid the use of a binding medium, as well as the heating of the briquettes, by employing very high pressures, rising gradually to 11,000 lbs. per square inch, for making briquettes; but although the briquettes made in this way have been very firm and offer great resistance to mechanical influences, they expand when exposed to heat and fall to pieces; besides which they are too dense, and this considerably interferes with their reducibility.

Of all systems of briquetting iron ores without a medium, by heating the briquettes after they have left the press, the Grondal process, so called after the name of the inventor, has proved to be the most successful for magnetic ore.

The following is a short description of this process: The raw ore, containing from 27 to 58 per cent of iron, is broken up by means of a stonebreaker and then ground to sand in a ball-mill. From this sand the pure magnetic ore, the so-called "concentrate," is extracted by means of magnetic separators.

The "concentrate," containing from 67 to 71 per cent of iron, is moistened and made into briquettes about 6 in. square and $2\frac{1}{2}$ in. thick, with rounded edges. No addition is made.

The briquettes are then laid in three rows edgewise upon small iron wagons and transported into a furnace heated with generator and blast-furnace gases. Heated and compressed air is supplied by means of a Korting blower in such quantity as to produce a highly oxidizing flame, whereby the magnetic ore (Fe_3O_4) is oxidized to peroxide of iron (Fe_2O_3). The sulphur is considerably reduced during this operation.

The comparative table (Table I) gives the chemical composition of the raw ores, concentrates, and briquettes in different cases.

TABLE I.

Name of place.	Raw Ores.			Concentrates.			Briquettes.		
	Fe%	S%	P%	Fe%	S%	P%	Fe%	S%	P%
Flogberget	27.3	0.31	0.003	67.4	0.04	0.003	65.3	0.007	0.003
Herrang	40.2	1.21	0.003	67.3	0.17	0.002	65.5	0.003	0.007
Lulea	58.2	0.11	1.23	71.1	0.015	0.005	69.3	0.005	0.005
Strassa	46.8	0.03	0.015	69.2	0.015	0.003	67.1	0.005	0.003
Cornwall, U.S.S.	30.6	1.60	0.012	69.9	0.036	0.003	67.9	0.010	0.005

One furnace, about 150 ft. long and $5\frac{1}{2}$ ft. wide, produces, on an average, 200 tons of briquettes in a

week, the cost of production being at 3s 4d per ton of finished briquettes, exclusive of general expenses and royalties.

The briquettes are hard and porous, and quite suitable for the blast-furnace; but the cost of production appears to be rather high, in consequence of which the Grondal process has not, except in Sweden, been much adopted. In Sweden there are at present 27 Grondal furnaces at work, with a total output of about 300,000 tons of briquettes per year.

The reason why the Grondal process is of special value to Sweden lies principally in the fact that the working of the blast furnaces in that country is almost entirely based on the use of charcoal. The slag of charcoal furnaces being—for well-known reasons—acid, it does not absorb sulphur, consequently nearly all the sulphur in the iron ore goes into the pig iron instead of into the slag. This is also the reason why the blast-furnace owners in Sweden object to use any iron ore containing more than 0.015 per cent sulphur. From the comparative table it will be seen that the raw ore contains considerably more sulphur than the above-mentioned limit, and it is therefore absolutely necessary to reduce the percentage of sulphur in the raw ore, not only by means of the magnetic separator, but also by heating the briquettes to a high temperature (given at 1,400° C.) and in an oxidizing flame.

Although the percentage of iron in the briquettes is somewhat reduced, owing to the transformation of magnetite (Fe_3O_4) into peroxide of iron (Fe_2O_3), this disadvantage is more than compensated for not only by the expulsion of sulphur, but also by the fact that peroxide of iron (Fe_2O_3) is much more easily reduced and requires considerably less fuel in the blast furnace than block oxide of iron (Fe_3O_4).

Considering that charcoal is becoming more and more scarce from year to year, and, as a consequence, more expensive, the question of economizing fuel is of special importance to Sweden.

Experience has shown that briquettes made of Fe_3O_4 (magnetite) require in the blast furnace 300 lbs. more charcoal per ton of pig iron than those made of Fe_2O_3 , equal weights and percentages of iron being taken in each case. This difference in the consumption of charcoal represents a reduction in the working expenses of about 6s 6d per ton of pig iron in favor of Fe_2O_3 against Fe_3O_4 . The success of the Grondal process, notwithstanding its high cost, is therefore justified for Sweden, as well as in such other countries in which charcoal, together with sulphurous ores, have to be employed.

Considering that the melting point, and therefore the sintering temperature, of magnetite is comparatively low, the briquettes made of "concentrates" (Fe_3O_4) begin to sinter and to harden at a temperature of from 800° to 900° C. However, as already mentioned, the temperature is raised to about 1,400° C. in order to expel as much sulphur as possible. It may also be mentioned that the process of converting the

Fe_3O_4 into Fe_2O_3 , being an oxidation process, is accompanied by a development of heat, thus economizing fuel to a certain extent.

The suggestion has been made of using briquettes made of "concentrates" directly in the open-hearth furnace, as a decarburizing and, at the same time, enriching addition to the metal bath.

The use of iron ore in the open-hearth furnace for that purpose is already well known, but such ore must be comparatively rich in iron, and it must not contain noxious substances to such an extent as to interfere with the quality of the steel produced. From the chemical analysis, given in the comparative table, it will be seen that the "concentrates" (and therefore the briquettes made therefrom) quite fulfill these conditions, the percentage of sulphur in the "concentrates" varying, with the exception of those from Har-rang, between 0.015 and 0.04 per cent. Considering, further, that, as a rule, only 25 per cent of iron ore is used in the open-hearth furnace, the contents of sulphur in the ore added will be indirectly reduced to one-fourth (0.004 per cent to 0.01 per cent) in the finished steel, which is not high enough to interfere with its quality. In the case of briquettes made, as suggested, from concentrates, for use in the open-hearth furnace, it would not be necessary to heat them up to $1,400^\circ$; it would also not be necessary to oxidize them from Fe_3O_4 to Fe_2O_3 , and to make them porous. For use in the open-hearth furnace they can be pressed at a considerably higher pressure, whereby they would become more compact, which is made advantageous for use in the open-hearth furnace, because they will then be specifically heavier, and therefore sink deeper in the bath, instead of floating upon the surface for a long time, like red or brown ore, which, with the exception of specular iron ore, is comparatively much lighter in weight.

As Fe_3O_4 commences to sinter at a temperature of from 800° to 900° C., the briquettes made of "concentrates" will become hard at this temperature, and need not be heated above it. This advantage, together with the fact that they need not be porous (and therefore not changed from Fe_3O_4 to Fe_2O_3), makes the briquettes considerably cheaper and richer in iron when used in the open hearth instead of the blast furnace.

In order to avoid the cost of briquetting, an attempt has been made to agglomerate the fine-grained ore in a rotary furnace. This method appears to be of special value in cases where the agglomeration can be combined with the roasting of the ore, should the latter be necessary or advantageous, as, for instance, when spathic ore, brown ore, or iron ore with a high percentage of sulphur, is to be agglomerated.

The fine-grained ore to be agglomerated in a rotary kiln is mixed with coke dust (which can generally be procured very cheaply) in order to reduce the oxygen in the ore. The Fe_2O_3 in the ore can thus be partly changed into FeO , which would enrich the ore in

Fe, and, at the same time, reduce the temperature necessary for agglomerating.

Considerable difficulties have, however, been experienced with this method of agglomeration. First, it has been difficult to regulate the temperature in the rotary furnace in such a way that the necessary sintering should take place without melting. Secondly, it has been found that the ore, as soon as it commences to agglomerate, adheres to the inner fireproof lining of the rotary furnace, thus causing scaffolds, which have to be removed at considerable expense.

These difficulties are more frequently experienced in cases where coal-dust firing is employed for heating the rotary furnace, because, in these cases, the ashes of the coal, consisting chiefly of silica and alumina, mix with the ore and cause scaffolds, besides indirectly reducing the contents of iron in the agglomerated ore. The use of gas instead of coal-dust firing may, therefore, prove advantageous in this case, partly because the temperature in the rotary furnace can be more easily regulated according to requirement, and partly because no ash becomes mixed with the ore.

The use of slaked lime as a binding medium for fine-grained ore has been known for many years, and it has also been found that briquettes quite suitable for blast-furnace use can be made when the slaked lime is mixed with some granulated basic slag. Compared with other methods this process of briquetting is rather expensive and also considerably reduces the percentage of iron in the briquettes, because from 7 to 8 per cent of binding material, containing no iron, has to be used. Besides this, the briquettes have to be exposed for a considerable time to fresh air in order that the calcium oxide in the briquette may absorb sufficient carbon dioxide, because hydrate of lime as such would lose its water in the upper portion of the blast furnace, and the briquettes would therefore crumble to dust. The transformation of CaO into CaCO_3 also means another indirect reduction of the percentage of iron in the ore, not to mention that a great amount of space and extra labor are required for stacking and for keeping the briquettes, in order that this transformation takes place. This method of briquetting ore has therefore not found extensive application.

Burnt lime, either alone or mixed with clay or ashes, has also been tried, but without success. This may easily be understood from the fact that—owing to the presence of steam and carbon dioxide in the upper portion of the blast furnace—burnt lime absorbs water and carbon dioxide, thus changing into hydrate of lime. These reactions—being accompanied with a certain evolution of heat—cause "upper heat," a serious inconvenience known to every blast furnace manager. Besides these there are other disadvantages attending the use of lime. Gypsum, as well as cement, has also been tried as a binding medium. The former contains too much sulphur, and neither material resists the

influence of heat, although the briquettes withstand humidity as well as mechanical influences.

More success has been achieved by using as a binding medium a mixture of sand and slaked lime. Equal parts of these materials are mixed and ground into fine powder, and then intimately mixed with dust ore. After being moistened the mixture is pressed into briquettes, charged into boilers, and exposed for some hours to the influence of superheated steam under pressure, after which the briquettes are ready for the blast furnace. About 6 per cent of the above-mentioned binding medium is required.

This method of briquetting iron ores depends on the fact that the silica in the sand—owing to the influence of superheated steam—is converted from the insoluble into the soluble condition, ready for combination. This method has given very good results in respect to the quality of the briquettes produced, but it has not found very general practice on account of its high working expenses as well as its high initial outlay.

Another method of briquetting, based on the same principle, consists in the use, as a binding medium, of basic blast-furnace slag, which was exposed to steam pressure as mentioned before. However, as nearly 10 per cent of this medium is required to obtain good briquettes, it has been found that the percentage of iron in the ore becomes too much reduced.

Spathic ore, mixed with lime, is also successfully used as a binding medium, provided that the briquettes are exposed to steam pressure for a certain time. This process is based on the fact that hydrate of lime and carbonate of iron (spathic ore), when intimately mixed and exposed to steam pressure, become converted into hydrated ferrous oxide and carbonate of lime (limestone). The former, being of a gelatinous nature, serves as a binding medium and changes, later on, into hydrated sesquioxide of iron. This process is carried out in the following manner: A mixture of two parts of spathic ore with one part of lime, reduced to powder, is prepared. Of this powder 15 per cent is added to the ore to be briquetted, the whole moistened—whereby the lime is converted into hydrate of lime—and made into briquettes under a pressure of about 6,000 lbs. per square inch. These briquettes are packed—edgewise—on wagons, after which they are charged into boilers, where they are exposed to a steam pressure of eight atmospheres for a period of $4\frac{1}{2}$ to $6\frac{1}{2}$ hours. In case spathic ore alone is to be briquetted an addition of 6 per cent of lime is sufficient. In the latter case cylindrical briquettes of about $5\frac{1}{2}$ in. diameter and $4\frac{1}{2}$ in. in height are generally made.

The briquettes made according to this method are very good, but expensive, besides which the plant requires a high initial outlay.

Great success has been obtained in briquetting blast-furnace flue-dust, which is principally due to the fact that this by-product already possesses hy-

draulic properties, because—like cement—it contains lime, alumina, and soluble silica ready for combination. In addition to this it has been found that by adding certain salts, such as chloride of magnesium, chloride of calcium, green vitriol ($\text{FeSO}_4 + 7\text{H}_2\text{O}$), or acids, such as sulphuric or hydrochloric acid, those hydraulic properties can be raised to such a degree that very suitable briquettes can be made from the dust.

Of these additions chloride of magnesium (MgCl_2) has been found to be, generally speaking, the most suitable, because it is cheap (being a by-product in the manufacture of certain artificial manures), and because it does not contain any noxious substances. Owing to the high temperature in the blast furnace, the chloride of magnesium is decomposed into magnesium and chlorine. The former enters into the blast furnace slag as MgO , and the latter combines with hydrogen to form hydrochloric acid. The acid further combines with the free lime to form chloride of calcium, which is finally absorbed by the water used for washing the blast furnace gases. In some instances it has been observed that the iron portions of the furnace top become slightly attacked, which is ascribed to the presence of hydrochloric acid not fully neutralized.

This process of briquetting, called after the inventor, the "Schumacher" process, has given very satisfactory results, not only with reference to the quality of the briquettes produced, but also as far as cost of production and initial outlay are concerned. It must, however, be pointed out that the blast-furnace flue dust used for briquetting must be used almost immediately after having been removed from the catchers, and that, like cement, it must be protected against humidity before briquetting. The installation for briquetting flue dust should, therefore, be situated in the close proximity of the blast furnace from which the flue dust is taken.

It is to be regretted that this process can only be used for briquetting blast-furnace flue dust. With the exception that in some instances where the chemical composition of the flue dust is particularly favorable to the purpose, and addition of about 40 per cent of purple ore to the flue dust has been found to be admissible; the latter thus acting, as it were, as a binding medium to the former.

Another method of briquetting flue dust consists in the use of basic blast-furnace slag, after having been exposed to steam pressure. Owing to the influence of compressed steam the silica in the slag changes from the insoluble to the soluble variety, and combines with the lime in the flue dust to a kind of cement, which acts as a binding medium. If the lime in the flue dust is insufficient, another 4 to $4\frac{1}{2}$ per cent is added. The briquettes made in this way are quite suitable; but taking into consideration that from 8 to 12 per cent of binding medium, containing no iron, has to be used,

the contents of iron in the briquettes is too much reduced to make this method profitable.

Organic matters, such as waste lye, resulting (as a by-product) from the manufacture of sulphite cellulose, and molasses have been used for briquetting flue dust as well as other iron ores. In the former case, the lye is concentrated to a syrup, thus constituting a kind of pitch called "Zellpech" ("cellulose-pitch") in Germany, where it is used. On an average, about 6 per cent of this binding medium is intimately mixed with the flue dust or ore, and pressed into briquettes under a pressure of not less than 9,000 lbs. per square inch. The briquettes made with "cellulose-pitch" are suitable, but, as the binding medium is very expensive (40s a ton), the financial success of this method is rather doubtful.

The use of molasses as a binding medium was tried for briquetting the ferriferous residues of the aniline manufacturing process as well as for briquetting flue dust—this process being as follows: One part of cheap molasses, by volume, is mixed with two volumes of silicious sinter ("Kieselguhr"), 0.4 parts by volume of carnalite and 100 parts, by volume, of the ore to be briquetted. The briquettes are allowed to dry in the open, after which they are charged into the chambers of an annular furnace, such as is used for burning ordinary bricks. Here they are heated up to 1,000° C., when they begin to sinter and to harden.

This method of briquetting is accompanied by high working expenses and considerable initial outlay. The employment is therefore only justified when the by-product of the aniline manufacturing process, containing about 67 per cent of iron, is to be made into briquettes.

As before mentioned, this process was also tried for briquetting blast-furnace flue dust, but it was abandoned, probably on account of the too high cost.

Besides the above-mentioned methods of briquetting fine-grained ores, or blast-furnace flue dust, etc., there are numerous other methods of briquetting with or without a binding medium. In the former case soluble glass (alkaline silicate), asbestos, basic slag, naphthalin, paraffin, resinous soap, as well as other inorganic and organic materials, are used, and in the latter case smelting is conducted in a reverberatory furnace. Treatment with liquid iron, or in an electric furnace, has been proposed and also tried in practice, but in all cases without result. As there is also no reason to believe that these methods will ever be tried again, the author does not consider it necessary to go into details about them.

From what has been said it may appear that a universal method of briquetting fine iron ore, fulfilling the conditions enumerated earlier in this paper, has not been invented as yet, and probably never will be.

Taking the financial results of the different methods of briquetting into consideration, it is found that the methods of briquetting practiced at Kertsch in Russia and at Ilsede in Germany appear to be the most

economic, the working expenses, as reported, amounting only to one shilling per ton of briquettes. As already previously mentioned, this method can, however, only be applied successfully when the ore contains hydrate of alumina to serve as a medium.

The manufacturing expense of briquetting by the Grondal process amount to 3s 4d per ton of briquettes, exclusive of general expenses, redemption and royalties, for which at least 1s 4d will have to be added, thus resulting in a total expense of 4s 8d per ton of briquettes. This would be far too high to allow of the general introduction of this process. However, the Grondal process is of particular value for Sweden, where its use has become almost unavoidable in order to be able to utilize the iron ore of that country; besides, its use is, in Sweden, combined with a saving of 6s 6d per ton briquettes in fuel, thus resulting in a final saving of 1s 10d (6s 6d—4s 8d) per ton in favor of the Grondal process. Similar results may also be obtained with this process where charcoal is used as fuel in the blast furnace.

Of all the different methods employed for briquetting blast-furnace flue dust, the "Schumacher" process has given the best financial results, the cost of briquetting according to that method amounting to 1s 8d per ton briquettes only, without royalty.

The other methods of briquetting iron ore flue dust and other ferruginous residues of other industries, as described before, have given less favorable financial results, the cost of briquetting amounting from 2s 6d to 3s 9d per ton briquettes. Besides this it must be mentioned that where a binding medium is used, on an average about 7 per cent is required to be added to the ore, thus reducing considerably the percentage of iron. In several cases it has also been found necessary to stack the briquettes for several weeks in the open before they could be used, thus requiring a good deal of space and extra labor for transporting and handling.

Finally, it may be mentioned that at present an establishment for making briquettes is being erected at a Belgian works according to a new process introduced. According to this method from 2½ to 4½ per cent of binding medium, costing about 16s a ton, is used. The initial outlay for a plant turning out five tons of briquettes an hour, amounts to about £2,500, and the manufacturing expenses, inclusive of general expenses, to 1s 6d per ton briquettes. The briquettes fulfill all the conditions enumerated in this paper, and can be used within 24 to 36 hours after having left the press, no heating or any additional work being necessary. By this method, not only fine ore, but also rolling-mill and forge-scale (Fe_3O_4) have been briquetted with success, the latter having been successfully used as an enriching and decarbonizing addition in the open-hearth furnace.

The Australian government is to establish timber seasoning depots in the various states.

The CHEMICAL ENGINEER

HARRY McCORMACK, Editor.

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FORTY-THIRD MEETING OF THE AMERICAN CHEMICAL SOCIETY.

The American Chemical Society held its winter meeting for 1910 at Minneapolis, from December 28th to December 31st. There were about 275 members in attendance and the papers on the program were of considerable interest. At the meeting of the Executive Committee of the society, Dr. Alexander Smith, University of Chicago, was elected president of the society for the coming year.

Another important matter coming before the council was the policy and editorship of the industrial journal of the society, the council expressing its opinion as to the policy "that the Journal should be one to impress the members rather than one in which they may express themselves." This policy being at variance with the opinion of the present editor, his resignation was accepted and Professor M. C. Whittaker of Columbia University was elected in his place.

The general program included addresses by F. W. Clark, "Report from the International Committee on Atomic Weights"; "The Lost Arts in Chemistry," by W. T. Richardson; "The Basis of In-

dustrial Efficiency," by A. D. Little; "Synthetic Metals from Non-Elements," by Herbert M. McCoy; "Progress in Food Chemistry," by H. E. Barnard; "Mechanism of Cell Activity," by Carl L. Alsberg; "Waste Wood and Some of Its By-Products," by Geo. B. Frankforter. In addition to these addresses President McPherson of Section C spoke on "The Formation of Carbo-Hydrates in the Vegetable Kingdom," Chas. F. Burgess on "The Efficiency of the College Graduate in the Chemical Industry," and President W. B. Bancroft on "A Universal Law."

The program was too extended to give detailed mention of all the papers presented.

The society met in the following sections: Biological, Organic, Pharmaceutical, Physical and Inorganic Chemical Education, Agricultural and Food Chemistry, Fertilizer Chemistry, Industrial Chemists and Chemical Engineers.

Some of the papers in the Chemical Education Section were very interesting and instructive. The three papers presented which probably aroused the most interest were the addresses of C. F. Burgess, as chairman, the papers by Drs. Walker and Lewis on "Laboratory Course in Chemical Engineering," and by Dr. Withrow on "Points of View on Teaching Industrial Chemistry."

Believing that our readers are interested primarily in the work as presented by the Industrial Chemists and Chemical Engineers, the complete program for this division is given below.

DIVISION OF INDUSTRIAL CHEMISTS AND CHEMICAL ENGINEERS. EFFICIENCY SYMPOSIUM. PAPERS.

1. J. T. Donald. "An Improved Process for Finishing Beef Extract."
2. A. D. Smith. "Self-Recording Efficiency."
3. F. B. Porter. "Efficiency in Acid Phosphate Manufacture."
4. Walter P. Schuck. "Chemistry as a Factor in Foundry Efficiency."
5. Jas. C. Lawrence. "Note on the Utilization of Lumber Waste."
6. W. S. Williams. "The Use of Peroxide for Silk Bleaching."
7. C. F. Wood. "Economical Steam Generation."
8. C. M. Ballard. "The Importance of Eliminating Air Leaks in the Manufacture of Sulfitic Acid."
9. S. W. Parr and F. W. Kressman. "The Spontaneous Combustion of Coal." Illustrated by lantern.
10. George P. Dieckmann. "The Modern Manufacture of Portland Cement from the Chemical and Mechanical Standpoint." Illustrated by lantern.
11. Harrison E. Ashley and Warren R. Emley. "Errors in Determining the Sixes of Grain of Minerals and the Use of Surface Factors."
12. Harrison Everett Ashley. "The Utilization of Smelter Smoke in Preparing Sulphates from Clays."
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26. Geo. L. Heath. "The Exact Electrolytic Assay of Refined Copper. 1. Standard Method. 2. In Solenoid with Revolving Electrolyte."
27. Geo. L. Heath. "The Determination of Arsenic and Antimony in Copper, Including a New Rapid Volumetric Method."

THE VOLATILE MATTER OF COAL.

The Volatile Matter of Coal is the title of the first bulletin to be issued by the new Federal Bureau of Mines. The authors, Horace C. Porter and F. K. Ovitz conducted their investigations at the Pittsburg station while it was under the Technologic Branch of the Geological Survey, the work being a continuation of the fuel investigations begun several years ago at the Louisiana Purchase Exposition, St. Louis, Mo. The results obtained at that plant showed that the work of determining the fuel values of the coals and lignites in the United States with a view to increasing efficiency in their utilization would be incomplete if it did not include systematic physical and chemical researches into the processes of combustion. Hence in their later investigations the authors carried on such researches, concentrating attention on those lines of inquiry which promised results of greatest economic importance. This bulletin is therefore a report on an investigation of the volatile matter in several typical coals—its composition and amount at different temperatures of volatilization.

Quoted directly the authors say: "The investigation has already shown that the volatile content of different coals differs greatly in character. The volatile matter of the younger coals found in the West includes a large proportion of carbon dioxide, carbon monoxide, and

water, and a correspondingly small proportion of hydrocarbons and tarry vapors. The older bituminous coals of the Appalachian region yield volatile matter containing large amounts of tarry vapors and hydrocarbons, difficult to burn completely without considerable excess of air and a high temperature. Coal of the western type, moreover, gives up its volatile matter more easily at moderate and low temperatures than that of the other type. The volatile matter produced at medium and low temperatures is rich in higher hydrocarbons of the methane type, such as ethane and propane, which contain a larger portion of carbon than is present in methane.

"These facts help to explain the difficulty of burning Pittsburg coal, for example, without smoke, the low efficiency usually obtained in burning high-volatile western coals, the advantage of a pre-heated auxiliary air supply introduced over a fuel bed, and the advantage of a furnace and boiler setting adapted to the type of fuel used. They bear directly also on the question of steaming 'capacity' of coal for locomotives, the designing and operation of gas producers for high-volatile fuels, and the operation of coke ovens and gas retorts.

"The results show further that certain bituminous coals of the interior and Rocky Mountain provinces give promise of good yields of by-products of coking, notably ammonia and high candle power gas, comparing favorably in these respects with the high grade coking coals of the eastern province.

"They show also that inert, non-combustible material is present in the volatile products of different kinds of coal to an extent ranging from 1 to 15 per cent of the coal."

The bulletin will be of interest to fuel engineers, designers and builders of gas producers, gas and coke manufacturers, superintendents of power plants, railway master mechanics and those engaged in the suppression of smoke. The bulletin may be obtained by applying to the Director of the Bureau of Mines, Washington, D. C.

The United States consul at Berlin, Germany, reports that fuller's earth is found in the English Jura, in the Belgian chalk formations, in Rosswein and Siebelem in Saxony, and along the Oder River in the Province of Silesia. Statistics giving the volume of the production in Germany are not available, but fuller's earth is extensively imported, especially from Florida and other parts of the United States. The volume of such import is, however, not separately given, it being included with "earths and raw minerals, not specially mentioned," and is entered duty free. Its principal uses in Germany are to dissolve fats and oils used in the soap and other industries, to scour and cleanse cloth, in the manufacture of colored paper and rugs, in the production of ultramarine, and also in the manufacture of an article to clean out spots.

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BARITE (BARYTES).

By SCHUYLER FRAZIER.*

OCCURRENCE AND METHODS OF PREPARATION.

Barite or heavy spar when pure is a white amorphous or crystalline mineral of a specific gravity of 4.35; chemical composition = BaSO_4 , barium sulphate. It is almost insoluble in water, soluble in strong sulphuric acid and has a melting point of 1580°C . This mineral is frequently found in the form of solid veins varying from a few inches to several feet in thickness, sometimes in contact with limestone and partly combined with it. Often barite occurs in connection with iron ore and other deposits, usually in the shape of irregular and rounded boulders of all sizes up to several hundred pounds in weight. Surface deposits of greater or less extent, of barite boulders occur frequently. Part of the mineral is intermingled with clay of yellow and red color and pieces of quartzite, etc., while the surface barite may have been weathered tolerably clean. Barite of this character is generally superficially stained red with iron oxide, and, to some extent, rather thick crusts of iron oxide adhere to the mineral.

The quality of barite varies greatly, in different deposits, of the nature just described, in respect to the fact that in some cases the adhering clay and iron oxide can be pretty thoroughly removed from the mineral by suitable washing processes, while in other cases the iron oxide adheres strongly to or penetrates the barite and can only be separated with difficulty and by special treatment. There are a few deposits of barite, particularly of the form found in a solid vein, in which the mineral may be found practically entirely free from impurities. Mines of this nature, if of sufficient extent, are of greater value in respect to eliminating the cost of purifying the mineral to make it marketable.

Barytes is used as an ingredient in the filler in manufacturing certain grades of paper and as a pigment; also for other purposes requiring a substance of high specific gravity and practically indestructible as to action of the atmosphere and most chemical compounds.

The trade demands that the mineral shall be ground as finely as possible and that for the better grades the color shall be very white.

To obtain the necessary fineness the mineral after grinding has to be floated in water or separated in

an air current because it is impossible to perform this operation to so delicate a degree by means of screens or bolting cloth. To obtain the necessary degree of whiteness any iron in the mineral must be removed by boiling in acid, also an ore must be selected of low silica contents for the reason that silica tends to produce a gray color. Silica is objectionable also because it is difficult to grind and causes rapid wearing of the mills. Besides this, silica having a lower specific gravity than barite will float larger particles in the separating process and cause a gritty texture in the fine floated barytes.

These considerations indicate the desirability of selecting an ore deposit of the best possible grade if it is expected to produce ground and floated barytes profitably.

Barytes is merchantable as tripple floated, viz., free from grit and uniformly fine as tested between the teeth and examined by microscope. As to color, usually three to five grades are recognized by comparison of samples wetted with turpentine and the price ranges accordingly between \$16 to \$8 per ton in barrels f. o. b. New York.

Reference may be had to various articles heretofore published in *THE CHEMICAL ENGINEER*, *The Engineering and Mining Journal*, *Oil Paint and Drug Reporter*, various bulletins of the United States Geological Survey, etc., all containing interesting information as to deposits of barytes, and manufacturing methods employed. The ore is found and mined largely in Missouri, Kentucky, Virginia, Tennessee and Georgia, and to some extent in other states. The deposits known are numerous and there are many yet to be discovered and prospected in the parts of the last three states where mineral developments have been retarded for lack of railroad facilities. Some of this territory is now being penetrated by new railroads and its varied mineral resources will repay careful investigators.

In many localities it is possible to purchase a mineral deposit from the land owners—largely uneducated white mountaineers—for a few hundred dollars, or to lease on a basis of 15 cts. to 25 cts. per gross ton royalty on mineral extracted. Including a 25 cts. royalty it is generally possible to mine and deliver barytes to the nearest railroad on a three-mile to five-mile haul at a cost not exceeding \$1.75 per ton.

Should the ore require washing to remove clay, etc.,

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it is advisable to provide a semi-portable apparatus for this purpose and break and wash the ore at the mine. The ore is usually somewhat soft and in breaking and washing considerable loss occurs amounting often to 30 per cent or more. The ore cost at the mine 75 cts. plus 10 cts. washing expense, equals 85 cts. per ton crude ore at 70 per cent recovery, equals \$1.22 net cost of one ton of clean ore at mine, or \$2.22 at railroad; to which add \$1.50 average freight to factory, located at some convenient point, equals \$3.72 f. o. b. factory. Compare these figures with crude ore \$1.75 at railroad, plus \$1.50 freight to factory, plus 10 cts. washing charge, equals \$3.35 per ton of crude ore with 30 per cent loss washing as before equals \$4.79 per ton cost of clean ore at factory, equals difference \$1.07 per ton clean ore.

Evidently it will pay to have the loss occur at the mine rather than at the factory. In case of poorer ores this fact will be more in evidence, and in some cases it may pay to jig the ore at the mine after crushing to $\frac{3}{8}$ -in. mesh so as to separate all iron and silica possible. There are locations in which it pays to locate the milling plant right at the mine, particularly where the deposit is of sufficient extent to keep the mills in operation ten years or longer and at the same time allow of culling out a fancy grade of ore for milling; that is to say, an ore sufficiently good to require little or no bleaching by acid treatment. Some locations of this nature permit of utilization of water power, either directly or by electrical transmission; and this combination of cheap power and good ore, together with taking the ore losses at the mine will result very profitably.

Experience has demonstrated that the plant required to handle barytes can be greatly simplified and cheapened if the mineral is of high quality and that if the barytes is not of the best and whitest grade originally it will pay better to use it, not for milling, but only to convert into chemical compounds.

For the latter purpose almost all deposits of barytes can be utilized because usually the washed ore will average 90 per cent BaSO_4 , and the impurities which are principally iron and silica do not interfere in the chemical process.

PLANT FOR GRINDING AND FLOATING BARYTES.

Formerly the grinding was performed by buhr mills or on a slip mill—the latter plant being as described, Vol. XI, No. 3, THE CHEMICAL ENGINEER, in article written by A. A. Steel, Professor of Mining, University of Arkansas. Of recent years the tube mill has come into service for this purpose, the first installation having been introduced by the writer four years ago in a large Virginia plant. In this instance, 12 sets of buhr mills were replaced by 3 tube mills, 16 ft. long, $4\frac{1}{2}$ ft. diameter, cylinder type, charged and discharged continuously through the hollow trunnions. The labor of 4 men at least was saved, including those dressing millstones.

These tube mills were operated by one man daytime and one man at night, with greatest ease, and these men also attended the floaters and dewatering tanks. The tube mills regularly produced 1,200 to 1,500 lbs. per hour each of tripple floated barytes and required no repairs excepting renewal of feed conveyor about four times yearly.

The barytes was fed into tube mills at $\frac{3}{8}$ mesh size, together with a regulated stream of water. The discharge of tube mills was lifted by bucket elevator, 12-in. by 6-in. buckets, to the floaters located above the tube mills. The tailings from the floaters returned into the feed of the mills by gravity. The floated material was delivered by centrifugal pump to the dewatering tanks. The overflow water from these tanks returned to the mills by gravity. A portion of this water was utilized in washing the crude ore, this being a means of preventing an accumulation of impurities and permitting the constant introduction of some fresh water into the circulation.

Preparation of the barytes for delivery to tube mills was accomplished by the usual combination of jaw crusher rolls, sizing trommel, also jigs for separating the barytes from iron and silica. The crude ore was fed into the jaw crusher, together with a stream of water which facilitated movement of the mineral through the machinery and washed out the bulk of the clay. The action of the jigs, on which a large volume of water was used, completely cleansed the ore so that the product fed into the tube mills was entirely free from clay and contained not over 25 per cent of the original impurities such as iron and silica.

The iron—largely pyrite—being of high specific gravity is difficult to separate from barytes by means of jigs, consequently a large loss of barytes occurs in the effort to get rid of the iron. Considerable iron and silica is so combined in the fragments of barytes as to be impossible of separation by jigging. It was found, when operating on these impure ores, to be economical to classify the ore at the mines, by hand picking, into two grades, of which the better grade was used only for milling and the poor grade reserved for conversion to chemical compounds.

This procedure permitted us to dispense with five jigs and thus save five-sixths of jig power and expense. One jig was retained in service merely as a very efficient washing machine for removal of clay from the crushed and screened mineral. The cost of hand selection of ore at the mines was found to be less than cost of operating jigs and the loss of mineral was reduced more than half.

The quantity of acid required for bleaching remained practically the same, which indicated the hand-picked ore to be equal in quality with the jigged ore.

After the barytes had been crushed, jigged and ground with water in tube mill, floated and dewatered as already described, the product was drawn from bottom of the dewatering tanks at a consistency of 80 lbs. barytes per 1 cu. ft. and carried by screw conveyor to

a set of four stock tanks each 10 ft. deep and 10 ft. diameter in which slow moving mechanical stirrers prevented settling. A set of six bleaching tanks was employed, these tanks being 15 ft. deep and 7 ft. diameter, having acid proof lining and provided with pipes delivering compressed air for agitation and live steam for heating. In these tanks the mineral was boiled with acid until bleached and then passed through a series of six washing tanks in which the acid was removed. The bleached and washed product was then dewatered to a consistency of 80 lbs. to 100 lbs. per cu. ft., and in this condition passed over mechanical dryers, thence through a disintegrator and packed in barrels. The type of dryer ordinarily used consists of a hollow steel cylinder rotating slowly on hollow trunnions and heated internally by steam at 30 lbs. pressure. The moist barytes of consistency of cream is distributed on the surface of the cylinder, becomes dry while passing three-fourths of a revolution, and is scraped off by a knife. These dryers are objectionable as to rusting causing discoloration of product, as to irregularities of feeding and drying causing excessive labor attending, and as to limited capacity. Evidently an improvement may be made by introduction of a large capacity dryer of type guarding against contamination of the barite by iron rust, also introduction of filter presses for removal of bulk of water before delivery of barite to the dryer.

An ordinary cage disintegrator is entirely satisfactory for pulverizing the dried barite. This should be located above the bins feeding the barrel packers.

PLANT FOR PRODUCING BARIUM COMPOUNDS

As stated above, sulphate of barium ores of inferior grade may be utilized for reduction in chemical processes, and for these purposes mines may be selected the ores from which would be of little value for purely milling purposes. It is advisable to break the ore to $\frac{3}{8}$ -in. size and pass it over one washing jig to get rid of all clay and to perform these operations at the mine so that the incidental losses may occur before the cost of ore has been increased by hauling and shipping charges. No purification from iron, etc., is necessary. Separation from limestone or other rock should be made in the mining operation just the same as in case of milling ore. The washed ore may be delivered directly to the factory and should be stored in bulk under large sheds in sufficient quantity to allow time for the ore to become practically dry, or else the ore should be run through a quick acting open flame dryer. Unless the ore is dry it is difficult to weigh and deliver mechanically to mills, because when damp the fine particles of the ore pack very badly in elevators and conveyors.

For a capacity of 30 tons daily of barytes to be reduced to barium sulphide the following plant is required:

- One 36-in. cage disintegrator to powder coal.
- One bin and bucket elevator for coal.

Two tube mills to grind barytes and coal together.

Two bins and one bud at elevator to feed tube mills.

Two hopper scales—one for barytes, one for coal.

One rotary cement kiln, 80 ft. by $7\frac{1}{2}$ ft., with fire box, feed and discharge conveyors.

The crushed dry barytes is dumped by tram car into hopper scale.

The disintegrated coal is run from bin into hopper scale.

Discharge slides of both hopper scales are opened slightly and both the barytes and coal allowed to flow together into the large capacity bucket elevator, delivering the mixture into either of the two bins located above tube mills.

Each tube mill is fed by a 4-in. fast running screw conveyor—300 r. p. m.—entering the hollow trunnion. The mixture of barytes and dry powdered coal flows from bin to hopper located over feed conveyor, and the hopper is provided with mill shaker regulating amount of feed. A regulated stream of water passes into the feed conveyor together with the mixture. The pulp discharged from mill is of consistency of thick cream and passes directly from the two tube mills to furnace by means of slow speed 8-in. spiral conveyor. The pulp is fed into the cement kiln by a specially designed feeder which is water jacketed as a protection against heat. The product from the tube mills is tested frequently and the fineness regulated so that 90 per cent will pass a 100-mesh sieve. The cement kiln is rotated one turn per 2 minutes. The fire box is of special construction for the purpose of maintaining a reducing flame and coal is used as fuel. One fireman attends the furnace. One engineer looks after a 10-hp. engine operating the kiln, a 60-hp. engine operating the mills, also attends oiling of all machinery. One mill man and one helper attend the tube mills. Four laborers handle the ore and coal. This represents the day force. At night no work is done except that one fireman is on duty to complete the roasting of material in the furnace, which is generally finished by 10 p. m., and to attend the engine and kiln which continually rotate and to keep the kiln hot during night. A kiln of this size should handle 50 to 60 tons of barytes per 24 hours if in constant operation, but close supervision of a competent chemist is necessary, so night work is avoided except when there is an extraordinary demand for the product. The amount of coal required for heating the furnace is about 30 per cent of the weight of barytes and a similar quantity of pulverized coal is needed to be mixed in the mineral fed into the kiln.

The product of the kiln should average not less than 60 per cent water soluble, 20 per cent acid soluble, 20 per cent insoluble, and with very close supervision has been obtained 70 per cent water soluble, 25 per cent acid soluble, 5 per cent insoluble. These results vary mostly according to the reducing quality of the flame and to some extent according to the nature and amount of impurities in the ore. High silica and

high iron tend to decrease the water soluble BaS and to increase the acid soluble BaSO_3 , BaCO_3 , BaSiO_3 . The red hot product from the kiln is delivered by a special conveyor directly into an automatic dissolving apparatus which removes quickly all the water soluble BaS and delivers the material insoluble in water into specially designed agitators in which the acid soluble substance is converted into barium chloride solution. The insoluble residue is frequently analyzed, and if low in barium sulphate content is wasted, but if occasionally high it is refurnaced.

The barium sulphide solution is filtered and stored in tanks of large capacity, and is used as the base for preparing various compounds such as blanc fixe, barium carbonate, lithopone, barium nitrate, barium hydroxide, barium peroxide, barium chloride.

The barium chloride solution is purified, filtered and concentrated to produce barium chloride crystal, and to some extent is converted into other products.

So far as known, the rutile (titanium oxide) deposits at Roseland, Va., are the largest in the world. The use of titanium in making steel rails increased considerably during 1909, and an American railroad which has given rails treated with ferrotitanium a long trial reports that they are proving entirely satisfactory. The use of titanium in arc-light electrodes is also growing. Of these, there are two principal types, one of which is an electrode made of finely ground titanium carbide, the other is composed of magnetite, chromium oxide, and rutile.

Autogenous methods of welding metals are extensively used in Germany, practically all of the larger manufacturers of metal wares, tubes, automobiles, bicycles, steam boilers, etc., having their own welding plants. Recently there has been a considerable increase in the number of small welding apparatus—that is, those having a carbide capacity of 2 kilos (4.4 lbs.) or less. These can be used without having the special license from the police authorities required for apparatus having a capacity of over this amount. There are comparatively few independent welding plants. The greater number of factories use acetylene for welding instead of hydrogen gas, those plants using acetylene having their own generating apparatus. German engineers claim that although the prices of acetylene and hydrogen are practically the same, it requires from three to five times as much hydrogen to do a given piece of work, and, furthermore, hydrogen is not suitable for welding pieces of more than 8 millimeters (0.315 in.) in thickness. The price of hydrogen is about 1 mark (23.8 cents) per cu. meter (1.3 cu. yds.), which is approximately what acetylene costs the producer having his own generating plant. The price of oxygen is about 2 marks (47.6 cts.) per cu. meter.

TECHNICAL GAS CALORIMETRY.*

By J. H. COSTE, F. I. C.

When a fuel containing unoxidised or partially oxidised carbon and hydrogen, in any form, is burnt under suitable conditions the carbon is burnt to carbon-dioxide and the hydrogen to water; it depends on the conditions under which combustion takes place whether the water vapor formed is condensed or not. If it is condensed the latent heat of vaporization of the whole of the water formed is included in the heating value of the fuel. When this happens the calorific value thus obtained is called the gross calorific value. It is only in such appliances as geysers that the heat given out from the water vapor in this way is utilized. In most cases the net calorific power, i. e., the heat evolved by combustion when all the water is kept in the form of vapor, is all that can be utilized in heating appliances. The difference between gross and net calorific power is, therefore, a measure of the total amount of hydrogen present in a fuel.

It is obviously convenient to state the calorific power of gases in units of heat per unit volume. On the Continent the cubic meter is usually used in conjunction with the (great) calorie. In this country (England) illuminating gas is bought and sold, under various acts governing the gas industry, by the cubic foot, and all the results of statutory testing are corrected to a cubic foot at 60° F. and 30 in. barometer.

The official testings of illuminating gas in London are returned in calories per cubic foot and the only parliamentary standard for calorific power of illuminating gas is expressed in this manner. In all figures given in this paper the calorific power is expressed according to this official usage. The number of calories multiplied by 3.968 gives British Thermal Units.

CALORIMETRY.

Any calorimetric method should give a means of determining, with such accuracy as may be possible, both gross and net calorific power. This object may be attained by either an indirect or a direct method.

The indirect method of determining calorific power.—The researches of Julius Thomsen and of Berthelot have placed at our disposal values for the heat of combustion of hydrocarbons and other combustible gases and vapor which are sufficiently reliable for most purposes. By means of these it is possible, if the composition of a gaseous mixture is known, to calculate its total calorific power.

In the case of coal gas and similar mixtures a difficulty arises in connection with the unsaturated hydrocarbons. It is difficult and in most cases impossible to decide their actual nature, and as the various hydrocarbons which may enter into their construction have very different calorific values the matter is of importance as will be seen from Table I (calculated from Julius Thomsen's figures) giving the calorific values

*From the Journal Society of Chemical Industry.

in great calories per cubic foot of various gases (A) measured at 0° C. and 760 mm., and (B) at 15.5° C. (60° F.) and 762 mm. (=30") the products in each case being cooled to 18° C.

TABLE I.

	(A)		(B)	
	Gross.	Net.	Gross.	Net.
Methane	268	240.5	254.3	228.2
Ethane	468	427.0	441.0	405.2
Ethylene	421	394	399.5	373
Propylene	622	581	590.2	551.3
Acetylene	392	378	371.9	358.7
Benzene	1,010	969	958.3	919.1
Hydrogen	86.4	72.8	82.0	69.1
Carbon monoxide	85.9	85.9	81.5	81.5

It will be noticed that the difference between gross and net calorific power for each molecule of water formed by the combustion of a molecule of the gas is in (A) 14 calories, and in (B) 13 calories per cubic foot. In other words the difference between gross and net calorific power multiplied by 100/14 or by 100/13, as the case may be, will give the total percentage of hydrogen in the gas.

The gas, the calorific power of which had been found by direct means to be gross, 150.3 calories, net, 133.2 calories, was found to have the composition indicated in Table II. The unsaturated hydrocarbons were assumed to be (1) ethylene, (2) propylene, (3) butylene.

TABLE II.

Composition.	Calorific power (calculated).		
	(1) gross.	(2) gross.	(3) gross.
Carbon dioxide... nil	..	..	..
Oxygen	..	..	..
Unsaturated hydrocarbons	5.38	21.52	31.74
Carbon monoxide	13.09	19.73	19.73
Hydrogen	42.45	34.85	34.85
Methane	27.71	70.43	70.43
Nitrogen	11.10	..	..
100.00	137.53 cal.	147.75 cal.	157.97 cal.
Per cent total hydrogen.....	108.63	111.01	119.39
Deduction for latent heat...	14.10 cal.	14.82 cal.	15.51 cal.
Net calorific power.....	123.43 cal.	132.95 cal.	142.46 cal.

It will be seen that the assumption that the unsaturated hydrocarbon is propylene gives results closely approaching those found experimentally. It is in fact found to be a good working convention for general purposes that the average calorific value of the unsaturated hydrocarbons is equal to that of propylene.

Table III serves to illustrate the agreement between results calculated in this way and those obtained directly.

TABLE III.

Experimental.		Calculated.	
Gross.	Net.	Gross.	Net.
166.6*	149.3	167.4	150.0
150.3	133.2	148.2	132.8
143.0	127.7	148.2	128.2
148.0	130.3	143.5	129.5
143.5	127.9	143.6	128.6
145.7	129.3	143.9	128.8
145.8	131.0	147.3	131.8
143.3	128.3	144.7	129.2
143.1	126.2	140.0	125.5
152.6	134.2	152.3	134.9
143.5*	127.7	142.5	128.5
145.7*	129.3	143.8	128.3
144.5*	129.5	146.3	130.5

*Pure coal gas.

Direct methods of determining Calorific Power.

Various forms of calorimeter have been devised for determining the calorific power of gases. They all depend on a known amount of water being heated to a measured extent by a known volume of gas.

Two principal types of calorimeter have been proposed (a) in which a small and usually a fixed amount of gas is allowed to burn out and to heat a small and fixed amount of still water; (b) in which gas burnt at a measured rate is allowed to heat to an observed temperature a constant stream of water, a definite amount of which is collected during the period of observation. In the former class the highest temperature of a rise—a transient equilibrium—is noted, and in the latter the temperature which is reached as the result of a continuous equilibrium. The flow calorimeters are to all intents "geysers" in which special care is taken to insure complete combustion of the gas and complete absorption of the heat evolved. In either case it is obviously desirable that the proportions of gas burned and water heated should be so adjusted that the rise of temperature may always be sufficiently great to insure reasonable sensitiveness of reading and not high enough to render loss by radiation a serious consideration. This optimum range of rise will vary in different instruments according to the efficiency of the measures taken to prevent radiation.

For the determination of the calorific value of a gas supply of any kind which can be burnt in practically unlimited quantities no doubt the flow type of calorimeter is the better, as the apparatus can be adjusted to remain in a state of thermal equilibrium for a considerable time, during which a relatively large volume of gas can be burnt and measured, and a correspondingly large volume of water heated and measured. Several readings of the temperature of the water at inlet and outlet can be taken and average temperatures computed for both inlet and outlet with a high degree of accuracy. The amount of water condensed in the abstraction of heat from the products of combustion can also be arranged to be sufficiently great to secure a good determination of net calorific power.

On the other hand the classical work of both Thomsen and Berthelot was carried out in calorimeters of the burn-out and fixed water type, and for technical purposes both Hempel's, and Simmance and Abady's portable calorimeters, which are of this type, give, if carefully adjusted and used, results of a fair degree of accuracy. Only the gross calorific power is determined in the two latter instruments.

Calorimeters of the still water type have been devised by Berthelot (bomb), Thomsen, Fischer, Hempel, Simmance and Abady, and of the flow type by Hartley, Junkers, Boys and others.

It is, perhaps, desirable to summarize the principal features of some of the more important of these instruments.

STILL WATER CALORIMETERS

Hempel's Calorimeter.—This instrument which has

been fully described in the Journal of the Society of Chemical Industry in 1901, p. 880, is adapted for the combustion of small quantities—2 to 3 litre—of gas. The instrument is calibrated by means of pure hydrogen. No correction is made for radiation, and no precautions are taken to guard against loss of heat from this or other causes. No doubt if the calorific power of a gas is not very different from that of hydrogen the results obtained are fairly accurate, but in the case of coal gas the loss of heat by radiation would be greater and with producer gases less than with hydrogen. In an example given the rise of temperature for a reservoir full of hydrogen was 10.838° , which seems to be a rather large rise to measure in an unjacketed vessel. This is essentially a laboratory, not a field apparatus.

Simmance and Abady's portable calorimeter.—In this instrument the gas is collected in a holder capable of containing considerable more gas than is required for the actual determination. The gas is burnt from a small open tube burner like a Bunsen without the side holes. The calorimeter itself has an outer casing of polished metal separated from it by an air space. It consists of (1) two parallel cylindrical chimneys of different diameters communicating by means of a drum-shaped vessel into which their upper ends open and (2) water jackets for each of the chimneys opening into a larger one surrounding the upper drum-shaped vessel. This water vessel is covered and has a neck through which a cork can be inserted either for the purpose of shaking or to carry a thermometer. A suitable stand holds the calorimeter at such a height that the burner can be slid under the broader chimney which serves as a combustion chamber, the narrower one being for the escape of the exit gases after passing up the wide chimney and communicating their heat to the water during their passage across the drum, which has thin metal spirals depending from its inner upper surface. These serve both as baffles and collectors of heat. About 1 litre of water is placed in the vessel, the quantity varying according to the heat capacity of the instrument. The calorimeter is well shaken and the temperature of the water taken. The holder having been filled with gas is arranged to deliver, and the gas is lit as it issues from the burner. When the water confining the gas has risen to a certain mark, indicating that the proper quantity of gas is now in the holder, the burner is slid under the chimney and this quantity—about 0.05 cubic foot—allowed to burn out. The rise in temperature is noted after shaking, and this, with the amount of water and the known constant of the instrument, gives the data necessary for the calculation of the gross calorific power. It is best to calibrate the instrument with gas of known calorific power—preferably approximating to that of the kind of gas for which the instrument is most likely to be used.

FLOW CALORIMETERS.

Flow calorimeters may be considered as of two

classes (1) those in which the passage of both water and gas through the apparatus is, by one or many paths, very short and rather of the nature of a parallel arrangement and (2) those in which the water flow is through a long tube around which the air slowly passes. The former class includes most of the flow calorimeters, Junkers' being both the best and the best known. Boys' instrument is the only representative of the latter.

Junkers' calorimeter is one of the earliest and best of its kind. A very good description with illustrations is given in this Society's Journal (1895, p. 631) by Hermann Kuhne. The calorimeter is portable, fairly convenient and accurate, the height of the instrument and distance apart of the thermometers are the most inconvenient features. An arrangement exists for determination of net calorific power by measuring the condensed water. The calorimeter is sealed and apparently would hardly be worth repairing in case of leakage. The water content of this calorimeter is about 1700 cc.

The method of using Junkers' calorimeter is somewhat similar to that described for Boys' calorimeter. See below.

Boys' calorimeter was designed by Mr. C. V. Boys, in his capacity as a Metropolitan Gas Referee, for the purpose of official testings in the County of London. It is described in the Proceedings of the Royal Society (A. Col. 77, 1906, p. 122), in the Notification of the Metropolitan Gas Referees and in this Journal of the Society of Chemical Industry (1906, p. 234; 1907, p. 355). The water content of the coil and equalizing box is only 300 cc. and of the space in the vessel used as a casing and to collect condensed water is only 400 cc. up to the overflow. Another feature peculiar to this instrument is the free passage of the products of combustion through the apparatus. Mr. Boys is of the opinion that "an inversion of the usual plan is essential to secure uniformity in the outlet temperature; that, in fact, the gases should have plenty of space to pass gently through the instrument and that the water should be taken through every channel strictly in series, all alternative parallel flow being prejudicial."

The instrument is convenient to manipulate, and it is not unduly subject to damage. The fact that it can be kept when not in use in an alkaline solution is a great advantage from the point of view of permanency.

It has stood the practical test of daily usage for nearly three years in each of the 19 gas testing places under the control of the London County Council. The most serious defect noticed has been leakage from the coils or unions. This can be tested for by placing the calorimeter in situ, allowing it to drain well, and then allowing water to flow at the usual rate for the same time as that during which the condensed water is collected. An appreciable collection of water indicates a leakage. It appears improbable that any other calorimeter at present in use would stand constant usage as well as this does when fairly treated. Cer-

tainly no other calorimeter is so easily taken to pieces for repair or for examination.

The official method of using the Boys' calorimeter is described in the Notification of the Gas Referees. It is briefly as follows: When a thermal balance has been established the temperatures of inlet and outlet water are taken at intervals during the combustion of 1/3 cu. ft. of gas and the water flowing through the instrument during this period is measured. The difference between the average inlet and outlet temperatures $\times 3$ and by the liters of water collected gives the gross calorific power. The condensed water is collected with the gas burning at the same rate as during the above experiment for not less than half an hour and from its volume and the gross calorific power the net calorific power is calculated. A correction is made for the gain or loss to the system due to the flow of gases through the apparatus, and based on the difference of temperature between inlet and exit air, the composition of the exit air and the specific heat of its constituents. The above method of working is applicable to the Junkers or other similar apparatus. It is better to allow $\frac{1}{2}$ to $\frac{3}{4}$ hour to elapse after starting before collecting condensed water as the surface of the coils is large and water only runs down quite slowly. It is necessary to place the calorimeter centrally in the outer vessel as otherwise the flow of condensed water is very irregular. It would be an improvement to have three centering studs fixed inside the outer vessel or on the calorimeter casing.

Source of error affecting the results obtained from flow calorimeters.—The following sources of error for the effects of which due allowance may be made are likely to affect the results of calorimeters of the Junkers' or Boys' type:

(1) Heat lost or gained in the exit gases according as they are hotter or cooler than the inlet air. (2) Heat lost or gained by condensation or evaporation of water during the passage of air through the calorimeter. (3) Losses by radiation from the body of the calorimeter. (4) Errors due to the assumption that 1 litre of water weighs one kilogram. The values of these errors and the proper assignment of corrections to gross or net calorific power have been worked out in the case of the Boys' instrument.

(1) The volume of exit gas compared with that of gas burnt has been determined by analysis of both gases. See Table IV.

The heat capacity of the products from 1 cu. ft. of gas, on the assumption that 7 ft. are obtained is about 0.06 cal. Experiments under varying conditions have led Boys to adopt the empirical correction of 1/6 cal. For a difference of 1° between inlet and outlet air, indicating that other influences are included in this correction which under any conditions is small.

(2) If the temperature of the exit gases is above the dew point of the atmosphere of the laboratory, the exit gases which are fully saturated, will rob the system of an amount of heat equal to the latent heat of vaporization of the additional water contained in them

and further the amount of condensed water will be reduced to a corresponding extent. On the other hand if the exit gases are cooler than the temperature at which dew is deposited in the laboratory the calorimeter will gain an amount of heat and of water corresponding to the reduced water-holding capacity of the air.

The extreme magnitude of this error can be illustrated by an example. Suppose the temperature of the laboratory to be 21° C. and the air fully saturated with aqueous vapor the temperature of the water supply being low, the exit gases having a temperature of only 10° C. If during the burning of 1 cu. ft. 7 ft. of these gases pass through the instrument corresponding to $7\frac{1}{2}$ ft. of air entering the total amount of water vapor entering will be 4.3 grms. (0.5 grms. in gas 3.8 grms. in air).

The total amount of water leaving will be 1.8 grms. and the excess water condensed $(4.3-1.8) = 2.5$ grms. The heat of vaporization of 1 gm. of water at 100° C. is 0.536 great calorie, and at 0° is variously given as from 0.589 to 0.606 so that in any case not more than $2.5 \times 0.6 = 1.5$ calorie per foot of gas burnt could be the amount of error due to this cause. The case of a low dew point and high outlet temperature may also be considered. It is seldom that the humidity of the atmosphere is less than $\frac{1}{3}$ of the maximum and the temperature of exit gases is seldom above 25° . Assuming both of these conditions and an air temperature of 21° the water vapor entering during the burning of 1 cu. ft. would be $0.51 \times 7.5 \times 0.51 = 3$ 1.78 grms. and that leaving the apparatus 4.55 grms., so that in this case the total loss would be 2.77 grms. per cubic foot = 1.66 calorie per cubic foot of gas burnt. That is, under quite extreme conditions the gross calorific power as determined without correction may be +1.6 calories from the truth owing to differences of humidity of the atmosphere. As the water condensed or evaporated gives up or takes away a corresponding amount of heat the net calorific power is unaffected.

The following table calculated from Regnault's values for water in 1 kilo of saturated air was used in the above calculations:

WEIGHT OF WATER IN 1 C. F. OF GAS SATURATED AT T° C.			
t.	Grms. water.	t.	Grms. water.
10	0.26	18	0.43
11	0.28	19	0.45
12	0.30	20	0.48
13	0.32	21	0.51
14	0.35	22	0.54
15	0.36	23	0.57
16	0.38	24	0.61
17	0.40	25	0.67

(3) It is difficult to conceive that an instrument so well designed as Boys' calorimeter should allow appreciable loss by radiation or conduction. The outer casing is bright nickel; between this and the casing of the coils is an air space. The outer coil has cold water, usually at a lower temperature than the surrounding air circulating through it. A brattice of sheet brass

lined with cork dust separates the outer coil from the inner hotter coil and the mixing chamber.

TABLE IV.

	I.	II.
Rate of burning, feet per hour.....	4.6	4.6
Carbon dioxide formed on explosion of gas, per cent.....	56.0	57.1
Composition of exit gas:		
Carbon dioxide.....	8.76	8.52
Oxygen.....	6.17	4.47
Ratio gas-air.....	1.69	1.72
Gas:exit gases.....	1:6.4	1:6.7

(4) The temperature of exit water varies between, say 24° and 40° C. Table V shows the percentage error caused by the supposition that one litre of water measured at various temperatures has a weight equal to 1000 grms.

TABLE V.

Temperature.	Percentage amount of error (+).
20°	0.177
25°	0.293
30°	0.433
35°	0.594
40°	0.776

That is, for gas of 150 calories gross calorific power the results, both gross and net would be too high by from 0.26 to 1.16 calories according to the temperature of exit water.

Junkers' and Boys' calorimeter may be considered as the two most in use for practical purposes and it may be well to discuss at some length their respective advantages and disadvantages and the necessity for applying certain corrections to the results obtained with them.

Although both instruments are fairly portable the Junkers' certainly has the advantage in this respect: Its simplicity and the fixed arrangement for obtaining a proper head of water render it particularly suitable for fitting up in any locality. The flow of water can easily be regulated by means of the quadrant tap and for gases of very varying calorific power the use of a Bunsen flame enables perfect combustion to be attained at any reasonable rate of consumption.

The rise of temperature attained by the exit water can be kept constant by altering the amount of water passing through the instrument. The Boys' instrument is easier to read owing to both thermometers being on the same level. It is more sensitive to rough handling.

The temperature of the exit water is more constant in the Boys' than the Junkers' instrument as examples in Table VI will show.

There appears to be no reason to suppose that the average of several readings does not in either case accurately represent the average temperature of outlet water.

The agreement of the results obtained by the simultaneous working of two Boys' calorimeters with the same gas is shown in Table VII.

A similar series has been carried out with the Boys'

and Junkers' instruments, and the results are shown in Table VIII.

TABLE VI.

Junkers.		Boys.	
Inlet.	Exit.	Inlet.	Exit.
(A)	(B)	(A)	(B)
16.93	16.99	27.13	37.88
		.09	.89
		.10	.82
		.11	.86
16.92	16.99	.11	.86
		.05	.83
		.13	.78
		.08	.78
16.93	16.99	.10	.85
		.08	.86
		.08	.82
	16.99	.08	.82
		.80	.80
		.82	.82
Average		37.83	42.25
Average variation		0.018	0.027
		..	0.014

The behavior of both instruments with gas burning at different rates, the water flow being maintained at a constant rate, is satisfactory as shown in the following series of experiments; Table IX:

TABLE VII.

	No. 1.		No. 26.	
	Gross.	Net.	Gross.	Net.
I	146.0	129.4	146.0	129.5
II	145.8	129.0	145.0	129.2
III	146.7	126.9	147.0	128.5
IV	147.9	130.9	145.9	129.9
V	146.1	129.0	146.3	129.9
VI	146.8	130.1	145.8*	130.2

*Gas burning at 2½ feet per hour. All others at 4-5 feet.

It has been suggested that combustion is not perfect in the Boys' instrument owing to the use of flat flames instead of atmospheric burners. To test this point a tube was inserted in one of the holes in the top of the calorimeter and the exit gases slowly aspirated (1)

TABLE VIII.

	Boys' No. 26.		Junkers.	
	Gross.	Net.	Gross.	Net.
I	144.9	130.1	145.8	130.8
II	145.1	130.2	146.1	131.4
III	147.3	132.9	146.7	132.0
IV	145.9	131.0	144.4	130.9
V	144.6	130.6	145.0	130.2
VI	143.8	129.3	144.7	130.1
VII	143.9	139.8	144.6	129.8
Average	145.1	130.6	145.3	130.7

over pumice saturated with caustic potash solution (2) sulphuric acid (3) a weighed soda lime tube (4) a column of about 12 ins. of heated copper oxide (5) a weighed sulphuric acid tube (6) a weighed soda-

TABLE IX.

Consumption of gas in ft. per hour.	Calorific power.			
	Boys'.		Junkers.	
	Gross.	Net.	Gross.	Net.
5	146.3	130.9	146.0	131.6
4	145.1	130.1	145.1	130.8
3	146.5	130.4		
2	144.2	128.7	146.8	131.9

lime tube. 8 litres of carbon-dioxide-free products corresponding to nearly 9 litres of exit gases were collected, tubes (3) and (6) had not altered in weight

and tube (5) had gained 0.008 gm. (= 0.0009 gm. hydrogen); any unburnt carbon should have been oxidized by the copper oxide and weighed as carbon dioxide in (6). The gas was burning in the calorimeter at 5.3 ft. per hour—above the maximum rate allowed in official testings. The error due to imperfect combustion is therefore negligible.

Both instruments may be expected to give the best results when working under the conditions which should prevail in a physical laboratory and both give very good results under conditions much less satisfactory.

THE CALORIFIC POWER OF GAS CONSIDERED IN RELATION
TO METHODS OF MANUFACTURE AND TO
ILLUMINATING POWER.

The calorific power of gas depends on its composition. The relation between calorific power and composition is, however, much more direct than in the case of illuminating power. It is, as has been shown, easy to calculate with considerable accuracy the calorific power from the results of analysis of a gas. In the case of illuminating power it is at present impossible. A consideration of the calorific power of the three principal constituents of coal gas: Carbon monoxide, 81.5 gross; 81.5 net; hydrogen, 82.0 gross; 69.1 net; methane, 254.3 gross; 228.2 net, indicates very clearly that a gas which has been made under conditions favoring the formation or survival of a large proportion of methane should have a higher calorific power than one where the temperature of carbonization tends to the production of a large amount of hydrogen and the use of an exhaust leads to the formation of large quantities of carbon monoxide. "Blue" water gas lowers the calorific power of coal gas with which it is mixed for similar reasons. The effect of the addition of carburetted water gas varies according to its composition. Its general effect is to lower the calorific power or at any rate not to increase it even in cases where its use has the effect of increasing the illuminating power. In the case of a pure coal gas more than half of the net calorific power appears to be due to methane whereas in mixed gases the methane seldom accounts for more than 46 per cent, the usually somewhat increased amount of unsaturated hydrocarbons more or less making up the deficit due to the lower proportion of methane.

It was formerly believed that some direct relationship between calorific and illuminating power existed, but an examination of thousands of tests shows that, although the general tendency is for gas of a high illuminating power to have a relatively high calorific power, such numerous exceptions and anomalies are observed, even with pure coal gas that no formula usefully connecting these two properties can be found.

Table X shows what a wide divergence occurs between observed calorific power and the figure obtained by multiplying individually results of illuminating power by the average number of calories per candle. The results at five places supplied with pure coal gas

were examined: the figures represent averages of daily testings for two succeeding months:

TABLE X.

	(A.) Illumi- nating power	Average.		(C.) Calories per candle	Greatest differ- ence between observed calor- ific power and product of ob- served illumi- nating power and the figure in column (C). Calories. Per cent.	
		(B.) Gross calorific power.	(C.) Calories per candle			
			A	B		
I	16.88	151.0	8.95	8.9	8.9	5.9
	16.97	151.3	8.67	14.4	14.4	9.5
II	17.11	150.6	8.87	10.1	10.1	7.2
	17.03	150.2	8.85	9.7	9.7	6.4
III	16.77	146.4	8.83	4.5	4.5	3.1
	17.07	146.7	8.60	9.8	9.8	6.7
IV	15.75	151.6	9.62	5.2	5.2	3.5
	16.01	152.7	9.57	6.7	6.7	4.4
V	16.36	148.3	9.04	8.4	8.4	5.8
	17.17	150.4	8.76	5.2	5.2	3.4

Although the average ratio from month to month agrees fairly well, showing that considerable regularity of manufacture is attained, it is clearly impossible to assign to gas of a known candle power a definite calorific power on mere a priori grounds. Standards can only be fixed as the results and no means have been devised for determining the calorific power of a gas except by analysis or calorimetry. It can be safely stated that for the same candle power, "pure" coal gas will in most cases have a higher calorific power than a mixed gas. This is, perhaps, more apparent in the case of low grade gas. The average calorific power of coal gas of from 14 to 15 candles illuminating power is about 130 calories net whilst mixed gas of the same illuminating power has an average calorific power (net) of about 125 calories, 16 candle coal gas on the other hand has an average calorific power of not more than two or three calories above 16 candles mixed gas.

The difference between gross and net calorific power is greater as the calorific power increases. An examination of three sets of results, two (A) and (C) mixed gas of 16 and 14 candles nominal illuminating power respectively and (B) pure coal gas, show that the difference is proportionately greater in the case of coal gas than for mixed gases. These figures agree with the results for total hydrogen in the gas, deduced from analysis. As to the actual differences between gross and net calorific power it appears unlikely that this should ever exceed 20 calories in the richest coal gas or fall below 12 calories in the poorest mixed gas now supplied.

TABLE XI. DIFFERENCE BETWEEN GROSS AND NET CALORIFIC POWER EXPRESSED AS PERCENTAGE OF GROSS.

Periods of 4 weeks (daily testings).	A.	B.	C.
I	10.64	11.25	10.92
II	10.80	11.46	10.87
Averages	10.72	11.35	10.90

In the following table (XII) the composition, calorific power and illuminating power of (A) coal gas made in a small works using clay retorts but no exhaust, (B) coal gas supplied by one of the Metro-

politan companies, (C) mixed gas supplied in the Metropolis, and (D) gas from Glover-West vertical retorts analyzed by Dr. H. G. Colman, are shown. The calculated calorific power for (D) was 142.6 gross and 126.9 net. It will be noticed that although the 21 candle gas (A) shows a marked superiority in heating value the nominal 14 to 16 candle gases do not, with one exception, which in composition closely approaches the 21 candle gas, differ greatly in calorific power among themselves.

In conclusion, it is most desirable that gas authorities should follow the example which has recently been set in the Metropolis by instituting regular testings for calorific power in order that a standard which will meet local conditions may be fairly settled when a suitable occasion arises.

I wish to thank Mr. B. R. James for valuable assistance in connection with this paper.

DISCUSSION.

Prof. C. V. Boys said that when calorimetry as a practical question in relation to official testing first arose, he made a great many tests with the Junkers and other calorimeters. He considered that the spas-

He thought he was justified in that conclusion because his own outlet thermometer was free from these jumps, appearing quite constant sometimes for minutes together almost as if it were stuck; and he had not seen that in any other calorimeter. There seemed to be an idea that the real scientific test of the value of gas was the net test; but he was by no means sure that this was a very scientific test, for this reason: In taking a net test, the heat due to the condensation of most of the steam and the cooling of the resulting condensed water to the outlet temperature, was subtracted from the total observed heat, but this did not represent any actual use of the heating power, as the heat given out by the permanent gases in cooling from the boiling point to the outlet temperature was included in the net value. The figure so obtained was absolutely artificial; it meant nothing. As the Gas Referees had prescribed a net test his position might seem inconsistent, but calorimetry was practiced largely, and the gross and net values were in universal use long before there was any question of making official testings of the calorific value of gas, and it was obviously desirable in instituting an official test not to depart from recognized practice. There

TABLE XII.

	A.					B.					C.					D.				
Carbon dioxide	0.38	nil	nil	1.48	1.75	2.11	2.85	1.45	1.74	2.13	5.00	1.51	3.83	1.00						
Oxygen	nil	nil	nil	0.26	0.52	1.05	0.20	0.73	0.37	0.28	nil	0.53	0.29	0.05						
Unsaturated hydrocarbons	6.17	5.57	6.00	3.39	1.73	2.98	3.60	1.91	1.59	5.39	5.50	5.07	5.45	2.85						
Carbon monoxide	4.86	6.03	9.24	6.17	7.18	10.00	7.43	14.46	13.22	16.15	16.00	13.31	15.02	8.70						
Hydrogen	41.72	39.83	40.00	49.21	40.50	48.41	46.66	46.73	50.76	39.24	39.30	43.62	42.10	54.70						
Methane	40.67	41.78	40.48	31.22	39.50	30.80	31.54	24.57	26.08	24.79	24.00	26.17	23.45	29.05						
Nitrogen	6.20	6.79	4.28	6.27	5.82	4.65	7.66	7.15	3.24	12.02	10.20	6.79	9.86	3.20						
																				99.55
Gross calorific power	175.8	179.6	176.0	144.5	166.6	145.7	143.4	144.8	145.1	145.2	141.4	143.5	143.3	144.3						
Net calorific power	160.5	161.1	157.5	129.5	149.3	129.3	128.9	129.2	129.1	128.5	124.5	127.9	126.3	129.6						
Illuminating power by Metropolitan Argand No. 2 at 5 feet per hour	21.39	21.77	21.10	18.60	18.21	14.48	14.65	16.65	16.58	16.21	15.80	17.29	16.81	15.56						

modic variation of the thermometer at the outlet was objectionable; there were constant little jumps which would make it tiresome for gas examiners to use it. In making tests day by day it was desirable that as much as possible should be done by the apparatus, and that as little as possible should be left for the daily attention and judgment of the user of the instrument. It seemed essential that the outlet thermometer readings should be as smooth and even as that of the inlet water; so that one would merely have to make readings at definite times, and would not have to judge what the fair mean reading would be. The imperfect mixing of parallel flows of water through alternative channels would almost certainly give rise to jumps which no mixing-chamber, such as was at the top of the Junkers' instrument, could possibly overcome. There was an instability of conditions which a small mixing-chamber could not possibly overcome. He felt that the difficulty could be overcome only by arranging that the water should take every part of the instrument in series; and that there should be no place where a pocket of water could accumulate until it got hot enough to force its own way on and overwhelm the other stream.

was no reason for selecting one particular net value, even if it did correctly represent the use of gas; every user of gas should be equally entitled to a special net value to meet his requirements.

He thought it right to describe the calorific value by the use of calories rather than by B. T. U. It was more easy to obtain appliances, which should be cheap and accurate, based upon the metric rather than upon our system, and having so made the measurement it was most natural and direct so to state it. With regard to the centering of the instrument inside the case, there would be no difficulty in providing that the inner casing should not bear up against the condensed water outlet.

Mr. J. W. Helps agreed with the author as to the impossibility of being able to deduce the calorific power from its illuminating power of a gas, and vice versa. He carried out some experiments on that point 10 years ago; and the figures now shown agreed closely with those he had obtained. He did not see the necessity for the two tests; the calorific test would be sufficient; the other was really of no practical use. Whatever burner was used for testing illuminating power, the extent to which a reduction could be made

was limited by the heating power of the gas; and as it was possible to vary the conditions under which the gas was produced, so as to increase its calorific power while actually reducing its illuminating power, the infliction of a test for the latter stood in the way of the customer being supplied with the most suitable gas for his various purposes.

That was an important question which would have to be considered and, in his opinion, it would not be long before they had the calorific standard.

Mr. W. J. Dibdin said that there could be no question but that Professor Boys had devised a thoroughly trustworthy calorimeter, by means of which the calorific value of coal gas could be readily ascertained. The only way of getting a result with the old form of calorimeter was to take a number of observations, expunge any out-of-the-way ones, and take the average of the rest, which was not a very scientific method. This instrument certainly made a great advance. He looked on the ordinary gas photometer as now used as a thing of comparatively little value. He hoped, with Mr. Helps, that the time was not far distant when the calorific test would be the one reliable test of coal gas, so far as it was in use for lighting. It was unnecessary to enter into any discussion of the merits of water-gas; but it was a question to what temperature an incandescent mantle must be raised to produce sufficient light, and what quantity of gas was required to raise a definite quantity of water to a certain temperature for cooking.

Dr. H. G. Colmans said that the figures given by Mr. Coste agreed closely with those he had obtained during the past 10 years, but he differed from him with regard to the table showing the relation between the results obtained by direct test with the calorimeter, and those calculated from the analysis of the gas. The calorimeter tests were always made with a wet meter, and the results calculated for moist gas 60° F. and 30 inches Bar., under which circumstances they contained about 1.5 per cent of moisture; the calculated results were made on the assumption of a dry gas, and if the moisture had been allowed for the calculated results would have come out about 1.5 per cent lower than the observed figures. For ordinary purposes it was sufficient to assume that the unsaturated hydrocarbons consisted of propylene, but in the newer methods of manufacture, especially in vertical retorts, the unsaturated hydrocarbons contained a smaller proportion of benzene and higher proportion of ethylene. He trusted that that day would soon come when a calorific standard would be substituted for one of illuminating power, and hoped that Mr. Coste's paper would help that day forward. He fully agreed with Mr. Helps, however, that the imposition of a double standard was much to be deprecated, and believed that in actual working it would soon be found impracticable. He had made thousands of tests with the Junkers calorimeter and had found it an excellent instrument. There were slight jumps sometimes in the reading of

the outlet thermometer, but these jumps were only fractions of a degree, and in the ordinary run of a test there was practical agreement between the two instruments.

The chief objection to the Junkers calorimeter was the different level of the two thermometers, which necessitated the jumping up and down of the operator during the test.

Mr. G. C. Jones wished to defend one class of manufacturers against the charge of not being able or willing to state the thermal efficiency of the goods they sold. Reputable gas-engine makers would always guarantee the efficiency of their engines. It was true they preferred to speak of so many feet of gas per H. P. hour, and natural that they showed an increasing preference for Manchester gas as the standard, but, if pressed, they would usually guarantee one H. P. hour for so many calories supplied to their engine in the form of gas. It would nearly always be found, however, that they refused to be debited with more than the so-called "net" calories. The general adoption of this term could be traced to certain discussions concerning the efficiencies of gas producers and gas engines which took place ten or fifteen years ago, when its use led to much confusion and some protests. Its adoption improved the paper efficiency of a gas engine, but that was hardly a reason for retaining in their vocabulary a word which had been characterized by Prof. Boys as unscientific. If the discussion led to the disuse of that ambiguous term, Mr. Coste's paper would be remembered with gratitude by many of them.

Mr. C. J. D. Gair said he had found the Simmance-Abady calorimeter gave excellent results. With the Junkers calorimeter there was a tendency for the outlet temperature to jump in working; but, with the Simmance-Abady this did not occur, neither was there the difficulty in the flow of the condensed water which in the Junkers was somewhat irregular, and, unless several cu. ft. of gas were burned and the products collected, a proper reading was not insured. The snap test could not be taken accurately with a Junkers calorimeter, but this was possible with the Simmance-Abady. It seemed an excellent suggestion to take propylene as the most correct hydrocarbon from which to calculate, but undoubtedly the calorific test itself was greatly to be preferred in obtaining the heating value of the gas.

Mr. Coste, in reply, said he was very pleased to hear Prof. Boys appreciated the point about the centering of the instrument. It was not made as a merely academical suggestion; but some little time ago they found that the differences between the "gross" and the "net" calorific power in certain results were such as they knew to be quite impossible, and a careful examination of the instrument showed that by placing the calorimeter eccentrically in the outer casing, similar erroneous results were obtained, while, by carefully centering it, the results at once became normal.

THE DOHERTY GAS CALORIMETER.

The accurate determination of the heating value of artificial gas is one of the important problems in the gas business of today. A number of devices for testing this heating value of gas have been placed on the market, the latest of these being the invention of Mr. Henry L. Doherty.

A British thermal unit (B. T. U.), it will be remembered, is the amount of heat required to raise the temperature of 1 lb. of water, 1° F., under standard conditions of pressure and temperature. The heating value of a gas, therefore, is known if the rise in temperature it produces in a unit weight or unit volume of water is known. This is the principle of all gas calorimeters.

The fundamental principle of the Doherty calorimeter is the direct displacement of the gas under test by the water which that gas heats, thus maintaining absolutely constant the ratio of volumes. The pressure on the gas is imparted to it by the displacing water; and this water secures its constant pressure from a constant static "head" in the regulator of the calorimeter.

Every cubic inch of water passing the flame is heated by the burning of a cubic inch of gas displaced by it. This being the case, determination of the heating value in B. T. U.'s becomes simply a question of measuring temperatures, with suitable corrections for existing conditions bearing upon the problem.

The Doherty gas calorimeter has two essential parts—the absorption chamber and the tank. Both of these are cylinders of annular section, the former being placed within the latter. A heavy insulating layer of non-conducting felt guards against interchange of heat through the walls. The Bunsen flame burns within the absorption chamber. The entire arrangement is thus compact and self-contained, and of extreme simplicity.

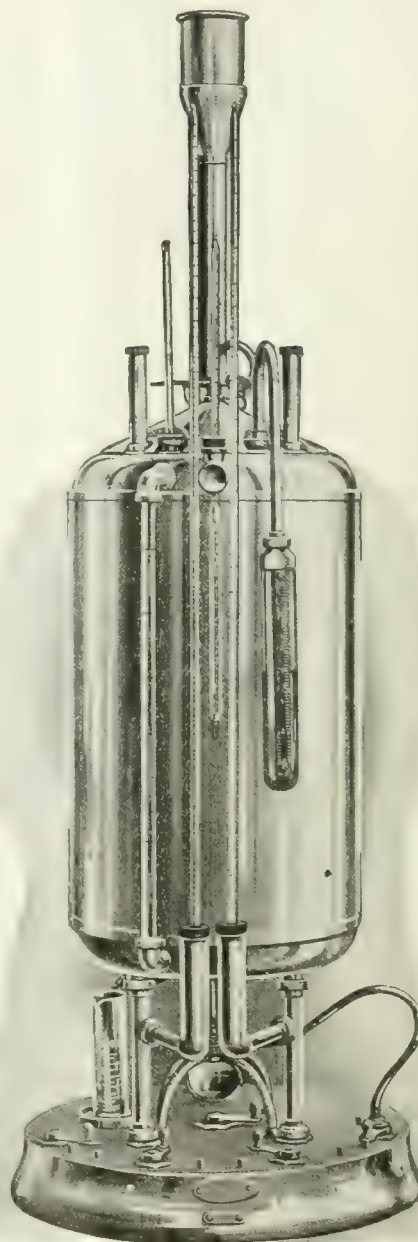
The regulator is simply a standpipe, in which a constant head or pressure is maintained by means of a supply of flowing water, part of which escapes to the drain, the remainder keeping the standpipe full as displacement takes place in the tank.

The tank has a gage glass for indicating the water level inside, and a U-gage showing the pressure on the gas (and water) in inches of water. The gage glass is divided into nine equal divisions, the volume between the extremes being exactly one-third of a cubic foot. In operation a pressure of about 2.6 ins. is maintained on the U-gage by suitable valves.

The water passes from the standpipe, through the absorption chamber, and into the tank. An arrangement of adjustable baffles within the absorption chamber makes it possible to adjust conditions so that the products of combustion escape to atmosphere at atmospheric or room temperatures. Five thermometers are used, accurately graduated, the two principal ones (those measuring the temperature of inlet and outlet water) are tested, calibrated and corrected by the

U. S. Bureau of Standards, a correction sheet being furnished for each. The five thermometers register the following temperatures: Water entering absorption chamber; water leaving absorption chamber; gas in tank; products of combustion; atmospheric or room temperature.

In making a test the tank is filled with the gas to be tested and the supply shut off. Water is admit-



The Doherty Gas Calorimeter.

ted from the standpipe, imparting regulator pressure to the gas. Pressure is adjusted to the desired two and six-tenths inches on the U-gage by the adjusting valve. Temperatures on all thermometers are noted. The Bunsen is lighted, when displacement at once begins. The exposed area of the baffles in the absorption chamber is adjusted until the waste gases emerge into atmosphere at room temperature. Meantime the water level in the gage glass will have risen to near the first graduation. Sufficient space is allowed below the graduations to permit these preliminary adjust-

ments before test readings are taken. With water level at the first mark on the gage glass, all temperatures are noted; and this is repeated as each mark is reached, giving ten readings at equal intervals. The pressure in the U-gage of course remains constant throughout. The average of the ten readings is taken as the basis of computation on that particular test.

The method of arriving at a result is as follows, the known quantities being as indicated by the symbols here given:

V = volume of gas corrected to 60° F. and 30 ins. mercury.

h = barometric pressure in inches of mercury.

h' = U-gage pressure in inches of mercury.

a = vapor tension pressure in inches of mercury.

t = temperature of gas in tank.

d = difference in temperature of water before and after heating.

$$V \frac{17.04 [(h + h') - a]}{400 t} \times 62.5 \div d$$

B. T. U.'s of gas

The calorimeter itself affords all quantities except "h" and "a." The value of "h" at time of test can be secured from a barometer at hand, or from some local barometric station of known reliability. The value of "a" is to be determined from tables furnished with the calorimeter showing the vapor tension at temperature of gas in tank. The value "h'" is 1-13 or .0769 times the U-gage reading in inches of water.

Making a test with the Doherty calorimeter, therefore, consists simply in reading the temperatures of inlet and outlet water, multiplying the difference between their two temperatures by 62.5, and multiplying this result, in turn by the value of $1/V$, which is secured from tables furnished with the instrument taking account of the other known conditions of the test. This process gives the gross heating effect, or calorific value, of the gas under test. It does not, however, take account of the latent heat represented by the water of condensation produced in burning the gas. To provide for this, and to determine the net heating effect of the gas, the following method is used:

A drip is provided around the lower end of the absorption chamber, which collects the water of condensation, discharging into a glass graduate, marked both in C. C. and in B. T. U. A test is started, and when the water level is at the lowest of the ten marks on the gage glass, the amount of condensation is noted in the graduate. The distance between upper and lower marks on the gage glass represents exactly $\frac{1}{3}$ cu. ft. When water level has reached the top mark on the gage glass, the increased amount of water in

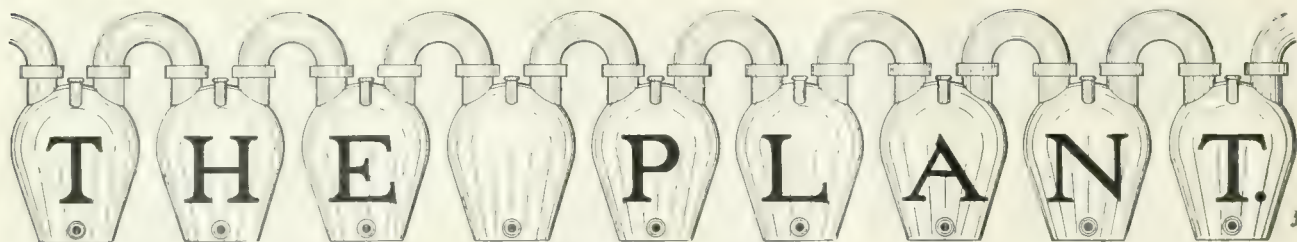
the graduate, read in B. T. U., shows the amount of heat represented by this water of condensation. Multiplied by three (for one-third of a cubic foot of gas was burned), this gives the B. T. U. value of the water of condensation per cubic foot of gas tested. Subtracted from the gross result secured by temperature readings, the remainder is the net heating effect, or net calorific value, per cubic foot of the gas under test.

The Doherty gas calorimeter is manufactured by The Improved Equipment Co., 60 Wall St., New York, to whom we are indebted for the information from which this article was prepared.

Advance chapters of United States Geological Survey publications state that although the figures from all the countries producing large quantities of tungsten are not yet available, enough are at hand to show that the world's production for 1909 was larger than in 1908, and was possibly equal to that of 1907. The estimated output for 1909 was 5,289 short tons of concentrates containing 60 per cent of tungsten trioxide, compared with 3,898 tons in 1908 and 6,062 tons in 1907. In very few of the returns, however, is the percentage indicated, and this is necessary in comparing figures. It is probable that most of the exported ore is richer than 60 per cent in tungsten trioxide, and that the figures are therefore low. The principal sources of the 1909 production were: United States, 1,619 tons; Argentina, 900 tons; Australia, 1,200 tons; Bolivia, 168 tons; Germany, 106 tons; Portugal, 609 tons; and the United Kingdom, 421 tons. The value of the tungsten ore produced in the United States during 1909 was \$614,370, compared with \$229,955 in 1908 and \$890,048 in 1907. With the recovery of the American steel trade during 1909 prices rose above those of 1908 and ranged from \$5 to \$9 per unit for tungsten ores, the average price being \$6 to \$6.50 per unit.

It is announced that the Bundesrat has considerably increased the quantity of potash which can be produced by German potash works during the period from May 1 to December 31, 1911. The amounts in metric tons of 2,204.6 lbs. pure potash (K_2O) which can be sold without being subjected to the special tax are now fixed according to a consular report as follows, with comparison of amounts as determined in June, 1910:

Classification	For home consumption		For export	
	Old Allowance Tons	New Allowance Tons	Old Allowance Tons	New Allowance Tons
Chloride with 9 to 11 per cent K_2O	600	2,770	11	0.0
Crude salts with 12 to 15 per cent K_2O	126,000	136,770	29,600	31,700
Salts for fertilizer with 20 to 22 per cent K_2O	8	78	11,000	11,800
Salts for fertilizer with 30 to 32 per cent K_2O	19	1,270	9,000	100
Salts for fertilizers with 40 to 42 per cent K_2O , including potash fertilizer with 38 per cent K_2O	37,400	11,100	11,700	11,500
Chloride of potash	1,750	10,000	33,600	11,300
Sulphate of potash with over 42 per cent K_2O	1,600	1,000	32,700	28,500
Potassium magnesium sulphate	100	1,000	1,600	1,000
Total	901,800	199,800	108,100	53,000



THE POWER PLANT OF THE BERGEN-PORT CHEMICAL WORKS.

By WARREN H. MILLER.*

The use of alternating current for light and power in the chemical works is apt to be regarded with many misgivings by the works engineer or superintendent, because of the alleged complicity in the power station and complexity of wiring computations. As a matter of fact both are very simple, and the beauty of it is that the plant will run and stay running with practically very little supervision and repairs. The acid chamber plant of the Bergenport Chemical Works, a subsidiary company of the Standard Oil Co., turning out about 300 tons of sulphuric acid daily on the chamber process, is an illustration of how very self-running and fool-proof such a plant can be. At night it is general headquarters for the works foreman whose duties take him all over the plant; by day the works electrician has his tool closet there, but is often out around the plant.

Yet, day after day, year after year, the plant runs along with no commutator troubles to worry about (for there isn't any commutator, nor brushes), with splash-oiled engines, and with no attention beyond an occasional glance at the instruments by the boiler-room foreman, who comes in at intervals to attend to the boiler feed-pump.

In laying out this plant we had five still houses, 200-hp. of air compressors, and 200-hp. process pumps to take care of, besides the electric light and power, amounting to about 75 kw. Estimating the five still houses roughly at 250 hp., this gave a total horsepower of 750 or four 250 hp. boilers, allowing one spare boiler. The type of boiler house, while by no means of architectural beauty, is of interest in showing what can be done with hollow tile enclosing steel-work, in getting up a light, fire-proof, acid-proof structure which can be erected on isolated piers in a soil not suitable for wall foundations. The roof is red "celadon" tile, its trusses and purlins being virtually all the steel that is exposed to the acid fumes. The site of the power establishment was a sandy substratum, covered 4 to 6 ft. deep with cinder and occasional layers of acid-soaked refuse of ash and nitre-cake. For both boiler house and power house concrete piers were put down, joined by reinforced concrete sills, 12 ins. wide by 18 ins. deep, running from pier to pier. They were cheaply made by leaving bearing notches in each pier and simply excavat-

ing a trench from pier to pier with a good shear at the pier junctions. These were filled with concrete and the reinforcing bars. Our experience on other jobs was that acid ash readily attacked rock concrete but did little harm to cinder concrete, so the mixture used was 1 : 2½ : 4 cinder concrete.

The next step was to set up and plumb all steel work, which was then braced and riveted, except the purlins, which were bolted to allow taking off for periodic tar paintings. As the steel is all enclosed in tile later, there is no place for diagonals to hold the steel work plumb and true, so we found that false wooden diagonals of 6-in. by 4-in. hemlock would answer very well and could be taken out as the tile work progressed.

The standard tile is an 8-in. by 6-in. by 16-in. red semi-vitrified terra cotta webbed tile laying in the wall 8 ins. by 16 ins. They have 8¼-in. by 6¼-in. by 16¼-in. actual measurement, and take about a ⅜-in. mortar joint. They weigh 32 lbs. each, cost 10 cts., and in the wall five of them lay up 42½ ins. high by 82½ ins. long. They form a light panel or curtain wall, grooving into tile pilasters which enclose the steel. The steel columns are plain 10-in. I-beams, and are completely enclosed by two shapes of pilaster blocks, which break joints in successive courses, provide a 16-in. pilaster, and also a 7-in. groove on each side to receive the ends of the curtain wall panels. They cost 11⅓ cts. and 14⅓ cts. each. For corner pilasters intended for tile gable ends with no steel work there are four forms, so as to break joints and provide 7-in. wall grooves on right angle faces, and there is also a set of T pilaster blocks for two meeting walls.

We found that one or two of the regular works burner masons, a few handy men and the works laboring gang made a force that would lay tile better and with far less grumbling than hired brick masons. The latter were full of inrooted prejudices against such makeshift stuff as hollow tile, and growled continually at the weight, the shape, the mortar and the job in general.

For lintels, sills and copings reinforced 1 : 3 : 4 gravel concrete, smooth cast with a raised keystone, contrasts effectively with the red tile, though flat arches can be cobbled out by cutting up standard tile. For cornices and gable copings the ordinary standard panel tile can be laid flat on the 8-in. side, thus giving a 2-in. projection. Always run the roof over the gable, as it is very difficult and expensive to flash it properly if butted up against an end wall.

The red celadon tile for the roof cost 11 cts. each,

*Bryn Hesel, N. J.

lay up 1.35 to the square, and weigh 9 lbs. each. Besides the standard plain roof tile there are half tile, one in each course, right and left gable tile, half ditto, plain ridge and end ridge tile, and glass skylight tile, costing 50 cts. each, and interlocking with the standard. The purlins are $2\frac{1}{2}$ ins. by $2\frac{1}{2}$ ins. by $\frac{1}{4}$ in. angles spaced 13 $\frac{3}{8}$ ins. and span for heavy winter snows 12 ft. between trusses, though we have a few shaky ones about the plant spanning as much as 16 ft. The last purlin at the eaves must have a 3-in. leg up-standing to allow for the fact that no interlocking edge of the next course below is under it, so that it will droop sorrowfully unless a special angle is provided.

For windows one must have inside and outside 4-in. tiles to cover up unevenness of the tile up and down the window opening, or else cut all the tile. The windows are first set up and tile laid towards them from the pilaster grooves. It then makes little difference whether a course over-runs or under-runs the opening by an inch or so, as it is all covered by the wide stile. The boilers were four 250-hp. Babcock & Wilcox water-tube, though the writer would have preferred fire-tube for the low pressure carried of 110 lbs., because the water contained four grains of solids in the gallon. This fact even with water softening necessitated cleaning once every three months, which operation is considerably more expensive with water-tube than with fire-tube boilers.

The coal used is buckwheat anthracite at \$1.10 a ton delivered, running about 20 per cent ash and burnt on 5.5 per cent pinhole grates with a $1\frac{3}{4}$ -in. blast from a 48-in. Sturtevant fan. The duct for the blast for the 1,000 hp. of boilers is 3-ft. by 44-in. walls of reinforced concrete running through the boiler foundations with outlets into the 20-in. side walls of the boilers, where cast-iron blast gates admit the air into the ash-pits.

In the power house are located the blast fans, feed pumps, feed water heater, main works fire pump, two electric generating units with foundation for a third, and foundation for a 70-hp. air compressor.

The blast fans are two three-quarter housing Sturtevant bottom-delivery steel-plate fans, driven by two 6-in. by 6-in. vertical, splash-oiled engines running at 325 r. p. m. They were provided with an automatic regulator, which kept the steam pressure within 3 lbs., plus or minus. It must, however, be properly set so as to keep the fan going nearly all the time. Otherwise it will shut it down for, say, 15 minutes and then run at full speed for about 5, repeating every 20 minutes, with the result that the regulator soon gets out of order and knocks develop in the engines, because the water of condensation in the steam pipe accumulates during the 15 minutes shutdown and makes trouble for both valve and engine when it finally starts up again.

The feed pumps are two 12-in. by 8-in. by 12-in. single Blake steam pumps mounted on concrete piers with an 800-hp. Berryman feed water heater on a

pier at the ends of the two pumps. This disposition not only gave a compact arrangement of piping for hot and cold feed with the necessary by-passes but it also permitted building a concrete drip-sump very cheaply of the space between the pump foundations and the heater pier, which already formed three sides to the sump. A closing wall at the end and two 4-in. partition walls cast across the sump divided it into three parts with pipe siphons in each wall, having a fall of 1 in. level of the three divisions. This made a very effective oil scrubber of the sump for everything but the engine drips, which went right down the sewer, having too little water and too much oil. But the drains of the two boiler feed pumps, always constantly cracked open, the two main engine separator drains, the condensation from the feed water heater and the drip from the main exhaust-head were all led into the first sump compartment and arrived at the third without a particle of oil, being quite clean anyhow, and this hot drip water, amounting to about 5,300 gals. daily, was returned to the boiler by a by-pass from the main feed pump suction.

The reason for choosing the feed water heater of 800-hp. size when there was only 250 hp. of steam available in the power house is that a feed water heater, heating up 800 hp. of feed water to 190°, will only use about 250 hp. of exhaust steam which it condenses and the rest all goes up the exhaust pipe. We therefore bought the 800-hp. size of heater to take care of all the feed for our three 250-hp. boilers, and gave it all the 250 hp. of the power house, all of which was condensed and returned to the sump as drip, there being only a slight wreath of steam coming out at the exhaust head.

As in all works, some sort of storage of feed water had to be provided—at least enough to run the plant four hours in case of sudden shutdown of the city mains from a burst pipe or other repairs. The most available spot was the space behind the economizer lean-to back of the chimney. Since a round iron tank would require expensive concrete foundations and could not be depended upon to last more than 15 years in an acid plant we decided to build an 18,000-gal. concrete feed water tank, 8 ft. wide, 12 ft. deep and 18 ft. long, sunk 6 ft. into the earth to sand subsoil. The tank was designed with expanded metal reinforced concrete panels and pilasters above grade, plain walls below.

To provide coal for the boiler plant the most economical scheme in the long run is a concrete trestle, up which the dump cars of buckwheat anthracite from the mine could be run. As the natural slope of the coal is 30 degrees, the center line of this trestle comes out 11 ft. from the wall of the boiler house for 13 ft. from grade to top of trestle rails, allowing the coal to slope well into the 12-ft. firing alley in front of the boilers. This trestle has a 5 per cent gradient with easements at top and bottom, and cost \$9 a foot complete with concrete piers and four 8-in. by 24-in.

wooden stringers. It was surrounded with a wooden storage bin holding 1,000 tons of coal having a floor of 4-in. concrete with a blind drain underneath, a strainer of four loose brick in the center of each bent serving to drain out the rain water in the coal. We found that the 6-in. hollow tile wall of the boiler house would not stand the strain of coal banked up against it and had to be reinforced with 8-in. I-beams bolted from column to column, two to each bent above the coal doors, which themselves had 8-in. channel lintels.

To get a "ring" or closed steam circle out of a battery of four boilers in a row, we had recourse to the 4-in. auxiliary main passing across the rear of the main nozzles and connecting all of them to the main piping. This would furnish 200 hp. at a pinch and keep the power house going if a shutdown should become necessary in the main line. This 4-in. auxiliary should remain normally empty, or else it will fill with condensed steam and there will be a frightful water-hammer as soon as you want to use it in emergency. The main steam bends were set horizontally, as there was no room for vertical bends, and on each connection was an 8-in. Crane automatic stop-check valve and a Pratt & Cady 8-in. gate valve. For the rest of the steam valves it is advisable in general to use globe valves for points normally closed and at angles, and gate valves for places normally open. To avoid multiplicity of division valves we put in $1\frac{1}{8}$ -in. cast-iron machined rings in between flanges in place of them, and kept in the power house some machined blanks which could be slipped in instead of the rings, thus blanking off a section in case of serious trouble. This arrangement let us out with only one main division valve.

The blow-off line is another matter of interest. We used the Hancock angle blow-off valves because they had a pilot on the disc which entered the seat first, thus straining out and pushing back all grit and scale before the disc came down on the seat. It was protected with an iron asbestos-packed straight cock. The blow-off line itself must always be of wrought iron screwed steam fittings, as the pressure generated by the blow-off water at 350°, flashing at once into steam as soon as it passes the cock, is very great, and will easily burst cast-iron soil-pipe or tile. Unless there is a convenient waste space or ditch to blow into it must sooner or later go into the main works tile drain. Even 200 and 300 ft. away from the blow-off valve it will still be over 300° and will crack any tile drain it is run into, so a sump must be provided. This sump is 13 ft. long and cost \$120; is continuously half full of water, over which the blow shoots out, escaping through an 8-in. vent in the manhole riser. The blow water and condensed steam mixes with the water in the sump cooling down to about 140°. It overflows from there into the works main drain but the steam is prevented from taking the same outlet by a concrete trap wall. The pressure in the sump is

about 5 lbs. per sq. in. and must be provided for by reinforcing.

The electric generating units in the power house comprise two 11-in. by 12-in. Harrisburg side-crank engines of about 80 hp. on 110 lbs. of steam, direct connected to 50-kw. Westinghouse three-phase, 60-cycle, 220-volt alternators. Both alternators in parallel furnish night-load light and power, and either one alternately runs the day power load. As the engines possess an excellent system of splash-oiling and the alternators have neither commutators nor brushes, this link in the power chain is safe and sure.

The next link, the exciters, are direct-current and hence not so reliable. In fact they gave the only trouble in the electric end of the plant. They were ordered totally enclosed to protect the brushes and commutators from acid fumes, but ran so hot in actual service that the covers had to be taken off. They were 3-kw. Westinghouse slow-speed Type S dynamos, 110-volt compound wound, either one large enough to excite both alternators. As it is absolutely necessary to have two exciters so as never to be completely shut down, the simplest and most economical drive is to couple them to the stub ends of the engine shafts. This is cheaply accomplished by casting a small concrete pier integral with the engine and alternator foundations, and coupling by a plain flange coupler. For this reason the engine shaft must be ordered turned down small at the end and provided with a keyway. The exciter and main shaft are lined up, the flange coupling bolted and the cast-iron base plate of the exciter is grouted up true.

The next consideration is the switchboard. Three panels are sufficient, one for each generator and one combined light and power feeder panel. As neither currents nor voltage are large, no extreme accuracy is required in the instruments; ordinary Weston round type or Westinghouse Type K. A. C. instruments, costing about \$19 apiece are good enough. The panels are in three sections of standard 32-in. width, 20-in. top section, 50-in. middle section and 20-in. bottom section, of 2-in. green Vermont slate finished in jet black enamel. It makes a very handsome board and one quite immune from acid, as there are no iron veins, such as are found in Pennsylvania slates. Of course all the varieties of marbles are doubtful because of the resulting sulphates of lime in the presence of acid fumes.

On each generator panel are: Top section, three Westinghouse Type K 200 amp. ammeters, one in each phase; middle section, one D. C. exciter 150-volt voltmeter, D. C. 50 amp. ammeter, exciter field switch, equalizer switch, synchronizer and voltmeter plugs, three-pole 200 amp. main generator knife switch, and exciter and alternator field rheostats mounted on the back of the board with concentric handles on the front; on a swinging bracket, two 300-volt A. C. voltmeters and a synchroscope. This last is entirely unnecessary in a plant as small as this, as synchronizing

is always done with the lamp, which is far more accurate than the synchroscope.

The panel for light and power combined has on top section three 100 amp. A. C. ammeters, one in each light circuit; middle section, three 110 amp. two-pole J. T. E. circuit-breakers for light circuit, one three-pole 200 amp. circuit-breakers and one three-pole knife switch, 200 amp., for power circuit. On swinging bracket one Westinghouse three-phase indicating wattmeter and one 300 volt A. C. ground detector voltmeter.

If you really want to know your plant thoroughly it is a good plan to connect up the board and generators yourself. The three generator leads are all tagged A B C at the factory and must lead to corresponding bus bars. Test out each separately by trying the alternators one at a time on some motor out in the works. If the leads have been mixed anywhere the motor will reverse. Mark the exciter positives and lead them to the same side of the alternator field switches, chalking an arrow here and there on the wire for the direction of the current. Be sure both synchronizer lamp transformers are across the same pair of leads of the two machines, say across A and B of both. To be in phase the main switches must be thrown when both machines are at the same position of the same sine wave in the same direction. To know this point to throw in, a small 110-volt transformer is connected permanently to each machine lead on the same phase of both and the secondaries of these transformers connected together with a 200-volt lamp in circuit. Now if both machines are making a plus sine wave at the same time the transformer secondaries will add together to give 200 volts, thus lighting the lamp to full brilliancy, and, on the contrary, if one is making a plus wave and the other a minus the secondaries will neutralize and the lamp will go out. The secondary leads are always crossed so that the lamp will go out in full phase and light up when full out of phase, because, owing to the heat in the filament the lamp stays dark a much shorter time than it stays bright, thus indicating the point to throw in with better accuracy. As, you perceive, that crossing the leads will make the lamp indicate exactly opposite, and as it is easy to confuse them in connecting up behind the board, it is best to fuse the main generator leads with temporary 100-amp. fuses before throwing them in for the first time. If everything is all right the fuses remain quiet and you can take them off and hook in permanently. If you've got it crossed anywhere the fuses will blow like pistol shots and all you have to do is to reverse the transformer secondaries and everything will be all right.

Once in right, the regular process of throwing them in is as follows: One engine will be already on the load. Start the other, plugging in its voltmeter and bring up to the same voltage as the loaded engine. Plug in the synchronizer and the lamp will at once go up and down, doing this slower or faster according as

the speeds are near together or far apart. Adjust the speed of the incoming machine with the throttle until the lamp goes from bright to dark about once in four seconds. Throw the switch at lamp dark just after the last red disappears, spin open the throttle and adjust the rheostats until both engines carry the same load. If they cannot be made to do this with the rheostats, one engine needs its governor springs adjusted as it is getting too much steam at the same speed. Adjust until equalized. To take an engine off, throttle it down until nearly all its load is transferred to the other engine and throw open its main switch.

For best efficiency all the ammeters should show the same in all phases, but as the lights are on three separate phases and cannot in practice be exactly balanced this cannot usually be done. Indeed it makes no difference if a whole light phase is out, as the unbalancing simply calls for a little more field excitation, of no real importance.

At twelve o'clock and at midnight every twenty-four hours about a chemical works the ground detector should be switched on all three phases and the ground noted in a log book. Usually it will run about 20 volts in a 240-volt circuit. If it gets as high as 50 volts in any phase it had better be traced up and attended to. It is usually acid salts or acid drippings or something or other, or else a lamp-socket ring not sufficiently tarred. Acid condenses on lamp bulbs and sockets, and usually establishes a light ground through the tar, which must be watched.

Finally, be sure that the electric company furnishes instruction books for setting up and connecting up all instruments, or else they will not be at hand when wanted, causing much vexation and delay.

The general figures of cost of the complete power equipment for the Bergenport Chemical Works follow:

COST OF COMPLETE POWER EQUIPMENT, BERGENPORT CHEMICAL WORKS.

Coal trestle:	
Excavating	\$ 380.79
Concrete	2,366.83
Mixer	70.77
Wood work	1,886.87
Miscellaneous	57.93
Track work.....	800.00
Total	\$ 5,562.49
Boiler house:	
General	\$ 27.75
Foundations	496.96
Iron work.....	1,571.58
Roof tile.....	424.31
Wood work.....	109.79
Floor	94.57
Brick work.....	1,548.10
Total	\$ 4,267.79
Boilers:	
Foundations, concrete.....	\$ 703.25
Brick work.....	2,764.76
Boilers	10,506.91
Steam lines (all over plant).....	5,170.14
Flues	937.98
Drains	344.49
Feed water tank.....	322.24
Miscellaneous	72.34
Total	\$20,822.11

Economizer:	
Apparatus	\$ 2,834.33
Foundation	311.76
Roof, miscellaneous	682.69
Total	\$ 3,838.78
Chimney	\$ 2,651.61
Feed water heater apparatus (including foundation)	\$ 474.27
Forced draft:	
Fans and engines	\$ 836.48
Foundations, etc.	241.58
Total	\$ 1,078.06
Pumps:	
Foundations	\$ 154.81
Pumps	459.96
Piping	162.15
Total	\$ 767.92
Electric plant:	
Foundations	\$ 585.23
Engines	2,456.77
Dynamos	2,381.49
Switchboard	1,175.51
Building	2,239.92
Motors	865.27
Electric light fittings	589.35
Motor wiring installed	1,195.22
Light wiring installed	732.85
Motor installation	652.91
Pulleys, belting and shafting	491.80
Miscellaneous	690.21
Total	\$13,256.53
Grand total	\$52,819.56

According to a United States consular report, an American, resident in Mexico for some years, who during that time has been experimenting in rubber-producing plants, has discovered in the Mazanillo district a tree, a vine, and three plants which yield first-class crude rubber. These plants are not yet botanically classified and are known here only by their Indian names. Analyses have been made of all these plants in the United States, and the result of each test is said to have been the production of a first-class crude rubber. The tree is of quick growth and can be propagated from slips put in the ground. Limbs that were thrown in the shade and left for six months germinated and on being planted took root and are now growing well. Trees planted from slips will be large enough to prune in three years; limbs cut from the trees furnish the rubber; pruning can be done every year. This tree produces about 6 per cent crude rubber. The three plants referred to grow wild and can be reproduced from the seed. In three years they become well rooted. The plant, being then from 4 to 6 ft. in height, may be cut to the root each year, using the cuttings for the extraction of the rubber, leaving the root to recover itself and produce the same each year. The mode of cultivating these plants is about the same as for corn; they are said to produce 8 per cent crude rubber. The vine gives a free crude rubber with no gum. It grows in the shade, extending over the surrounding trees, and is the richest in rubber, giving 10 per cent of the crude commercial product. Experiments in the cultivation of this vine are now being carried on.

USE OF HIGH PRESSURE GAS FOR INDUSTRIAL PURPOSES.*

The *Journal of Gas Lighting*, referring to this subject in a recent issue, notes that Herr F. Messinger, Official Gas Inspector for Charlottenburg, Germany, in describing a number of installations, mentions specific examples thereof. The author does not deal with high pressure gas for lighting purposes, nor with the supply through high pressure street mains. He refers to independent compressing plant for the supply of factories and even small workshops. The gas drawn from the ordinary low pressure supply mains is compressed by means of a rotary blower, which in the

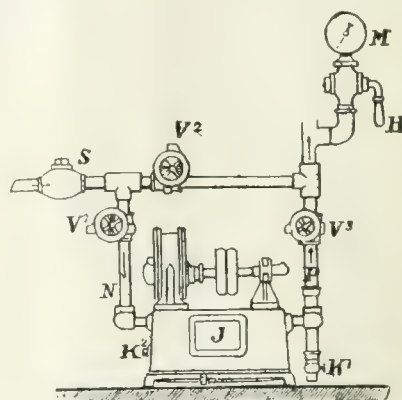


Fig. 1—Sketch of Compressor and Connections.

case of small workshops may, for economy of ground space, be mounted on a wall bracket. The blower may be driven from any power shaft in the factory, or by a gas engine or electro motor. The power required is about 1-horse power per 1,000 cu. ft. of gas compressed per hour. The arrangement of the blower and connections is shown in Fig. 1, in which *S* is a safety relief valve on the low pressure gas supply; *V*¹ is a high pressure valve on the low pressure gas supply; *V*², a high pressure valve on the by-pass connection between high and low pressure services; *V*³, a high pressure valve on the high pressure supply; *K*¹ and *K*², outlets for condensed liquid; *M*, a high pressure gas gauge; *H*, a cock on the connection to the gauge; *J*, an inspection lid to the chambers of the compressor. It will be seen that surplus compressed gas is returned through a by-pass to the low pressure side of the compressor. Any desired pressure may be secured by weighting the valve. Automatic lubrication is provided. A special high pressure valve is used at the outlet of the compressor, but the remaining taps are of the ordinary full-way type, though of sound and heavy construction. They must be capable of withstanding a pressure of 4 ins. of mercury without any leakage. A reciprocating piston pump may be used for compressing the gas; but this occupies more space, is more complicated, and gives a less regular pressure than a rotary blower.

The ordinary gas service pipes will serve for high

*From the American Gas Light Journal.

pressure gas provided they withstand the test of 4 ins. of mercury pressure. The author points out that it is a mistake to control the flow of high pressure gas to the burner by a gas tap in the ordinary way, because there is a certain intervening space between the tap and nozzle or nipple through which the gas issues. If the flow of gas is checked at the tap, the pressure between the tap and nipple is reduced, and the high pressure gas then becomes in effect low pressure gas. Care must be taken that the high pressure is maintained right up to the nipple. Dr. Fink of Berlin has devised a regulating nipple (which is shown in Fig. 2), the object of which is to control the rate of flow of the gas without any diminishing of the pressure. In these figures, *D* is the lever or index hand of the tap *C* traversing the scale *E* and altering the position of a conical needle by the eccentric movement of the tap on the cone *A*. Once the right position of the regulating nipple for the flame required has been ascertained, the same flame may be again and again produced by merely bringing the index hand of the tap to the same position on the scale. The needle moving to and fro in the nipple serves also to keep the latter free from obstruction. The orifice in the nipple is of as small diameter as will serve for the largest flame which will be required for the work to which the particular burner is put. The burners used must be constructed to exact dimensions. It is important, for most purposes, that the mixture of gas and air should be warmed before combustion in a properly constructed massive cast iron rose head, as shown in the burners in Figs. 3 and 4. The rose head becomes red hot, and the gaseous mixture, passing through the fine holes, becomes superheated, and thus the maximum heating effect is secured. Burners of this type are applicable to the heating of type foundries and typesetting machines, brass foundries, gold and coppersmiths' furnaces, and braziers' and other metal workers' apparatus. One type foundry at Berlin uses 120 of these burners, and another 200, while a third type foundry at Leipzig also has an installation of 120 burners. In these type foundries an economy of 30 per cent in the gas consumed was attained when the change was made from low to high pressure gas. Type metal melts at a temperature of 797°F.; but for the other metal processes referred to, temperatures of 2200° F. and over are needed. For many of these purposes it is impossible to prescribe a universal type of high pressure gas burner, since the particular form must depend upon the special requirements of each factory.

Special forms of burners are used for heating soldering bits and for brazing. A blow pipe flame is secured without a high pressure air supply, as the necessary air is drawn in automatically at the base of the burner. These blow pipes, or blast burners, are much more convenient than the old type with compressed air and low pressure gas. At the Opel Motor Car Works in Berlin a copper tube 0.32x0.06 in. was fused in barely 1 minute, and at the same works heavy solder-

ing bits weighing nearly 3½ lbs. have been replaced by the comparatively light high pressure blast burner. Generally speaking, for workshop use, high pressure gas is superseding coal fires or low pressure gas used in conjunction with an air blast, as the cost of the air blast pipes is avoided, and the requisite flame temperature can be attained much more certainly and be more easily controlled. High pressure gas is also generally applicable to smiths' work, and is cleaner, more

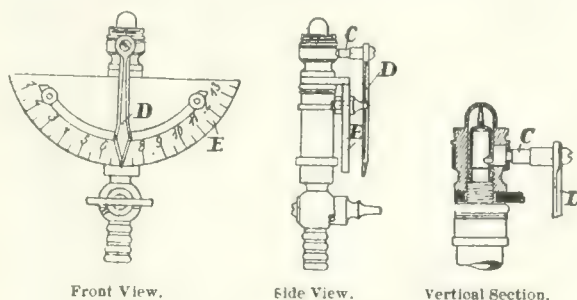


Fig. 2—Precision Regulating Nipple.

convenient and more rapid than the ordinary smiths' hearth, while the temperature attained can be more precisely regulated. High pressure gas also is largely used for muffle furnaces, where it takes the place of low pressure gas used in conjunction with compressed air.

These furnaces are applicable to all descriptions of hardening, annealing, and case hardening of metals and to enameling, and in the manufacture of metallic filaments for electric lamps. For furnaces for the

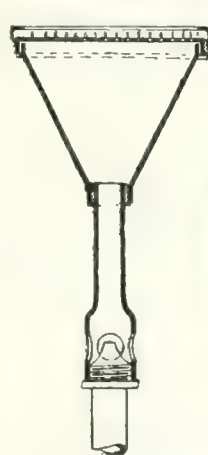


Fig. 3—High Pressure Burner.

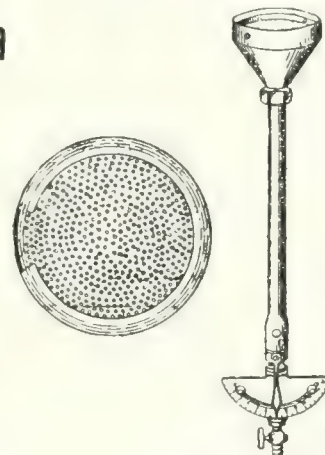


Fig. 4—Burner for Small Pots.

finer classes of work of this description, the burners used must be of the blowpipe pattern, but provided with the regulating nipple already described, in order to insure precision in regulating the flow of gas, and, as a result thereof, the temperature attained. The high pressure gas burner of suitable pattern may be introduced for the heating of existing muffle furnaces which have hitherto been heated with gas at ordinary pressure with an air blast. Special furnaces are used for hardening steel tires, as it is necessary in this instance to secure uniform heating of the steel, and at the same time to prevent the spent gases coming in contact with the metal. Other special furnaces are used for raising sheets and plates of metal to a red heat

for working prior to hardening, as in this case it is not necessary to avoid contact of the gases with the metal, and consequently a more economical type of furnace than the muffle, and one more adaptable to objects of different shape and size, may be employed. Crucible furnaces for high pressure gas are also applicable to the melting of metals of all kinds, and especially gold, silver, copper and brass. Similar furnaces may be employed for melting large quantities of more readily fusible metal (such as lead), in which case graphite crucibles prove the most durable, or for the hardening of fine steel objects, for which apparatus the bath consists of a crucible of steel or wrought iron. Oil baths, working at a temperature upwards of 570° F. for tempering and annealing articles of steel, may advantageously be heated by high pressure gas as a very uniform temperature may be maintained thereby, if the burners are distributed equally on both sides of the bath. Special high pressure gas furnaces are also employed for the brazing of steel and other tubes. In railway carriage and locomotive shops the experiments made hitherto in Germany with the use of high pressure gas for shrinking on and removing wheel tires have not proved very satisfactory, but it is hoped that a new type of burner will give good results. Boiling burners and smoothing irons may be heated with advantage by high pressure gas in workshops in which it is available, and, provided a proper regulating nipple is used, considerable economy of gas consumption is secured.

The high pressure gas burners for heating apparatus referred to by the author are intended to work with gas at the same pressure—namely, about 4 ins. of mercury, or 55 ins. of water—as high pressure burners for lighting purposes. This has the advantage that the high pressure burners for industrial heating may be supplied from the same service pipes as the lighting burners, and only one small compressor is needed for the compression of the gas for both purposes. Since high pressure gas for lighting effects an economy of 50 per cent over gas at ordinary pressures, it is desirable to secure this economy by employing high pressure gas lighting wherever a high pressure gas supply is adopted for industrial heating.

Nearly all the painting in the foreign settlements and all in the Chinese cities is done by native workmen with Chinese paint. The use of Chinese material and labor is because of cheapness, although a piece of work never has a finished appearance. Most of the work is done by the painter taking a bunch of cotton waste, which he saturates by dipping in the paint pot, and then rubbing it over the surface to be painted. Even if foreign brushes are furnished, the Chinese painter will not use them. He says he saves paint by his method, and gets it more evenly distributed over the surface. On exterior surfaces the Chinese paint is not durable, but the cost is so small that the job can be done two or three times for what one painting would cost in America.

THE SARCO MULTIPLE CO₂ RECORDER.

A new type of multiple unit carbon dioxide recorder, embodying several distinctive features, has recently been placed on the market by the Sarco Fuel Saving & Engineering Co., 90 West St., New York City. This recorder is designed to make continuous determinations of the CO₂ content in furnace gases and in its several sizes will make separate and simultaneous records from one up to ten boilers. The motive power for operating the instrument consists of, first: A direct connected motor-driven suction fan to draw to the machine a continuous stream of gases through the various pipe lines. Second: A motor-driven rotary pump to handle the paraffin oil in the successive operations which constitute a cycle for the measuring off of a sample of gas, the absorption of its CO₂ content, the recording of the same and the expulsion of the sample.

A clock mechanism substantial enough to drive by means of a shaft several drums on which are mounted the individual charts for each furnace is used. A sufficient quantity of paraffin oil, which being led to each unit from a central supply tank, performs, by virtue of its head, the successive operations of trapping off the gas, forcing it through the absorbent or K O H (caustic potash) solution, and finally, by being siphoned from each unit by a central siphon, enables the gas to be expelled and the incoming fresh gas to be passed through. The paraffin oil is all siphoned into a common tank, filtered and pumped up into the supply tank, being thus used over and over again.

The CO₂ recorder is composed of several distinct and separate units, though all depend upon the same central source of power for their operation. Each unit has its own measuring burette for trapping off a known volume of gas; connecting capillary tube to lead the sample to the absorption vessel, and connecting tubes from the same to a recording burette which carries a frictionless float. The latter carries with it in its travel the recording pen, which is suspended from a balanced and frictionless wheel. Separate exit tube to discharge gas after a record is made of the CO₂ content are provided, as is a 24-hour chart revolving on its own drum. These are all close connected and compact, so that the over-all dimensions of a four record machine are: width 42 ins., height 39 ins. and length 13 ins.

Each unit is capable of making from 30 to 40 records per hour, though for ordinary power plant work 12 to 15 is sufficient. Having separate inlets, there is, of course, no possibility of the gases from the several furnaces becoming mixed.

The special features of the new Sarco Multiple CO₂ recorder are: Separate units, elimination of adjustments except the initial one of setting the pen at zero by means of a turnbuckle, taking simultaneous records, cutting down of time lag, mixing of caustic so-

lution in recorder itself, elimination of temperature errors, automatic handling of liquids, paraffin oil and caustic potash, self-inking pen, and complete absorption of gas.

The accompanying diagram and description will make plain the working of the recorder. Furnace gas is led to the CO₂ recorder through individual pipe lines, each of which is connected to a seal (1). Each gas line has a $\frac{3}{4}$ -in. sampling tube perforated with holes extending into the last pass of the boiler, and just outside the flue, or boiler setting, a filter is placed

oil (6) located at the top of the machine. The oil is led to the units by the line (7) and by virtue of its head draws samples of gas through the tees located between the seals and internal gas line. The gas passes through inlet lines (8) to the measuring or inlet burette (9). As the paraffin oil which is being constantly fed to the units gradually fills the pipe lines (10) it rises in each measuring burette to the inlet connection (11) and prevents further gas being trapped. The oil continuing to rise forces the gas through the capillary copper tubes (12) into the bot-

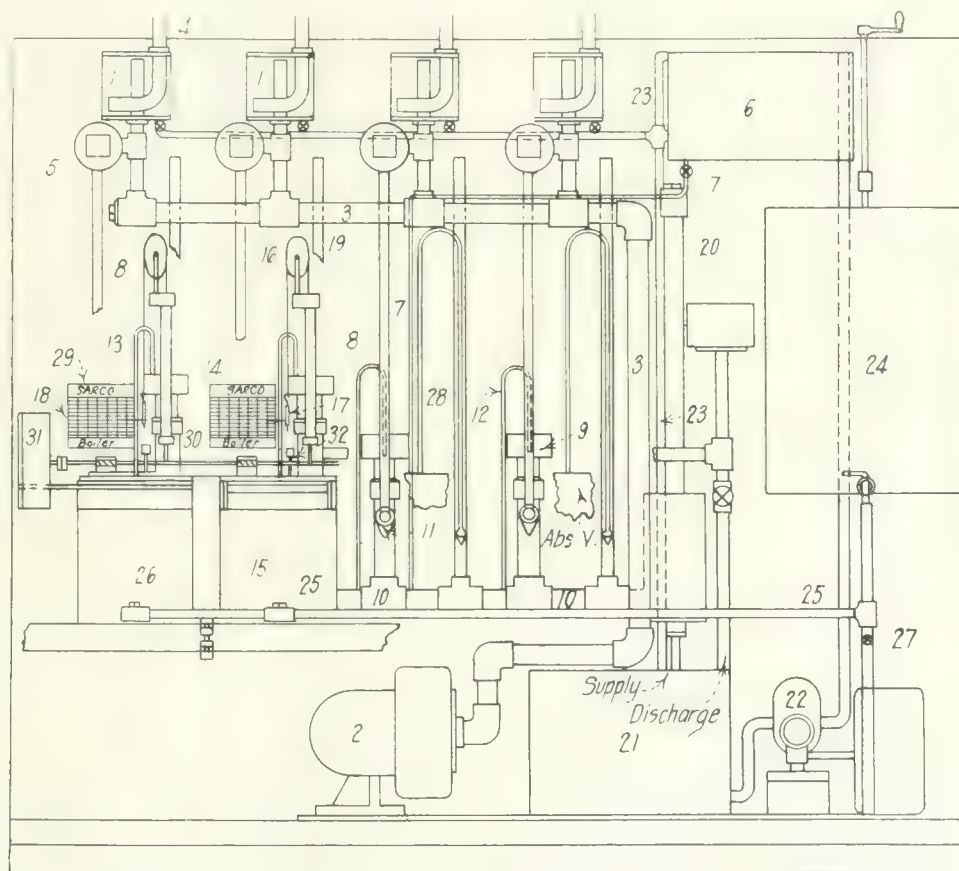


Diagram Showing Arrangement of Sarco Multiple Unit CO₂ Recorder.

to remove the soot from the gas. From the filters, $\frac{3}{8}$ -in. pipe is used to conduct the gas to each unit of the recorder, the small diameter being used to reduce time lag, and because no clogging will occur, as filtering and draining of moisture is done at the boilers.

Suction is produced by means of the suction fan (2), motor driven, located at the bottom of the machine. A steady stream of gas is drawn through the seals (1) and internal gas line (3) which is connected to the seals. The object of the latter is to produce an equal degree of suction for each unit which can be observed, as the seals are made of heavy glass tubing and the gas is bubbled through the paraffin oil in the seals, the inlet pipes (4) being perforated. Before passing to each unit, the gas is again filtered at the internal filters (5), which are cast iron cylinders containing a brush through which the gas must pass.

The motive power for drawing the gas through each unit is provided by an overhead tank of paraffin

oil (6) located at the top of the machine. The oil is led to the units by the line (7) and by virtue of its head draws samples of gas through the tees located between the seals and internal gas line. The gas passes through inlet lines (8) to the measuring or inlet burette (9). As the paraffin oil which is being constantly fed to the units gradually fills the pipe lines (10) it rises in each measuring burette to the inlet connection (11) and prevents further gas being trapped. The oil continuing to rise forces the gas through the capillary copper tubes (12) into the bot-

simply pumped up into the supply tank by means of the pump (22). The supply tank contains a filter to constantly cleanse the paraffin oil, and also has an overflow communicating to the discharge tank.

Caustic potash is placed in the tank (24), water added and the whole thoroughly mixed after dissolving. The solution is then fed to each absorption vessel through the line (25), the plug cocks (26) shut off, and if desired the remaining solution can be run off through valve (27) and the system washed with water to prevent crystallization of the solution which will, of course, stick to the inside of the tank and pipe line. When the solution in each vessel is saturated with the CO_2 gas and will absorb no more, the plug cocks (26) are opened and the solution run off as before.

When the siphon starts to flow, the paraffin oil in the system having the pressure reduced, recedes from the measuring burettes. The gas which has forced the float downward is then passed back into the absorption vessel at the top through tube (13) and leaves vessel by exit tube (28), whence it reaches atmosphere by riser tubes (19).

The recording charts (18) are mounted on drums (29) which are driven by worm gears on clock shaft (30). The latter is driven by a powerful clock (31). To remove chart drums, it is only necessary to lift them from their base plates. The latter receive their motion from spur gears located on their shank, driven by the worms on clock shaft.

As soon as the siphon has stopped flowing, one cycle is completed, the time being about three minutes, if 20 records per hour are to be taken. The gas is again drawn through the inlets to each unit and another operation commences. The pen is automatically inked by returning to ink-well (32) after each record.

As to renewals of solutions—the paraffin oil is never changed; the ink-well has a capacity for a week's run; the caustic solution, if made to about 1.25 specific gravity, will last ten days on 24-hour shifts. The only adjustment necessary is setting the pen to zero when running on air. Everything else, except the winding of the clock, chart and caustic renewals, is automatic.

The state of Victoria, Australia, has been operating a coal mine of its own for a year, the output for 1910 amounting to 162,216 tons, of which 125,793 tons were sold to its own railways. The cost of mining was \$1.95 per ton.

A company has been formed in Sydney for the purpose of establishing frozen-meat works in New South Wales and Queensland and to arrange distributing facilities for meat in England. The company is capitalized at \$12,150,000.

THE CAMPHOR INDUSTRY.*

Camphor, produced from the camphor tree (*Cinnamomum Camphora*), has a large use not only as a drug but more especially in the celluloid industry, where it is used in the transformation of cellulose nitrate to celluloid. It is used extensively in the manufacture of smokeless powder, in the making of substitute leathers, and generally in the plastic pyroxylin industry.

Imports of crude camphor into the United States for fiscal year ending June 30, 1910, were 3,026,648 lbs., valued at \$921,926; refined and synthetic camphors added to this 477,269 lbs., valued at \$179,965.

The sources of camphor are practically all foreign. Experimental plantations in the United States will not be ready for production within three or four years, and while the indications are all toward the success of this venture, the commercial production of camphor on a large scale in this country is not to be expected in the immediate future.

At present, barring the production of synthetic camphor, our sources of supply are the East Indian Islands, Borneo, Sumatra, and particularly Japan and Formosa. In Japan and Formosa the camphor trade is a government monopoly. All camphor and camphor oil produced must be sold to the monopoly bureau. The amount of camphor to be placed on the market, the price to the producer, and to the vendee, are fixed by the government.

The privilege of engaging in the industry is granted by the government to individuals and companies on application. The applicant is required to furnish some evidence of his financial standing. A certain territory is allotted to each operator and he must confine his operations to that tract.

The trees are found only in mountainous districts. In Formosa they are in the interior, where the collection of camphor is rendered hazardous by the presence of the head-hunting tribes which infest the region. Conditions have improved in this respect since the Japanese occupation of the island, owing largely to the conciliatory policy adopted by the Japanese officials, yet in 1909 there were 96 workmen killed by the savages. Owing to the danger of attacks by the tribes wages are high, being 90 cts. gold for a native and \$2.00 for a Japanese.

The condition and extent of the forests have been variously estimated. It is sometimes stated that the camphor forests are inexhaustible. Again, it is estimated that the Formosan forests will be exhausted in 20 years. As a matter of fact the truth is between these limits; perhaps 45 years of production at the present rate will use up all the trees now workable.

However, the government and private individuals have been planting young trees. About 15,000,000 trees have been set out since 1901, and with a continuance of the policy of yearly planting it is prob-

*Prepared by P. A. Bond from U. S. Consular Report.

able that there will always be a sufficient supply of producing trees.

All camphor trees do not produce camphor. In Borneo, for instance, it is said that only one tree out of 100 bears camphor crystals. The lumber from the trees is valuable, however, and is much used for building, as it is less subject to the ravages of the white ant than most woods. If the camphor odor is quite strong the boards may be used for insect-proof boxes.

In selecting the trees which are to be treated to extract the camphor there is nothing in the outside appearance to guide the searcher. Chips are taken from the trees, and if the camphor odor is absent the tree is not cut. The trees sometimes are 3 or 4 ft. in diameter and 180 ft. tall. If the odor from the chips is sufficiently strong a search is made over the tree for a flaw in which the camphor gum may have collected. The tree is cut off below the flaw and the wood split away in order to find the deposit. The flaw may be from 4 to 6 ft. long and may contain up to 14 lbs., though 7 lbs. would be above the average.

In Japan and Formosa the body of the tree which contains camphor is cut into chips by the workers, who use for this purpose a chisel-like knife. The finer the chips the more suitable they are for distillation. They may be taken from trunk, limbs, roots or twigs. The chips are now distilled in an apparatus of native manufacture which is crude, but better than mechanical devices which have been imported from Europe.

Over a fireplace is a pan filled with water. Above the pan is a very tight circular wooden retort, about 3 ft. in diameter at the bottom and 1 ft. at the top, and 5 ft. high.

This retort is filled through an opening in the top with the camphor wood chips. The opening is then closed by a double lid, and earth banked around the lower section of the retort prevents the escape of the camphor fumes. Just above the water pan are thin slats of perforated bamboo, through the apertures of which steam passes into the chips above, extracting the camphor. Near the top of the retort are two bamboo pipes, through which the vapors are conducted to a cooling box about 10 ft. distant. This cooling box is divided into two compartments; the lower is filled with cool water, and the upper contains a wooden screen, upon which the camphor crystallizes. Connected with the cooling box is another of exactly similar description, called the crystallization box. Both are immersed in water, which hastens the process of crystallization. Distillation requires about 24 hours. The chips are then removed through an opening in the lower section of the retort.

The two products of the distillation are crude camphor and camphor oil. The camphor is crystallized on the screens of the boxes. The oil is caught in leads and also skimmed from the water in the cooling boxes.

The camphor and camphor oil thus obtained are shipped to Kolbe, Japan, where the camphor oil is further treated. It is heated in small cast iron stills

and the fraction having a Sp. G. 0.825-0.920, which contains 45 to 55 per cent of camphor, is cooled to allow the camphor to crystallize out. When ready, the crude camphor is drained out of the oil by filtering through cloth bags.

After draining for a few days this product is packed in tubs, and after some months gives by rearrangement crystals quite as large and firm as those of the crude camphor produced from the wood.

These two grades produced from the wood and from camphor oil when mixed at the monopoly plants constitute what is known as Grade "B," or commercial crude camphor. This is the grade used by the American trade.

In the government refineries the "B" grade is somewhat purified and yields the "BB" grade, which is exported along with the "B" grade to European markets. This is also the grade supplied to the seven companies licensed by the government for refining. They sublime the camphor, producing a clear block of pure camphor.

There are two kinds of camphor on the market—the Japanese $C_{10}H_{16}O$ or ordinary camphor, and Borneo camphor obtained from *Dryobalanops camphora*, $C_{16}H_{17}OH$. The latter is more expensive, owing to the demands of the Chinese trade, and is seldom in American or European markets. It can, however, be produced by reduction of ordinary camphor. The odor of Borneo camphor or borneol is much like that of ordinary camphor, but it has in addition a somewhat peppery smell.

The tremendous increase in the use of camphor caused an increase in exports from Japan and Formosa of from 620,000 lbs. in 1868 to 8,427,000 in 1907, and at the same time a rise in price of from \$16.42 to \$168.50 per 220 lbs. Owing to the extraordinary prices industrial chemists were led to investigate the question of producing synthetic camphor, with such success that in 1905 and following they were able to compete with the natural product. In 1908 the effect upon the market of synthetic camphor was such as to reduce imports of natural camphor nearly one-half as compared with 1907.

Immediately following, however, Japan and Formosa were able to quote prices so low that practically all of the firms manufacturing synthetic camphor have been forced to discontinue operations, and natural camphor not only has regained its old market, but is striving with success to find new ones, mainly in the extreme east. It is true, however, that the rapidly increasing price of turpentine may have had something to do with this condition of affairs and also that in a few cases, notably that of the *Chemische Fabrik a. A. vorm A. Schering* of Berlin, it seems possible to make the artificial product and keep its market under present conditions.

Many patents have been taken out covering processes for the formation of synthetic camphor, but all are attempted improvements on the one original dis-

covery—i. e., that pinene, a hydrocarbon in turpentine, could be converted.

The methods for effecting the transformation are various, but all follow practically the same line. Pinene, $C_{10}H_{16}$, is treated with HCl or oxalic acid. In the first case the pinene hydrochloride is heated under pressure with acetate of lead in acetic acid, causing the formation of camphene, which has the same empirical formula as Pinene, $C_{10}H_{16}$, but is of a different structure, and is therefore easily oxidized to camphor, potassium permanganate being used to produce the change.

In the second case, where oxalic acid is used, instead of pinene hydrochloride, we have a mixture of camphor, pinyl formate and pinyl oxalate. This is washed with water and the oxalate and formate are saponified, whereby camphor and borneol are formed. The mixture is distilled and the borneol is oxidized by means of chromic acid to camphor.

Other modifications are very numerous, over 200 patents having been taken out, but these two examples are sufficient to show the general lines on which the synthesis is carried out.

The firm of A. Schering of Berlin, who have been able to continue the manufacture, do not disclose their special methods and laboratory equipment. It is generally understood that they have followed along the lines of original German patents, the second case mentioned above. How much further they can follow a downward move in prices is not known, but competent persons feel that the limit will soon be reached.

The establishment of the industry in the United States might not be so difficult. The obtaining of turpentine is an easy matter, and while not considered so good as the French turpentine, the American ranks next in favor with manufacturers.

Another fact in favor of American manufacturers would be the import duty of 6 cts. per pound which is levied on both refined and synthetic camphor.

GOVERNMENT TESTS OF WOOD FOR PAPER MAKING.

The experimental ground wood pulp mill which the United States Department of Agriculture has been equipping at Wausau, Wis., in co-operation with the American Pulp & Paper Association, has begun to grind. The carrying on of the tests now under way was provided for by a special appropriation, placed at the disposal of the Secretary of Agriculture by Congress last winter, to conduct tests of the suitability for paper making of plants and woods which seem likely to become valuable sources of supply of new material. It was considered that the best use which could be made of this money would be to conduct experiments on a commercial scale, with various kinds of wood. Some of these have already been studied in the laboratory, and found to be intrinsically suitable for pulp manufacture. Indeed, the Forest Service has actu-

ally made paper by one of the chemical processes from several of them. But in order to know whether they can profitably be utilized, under present conditions, it is necessary to test them under methods of manufacture comparable to those employed in actual business operations. In particular, it is desired to find out to what extent new woods can be used for ground pulp, the cost of which is usually less than that of chemical pulp. The Wausau mill has been built especially for the use of the government for as long as the experiments may require. Its inside dimensions are 40 by 100 ft., and it is equipped with electrical machinery and all necessary apparatus of the most up-to-date type. Part of the equipment is contributed by the American Pulp & Paper Association, and part is furnished by the government. The association will also furnish the wood for the tests. The wood now on hand includes carload lots of jack pine, spruce, hemlock and tamarack. The jack pine is to be the first wood tested. While the experiments are intended to cover woods from all parts of the country which, from the standpoint of physical properties and available supplies, promise to furnish new material for the paper-making industry, a special point will be made of tests of Western woods which are abundant in the national forests. There are enormous supplies of various soft woods in these forests for which there now exists little demand, and this fact constitutes one of the serious problems of management of the national forests. In order to have forests produce timber steadily they must be cut; but if there is a market only for timber from the most valuable kinds of trees the result of cutting is likely to be the disappearance of these trees and their replacement in the forest growth by species which are not in demand. Since the pulp mills take material too small for the lumber mills, species suitable for paper making can be cut to a low timber diameter, and thus the balance may be turned in favor of the reproduction of the more valuable kinds of trees. In addition to the benefit which the public will derive from the advancement of forest conservation in consequence of the wood pulp experiments of the government, there is the further benefit of cheaper paper which it is believed these experiments should make possible. The price of paper of the cheaper grades, including news paper, has been advancing rapidly of recent years as the supply of spruce dwindled, and American mills are now drawing a large part of their supply from Canada. If ground pulp of the requisite quality can be made from new and abundant woods as cheaply as it is made from spruce, one of the most serious problems of the newspaper publisher nowadays will or should be considerably simplified.

According to the United States consul at Birmingham, England, there are now 80 plants in the United Kingdom for the conversion of garbage of cities into electric power; they are increasing at the rate of 20 a year.



BLEACH PLANT ANALYSIS.

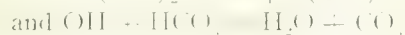
By ROBERT E. BRADLEY.*

The most generally recommended method for the analysis of bleaching solutions for the available chlorine content consists in titrating a known volume of the bleach with standard arsenious acid solution, and determining the end point by removing a drop to iodostarch paper, or to a drop of potassium iodide and starch on a porcelain plate. A very widely used modification of this consists in adding an excess of the arsenious acid solution to a known volume of the bleach, and titrating the excess by a standard iodine solution, using starch as the indicator. These methods work very well on solutions of chloride of lime, but much trouble has been experienced in titrating the available chlorine in bleach made in the wet way, i. e., by passing chlorine gas into an excess of milk of lime. A little consideration may point to the apparent reason.

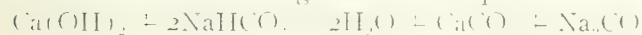
Chloride of lime is made by saturating slaked lime with chlorine gas. Under these conditions the compound formed has the composition $\text{Ca} \frac{\text{OCl}}{\text{Cl}}$, and the

amount of free alkali from the calcium hydroxide present would be absolutely zero, or even if imperfectly made, would be quite negligible. Bleach liquors made by passing chlorine into milk of lime, on the other hand, contain very considerable quantities of free hydroxide in the finished product, because an excess of milk of lime is always present.

When bleach made in the latter way is titrated, a considerable amount of OH ions is introduced into the reaction. Now, since



or summing the reaction up



i. e., the effect of the free lime is to produce sodium carbonate, which is alkaline in reaction.

It is well known that in an iodometric determination nothing but the bicarbonate may be present; all other alkaline carbonates reacting alkaline and absorbing iodine more or less rapidly according to conditions. This means that any method which uses iodine to color starch as the end point will be inaccurate if hydroxyl

ions are present. That hydroxyl ions are present in bleach made in the wet process has been shown theoretically, and the same may be very easily shown analytically. That hydroxyl ions are not present in bleach made from chloride of lime can likewise be shown.

Washburn² concludes that at the completion of the titration the solution should be neutral. A solution containing the salt of a weak acid (or base) together with an excess of the acid (or base) has the property of maintaining itself at a fairly constant hydrogen ion concentration. This means that the salt chosen as the neutralizing agent should be such that its ionization constant is equal to the desired hydrogen ion concentration, in this case being 10^{-7} . The salts Na_2NPO_4 , NaHCO_3 , and Na_3BO_3 may be used. It is calculated that at the end of the titration the solution should contain two mols of Na_2HPO_4 to every mol of NaH_2PO_4 to preserve neutrality, or that the solution should be saturated with CO_2 and be 0.12 molar in respect to NaHCO_3 . As the hydrogen ion is sensitive to the escape of CO_2 the titration should be done in a stoppered flask, and the solution kept saturated with the gas. The following directions are given for titrating an unknown solution of arsenious acid.

Make the solution neutral with HCl or NaOH as required, using phenolphthalein as indicator. The neutralizing agent is then added gradually from a burette, and then the titration is nearly completed, more of the neutralizing agent is added until the total amounts to about five grams NaHCO_3 or eleven grams $\text{Na}_2\text{HPO}_4 \cdot 12\text{H}_2\text{O}$ for every 100 cc. 0.1 iodine solution required in the titration. The volume at the end should be about 250 cc. for every 100 cc. iodine used. By following these directions, and using weight burettes, he claims it possible to attain an accuracy of 0.001 per cent.

The above, applied to the case at hand, would mean that the analysis of the bleach obtained by the wet process would have to be saturated with CO_2 at all times, or that it would have to be very carefully neutralized at the start. Neither of these will recommend themselves to a busy laboratory.

W. A. Puckner³ has shown that we are wrong in the supposition that sodium bicarbonate has no action upon iodine. He showed that when using one to two grams of the bicarbonate an error of 1.5 to 4.5 cc. 0.1 normal iodine may be introduced, even when the bicarbonate used is of exceptional purity, and especially proven free from carbonate, sulphite, or thiosulphate.

*251 Main St., Winchester, Mass.

²Smith's General Chemistry for Colleges, p. 324.

²J. Am. Chem. Soc., 30, 31-46; Chemical Abstracts, 1907, p. 237.

³Proc. A. Ph. A. 1904, 408; H. W. Selsamf, Manual of Vol. Anal., p. 1.

He shows that when bicarbonate is added to a tenth normal solution of iodine, residual titration of the iodine with thiosulphate will show a considerable loss of free iodine, which went into combination in some form or other (probably iodide), and that the quantity so lost is proportional to (1) the mass of bicarbonate, (2) the time of interaction, (3) the concentration of the solution, (4) the temperature, and (5) the size of the flask in which the reaction occurs.

These phenomena are due to the fact that NaHCO_3 , when dissolved in water, undergoes hydrolysis, thus, $2\text{NaHCO}_3 \rightarrow \text{Na}_2\text{CO}_3 + \text{H}_2\text{CO}_3$, and $\text{H}_2\text{CO}_3 = \text{H}_2\text{O} + \text{CO}_2$.

This breaking up of the bicarbonate into carbonate, carbon dioxide and water continues until the pressure of the carbon dioxide gas above the solution is equal to the pressure of the gas in the solution, i. e., until equilibrium has been reached. In concentrated solutions of bicarbonate the amount hydrolyzed is much greater than in dilute solutions. An elevation of temperature also increases the absorption of iodine.

Less iodine is lost when small flasks are used, provided the glass stopper completely cuts off communication with the atmosphere. The carbon dioxide will then escape from the solution until its pressure in the solution is equal to that of the gas above. Thus, since a large volume of air is contained in a larger flask, more carbon dioxide will pass from the liquid before equilibrium is established, hence more bicarbonate will be decomposed, and more iodine absorbed.

The above researches go to show that, except under the most exacting conditions, the arsenious acid method for the determination of the available chlorine in wet process bleach liquors is not as exact a process as could be desired, and that a more reliable method would fill a great want.

This want is almost perfectly met by Bunsen's process, where a definite volume of bleach is delivered into a large excess of potassium iodide, acetic acid added, and the liberated iodine titrated by thiosulphate. Here, since the solution is made acid, there can be no hydroxyl ions, and this trouble is avoided. Care must be taken not to use a strong acid, as this would attack the chlorates present more or less according to the time and temperature.

This method gives the most accurate results that could be desired. The only objection to it lies in the cost of the potassium iodide, and the somewhat unstable nature of the thiosulphate solution.

The London office of the Transvaal Chamber of Mines, Johannesburg, has been advised that the total gold output of the mines of the Transvaal for the calendar year 1910 amounted to \$155,742,172, against \$150,500,347 in 1909, \$145,788,710 in 1908, and \$133,360,292 in 1907. The yield of gold in New South Wales, Australia, for 1910 is reported as 225,172 ounces, valued at \$4,011,337, against 238,047 ounces, valued at \$4,231,645, in 1909.

THE MEASUREMENT OF TEMPERATURE.*

By JAMES CLINTON PEEBLES, E. E.†

It is the purpose of this paper to discuss some of the devices which are in use for the measurement of temperature, with special reference to the measurement of comparatively high temperatures, from about 500° F. up. Also, the errors to which such instruments are subject will be pointed out, and so far as possible, methods of avoiding these errors will be indicated.

THE MERCURIAL THERMOMETER.

The instrument in most general use for the measurement of temperature, is the mercury-in-glass thermometer. Other liquids than mercury are sometimes used in a glass thermometer, but the principle is the same. When such an instrument is used at a temperature of 500° F., or higher, it becomes subject to errors which are often considerable, and which the simplicity of the instrument tends to conceal.

The most common source of error in such a thermometer is boiling of the mercury, which may occur at as low a temperature as 300° F. This produces a vaporization of a portion of the mercury, with a subsequent recondensation in the upper, cooler part of the capillary tube. This trouble can be overcome by introducing an inert gas, such as nitrogen, into the capillary tube above the mercury. As the mercury rises in the tube upon an increase in temperature, the gas pressure is increased and the boiling of the mercury thus prevented. This precaution is observed in the higher grade thermometers, but instruments are on the market which are subject to a considerable error from the boiling of the mercury. When using a thermometer which has not been filled with nitrogen, a good precaution to observe is to keep the top of the mercury column as cool as possible, and so prevent boiling, or to keep the whole stem hot, which prevents condensation of the mercury. This precaution will be effective up to 350° to 400° F., but for higher temperatures a nitrogen filled thermometer should always be used.

After a thermometer has been made and calibrated it may undergo certain changes which will very seriously affect its accuracy. The most important changes, and the only one which need be considered here, is a permanent contraction of the glass which renders all the readings of the instrument high. This is caused by a slow readjustment of the internal stresses in the glass which were produced when the stem was made. Tests made by the writer have shown thermometers to read as much as 60° high at a temperature of 700° F., due to the contraction of the glass. Errors of this magnitude were found in supposedly high grade instruments, much above the average thermometer to be found in the market. It is quite probable that many thermometers in general use, the indications of which

*From The Armour Engineer

†Instructor in Experimental Engineering, Armour Institute of Technology.

are accepted as correct by the users, would be found on test to be subject to errors even greater than this.

In the manufacture of the best thermometers the glass stem is annealed before the scale is etched on. This is done by heating the stem at a temperature somewhat above the highest at which it is to be used, and maintaining it at that temperature from five to ten days. It should then be cooled very slowly, the cooling lasting from four to six days. This removes all the strains from the glass and produces a thermometer which will not change with age. In buying a thermometer for use in work where reliable indications are essential, only one which has been properly annealed in the making should be considered. Many thermometers which have not been thus artificially aged are in the market.

In perhaps the majority of cases where a thermometer is used, the bulb is placed in contact with the object of which the temperature is desired, while a considerable portion of the stem emerges into a very different temperature. For example: Suppose an experimenter wishes to determine the burning point of a sample of gas cylinder oil. The bulb and perhaps a small portion of the stem are immersed in the oil at a temperature of say, 650° F. The greater portion of the stem emerges into the air above the oil bath, the average temperature of which may not be much above 100° F. Most high grade thermometers are calibrated under a condition of total immersion, and are correct for that condition only. When only the bulb is immersed and the stem emerges into the air at a much lower temperature, the indications of the instrument will be considerably in error. The amount of the error will depend upon the difference in temperature between immersed and emergent parts, the number of degrees on the emergent stem, and the glass of which the stem is made.

Stem correction = $Kn (T^\circ - t^\circ)$, where K is a constant depending on the glass, n is the number of degrees on emergent stem; T° is the temperature of the immersed part, and t° the temperature of the emergent part. The value of K must be determined experimentally for each thermometer, as it differs for different glass. This can be done by comparing the reading of the thermometer when exposed to a given temperature under a condition of total immersion with the reading under a condition of partial immersion. This gives all the factors in the above equation except K , which may therefore be calculated.

In the thermometer mentioned above in connection with the test of cylinder oil, the value of K is .00009. The oil is at a temperature of 650° F., and the air above the bath at 100° F. Hence $T^\circ - t^\circ = 650^\circ - 100^\circ = 550^\circ$. Assume that the stem is immersed in the oil to the 50° point. The stem correction is

$$0.00009 (650 - 50) (650 - 100) = 29.7^\circ.$$

When the stem is colder than the bulb the stem correction must be added to the observed reading. Hence the correct burning point of the oil is $650^\circ + 29.7^\circ =$

679.7°. When the stem is hotter than the bulb the stem correction should be subtracted.

A reliable mercury-in-glass thermometer should be well annealed to prevent slow contraction of bulb and stem with age; the upper part of the capillary tube should be filled with nitrogen to prevent boiling of the mercury; and the stem correction should be known.

RESISTANCE THERMOMETER.

The practical limit of a mercurial thermometer is from 800° F. to 900° F. Above this it is very difficult to obtain reliable results with a mercurial thermometer, and so some other method becomes necessary. The electrical resistance of the metals is known to change with temperature, and since electrical resistance can be measured with considerable accuracy this furnishes one of the most reliable and accurate methods for the measurement of temperature.

Platinum is practically the only metal which has come into general use for this purpose on account of its high melting point and resistance to the attack of gases at high temperatures. The first electrical resistance thermometer was designed by Sir William Siemens, and was later improved and perfected by Callender and Griffiths. Siemens' instrument was made by winding fine platinum wire on a fire clay cylinder and surrounding it with a protecting tube of porcelain or quartz. This thermometer was found to be sluggish in its action, requiring considerable time to come to the temperature of the surrounding medium. It gave very good results, however, where the temperature was nearly constant, as would be the case in an annealing furnace.

In Callender's instrument the platinum wire was wound on a strip of mica and surrounded with a steel tube. The porcelain tube is very fragile and was found to break with the slightest blow when hot. It is also very likely to crack when exposed to sudden changes in temperature, and hence cannot be used with success in a metal bath. The Callender instrument, with steel protecting tube, was found to be quite sensitive and extremely accurate. In fact the resistance thermometer is probably without doubt the most accurate instrument that we have for the measurement of temperature. It is a matter of record that a thermometer of this type, designed and constructed by the United States Bureau of Standards, reached the temperature of the surrounding medium to within 1/1000 of a degree Centigrade in three seconds.

The measurement of the resistance of the platinum coil is an important point in resistance thermometry. The Wheatstone bridge method is the one usually employed, involving the use of a galvanometer, an adjustable resistance and a battery. The operation consists in adjusting the variable resistance in one branch of the bridge until a balance is obtained, indicated by a zero reading of the galvanometer when its circuit is closed. A certain amount of manipulation is therefore always necessary to secure a temperature reading with

a thermometer of this kind. Hence a resistance thermometer is not a directly indicating instrument, unless the necessary manipulation is done automatically. A well-known form of instrument for use with a resistance thermometer is a Whipple Indicator. This instrument is shown in Fig. 1, connected to the thermometer and ready for use. It consists of a Wheatstone bridge, battery and galvanometer, contained in a single case as shown. The adjustable resistance consists of a coil of wire wound upon a drum which is revolved by hand until a balance is obtained. Since the temperature sought depends upon the resistance of the platinum coil in the thermometer, and since this in turn is equal to or proportional to the adjustable resistance on the drum, at the time a balance is obtained, it follows that the temperature scale can be placed directly on

the measurement of the resistance of the thermometer coil. For this reason two compensating leads of the same size and length as the true leads, are placed side by side with the latter, and connected to two separate binding posts in the boxwood head. Any change in the resistance of the true leads is balanced by an equal change in the compensating leads. All that remains to be done is to connect the compensating leads to the adjustable resistance of the Wheatstone bridge. Thus any change in the resistance of these leads simply adds to or subtracts from the adjustable resistance in exactly the same magnitude as a change in the resistance of the true leads affects the resistance of the thermometer coil.

Since the adjustable resistance must be balanced against the thermometer coil when a reading is ob-

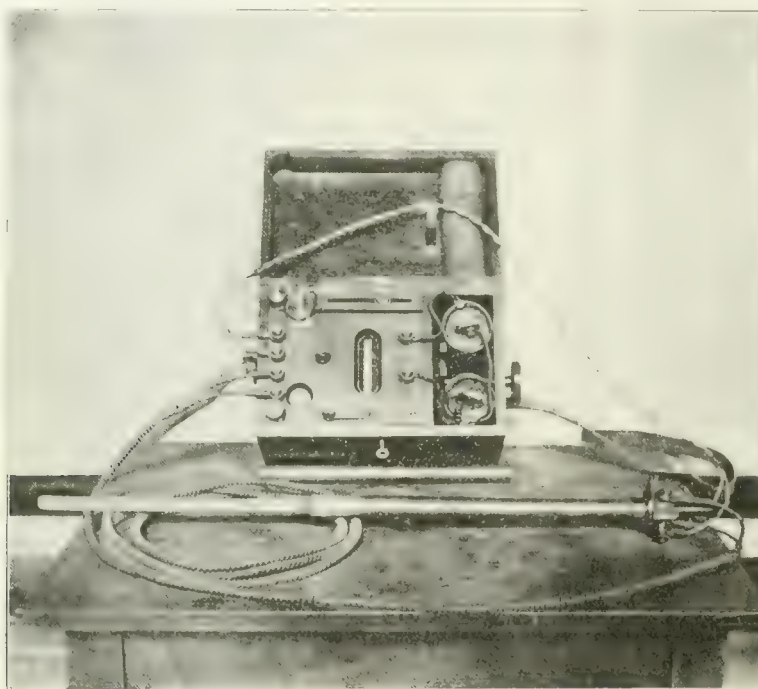


Fig. 1—Whipple Indicator with Resistance Thermometer.

this drum. Thus it is that instead of reading the resistance which has been wound upon the drum we read the temperature directly.

Of course certain known points must be located on this temperature scale, in order to make possible the step from resistance to temperature. The freezing points of certain of the metals are known with a considerable degree of accuracy, and this supplies a convenient and reliable method for calibrating an instrument of this kind.

Resistance thermometers are made in various sizes and lengths, up to 1 in. in diameter and about 30 ins. in length. The platinum resistance coil usually occupies not more than 4 ins. in the lower end of the protecting tube, from which platinum leads are run to binding posts on the boxwood head at the other end. It is important that no change in the resistance of these leads, due to temperature changes, should affect

tained, it follows that the changes in lead resistance are eliminated. Thus an instrument of this kind becomes independent of the depth of immersion, a very important point in high temperature thermometry.

A very excellent instrument for use with a resistance thermometer is the Callender Recorder, shown in Fig. 2. This instrument gives a continuous record of temperature, the chart covering a period of 24 hours. In this recorder the adjustments of the Wheatstone bridge are made automatically by means of magnets and clock work. Two magnets are made use of, one operating when the current through the bridge is in one direction and the other when the current is in the opposite direction. The adjustable resistance is operated by the clock work, the magnets simply serving to release a brake which holds the clock in check. The clock operates the adjustable resistance in one direction or the other, according to which mag-

net has operated. As soon as a balance is obtained the current ceases to flow through the magnet, which immediately lets go of the brake and stops the clock motion.

Thus, all the manipulation necessary for a measurement of resistance is done automatically and the instrument will give a continuous record of temperature.

Up to about 2200° F., the platinum resistance thermometer gives the most accurate measurement of temperature that we have. The only objections to it are a slight change in the resistance of the platinum with time, and the fragile character of a porcelain or quartz protecting sheath.

THERMOELECTRIC THERMOMETER.

When two dissimilar metals are fused together and the junction heated, the latter becomes a source of

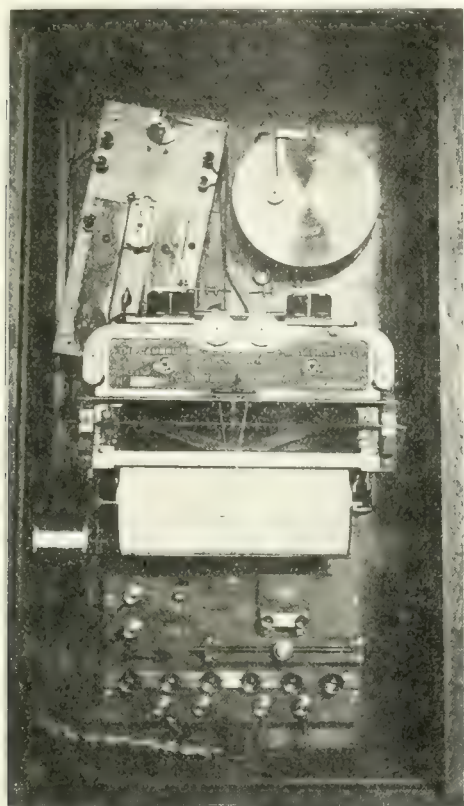


Fig. 2—Callender Recorder.

electromotive force. The magnitude of this electromotive force is proportional to the temperature to which the junction is raised. This fact offers a simple and convenient method for the measurement of comparatively high temperatures.

Such a junction of two different metals is known as a thermo-electric couple, and much study and investigation have been devoted by physicists to the thermo-electric measurement of temperature. The credit for finally placing thermo-electric pyrometry on a satisfactory basis belongs to LeChatelier. He made a couple consisting of one wire of pure platinum and the other an alloy of 90 per cent platinum and 10 per cent rhodium. This is known as the LeChatelier couple and is the one in general use at the present time.

In the commercial application of this principle, the two wires forming the couple are first fused together, and then are insulated from each other throughout their length by winding asbestos thread upon them. Each wire is then run through a small porcelain tube, the tubes extending almost to the junction. The whole is then covered by a large porcelain or quartz tube, and the two free ends of the wires led to binding posts on the wooden handle or socket to which the enclosing tube is fastened. A millivoltmeter graduated to read temperature in degrees completes the apparatus.

The magnitude of the electromotive force produced by such a couple depends, not upon the absolute temperature of the junction, but rather upon the difference in temperature between the junction and the other ends of the wires, where they are connected to the external leads. Hence, we have the terms "hot junction" and "cold junction" to designate the different ends of the wires forming the couple or "element." It is important, therefore, that the cold junction be kept at a constant temperature while the thermo-couple is in use. Neglect of this precaution may lead to considerable error in the indications of the instrument.

In addition to its simplicity and ease in handling, the thermo-electric pyrometer has the advantage of a very small time lag. It comes quickly to the temperature of the medium in which it is placed, and hence is suitable for measuring changing temperatures. In this particular it is superior to the resistance thermometer, but is not capable of such great accuracy as the latter instrument.

The sensibility of a platinum-rhodium thermo-couple diminishes rapidly below 500° F., and hence, it is not suited for measuring comparatively low temperatures. In the range between 300° F. and 900° F., the best results are obtained from the use of a couple of copper and constantan or iron and constantan. Such a couple gives a much greater electromotive force in this range than can be obtained from a platinum-rhodium couple, and hence is more satisfactory.

All metals disintegrate more or less when exposed to high temperatures for a considerable length of time. This disintegration of the metal forming a thermo-couple is also accompanied by a loss in electromotive force, and hence after long exposure to a high temperature, the indications of such a couple are likely to be somewhat in error. If platinum be kept at a dull red heat (about 1800° F.) for eight hours it will suffer a loss of about ½ per cent in electromotive force. Continued heating will not increase this loss materially, and when it is considered that all other metals suffer a much greater loss, it is easily seen that platinum is by far the best metal for a thermo-couple.

OPTICAL PYROMETERS.

There are many industrial processes carried on at temperatures where platinum either disintegrates rapidly or fuses. Such temperatures are to be found in the electric furnace, which now has a wide commercial

application. Manifestly none of the temperature measuring devices discussed thus far, as electric resistance and thermo-electric pyrometer, are suitable for use with such high temperatures.

For some time it had been the custom to estimate these temperatures by the trained eye of the experienced workman. But this method was only approximately correct at best for the same eye may vary considerably in the estimation of color. This crude method, however, furnished the clue to the discovery of a much more accurate and scientific method, whereby the temperature of a hot body is measured by the intensity of the light which it radiates. An instrument for measuring temperature by means of light radiation is known as an optical pyrometer.

The principle upon which optical pyrometry is based is known as the Stefan-Boltzmann radiation law. These

the optical pyrometer has been worked out. The method consists in comparing the intensity of the light from the hot body with that from a standard lamp, by ordinary photometric methods.

One of the best optical pyrometers is the invention of LeChatelier, the man who did much in the development of the thermo-electric pyrometer. Inasmuch as the principle used in this instrument is typical of all others, it will be described rather carefully, from which it should be possible to obtain a fair idea of the optical pyrometer in general.

The construction of the instrument may be seen from Fig. 3. A small gasoline lamp is placed at A, so that light from its central portion passes through the lense B, is reflected from the 45° mirror, brought to a focus by the eye-piece, and observed through a red glass.

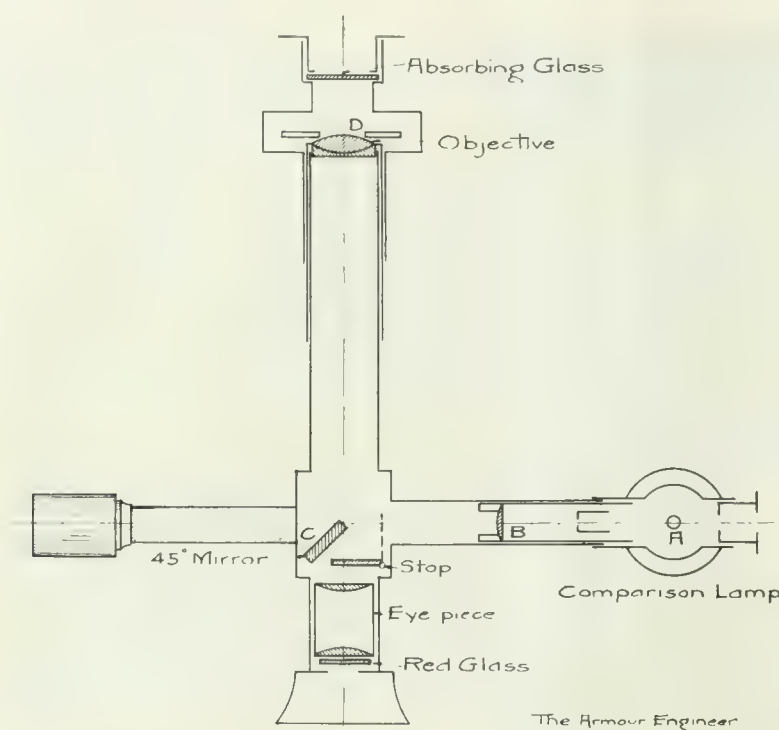


Fig. 3—LeChatelier Optical Pyrometer.

two physicists, after much study and research, succeeded in establishing the physical law that the total energy of radiation from a hot body is proportional to the fourth power of its absolute temperature. The research from which this law was deduced is discussed by Waidner and Burgess in Bulletin No. 2 of the United States Bureau of Standards.

It will be evident that if it is possible to measure the total energy of radiation with a fair degree of accuracy, we immediately have a very accurate measure of temperature, because the latter is proportional to the fourth root of the energy of radiation. Thus a considerable error in the measurement of the total energy of radiation will give a very small error in the determination of the temperature.

Photometric methods have been made use of for the purpose of measuring the intensity of the light radiated from an incandescent body, and along this line

This provides a red comparison field of constant intensity. The lamp A is mounted eccentrically, and may be turned so that the image of the flame is exactly bisected by the edge of the mirror C. Light from the incandescent body under observation is focused by the objective, passes by the edge of the 45° mirror, and forms a red field immediately beside and touching the first.

A measure of temperature is made by bringing the two red fields to the same brightness. This is done by opening or closing the iris diaphragm D in front of the objective, thus admitting more or less light from the body whose temperature is sought. For very high temperatures additional absorbing glasses of known coefficients of absorption, are placed below the objective, and for lower temperatures before the comparison lamp. The opening of the iris diaphragm, when equal intensity has been established, is read upon

a scale, the square of whose reading is a measure of the intensity of the light from the incandescent source.

Since, according to the Stefan-Boltzman law, the temperature is proportional to the intensity of the radiation, we have immediately a measure of the temperature when we have measured the intensity of the radiation. All that is necessary is to have two sources of light of known temperatures, molten metal for example. Note the reading of the pyrometer when focused upon each of these bodies and plot two points having for their co-ordinates temperatures and scale reading on the iris diaphragm. Draw a straight line through these two points, produced in both directions, and the pyrometer is calibrated for all temperatures.

There is one important point to be kept in mind in connection with the Stefan-Boltzman law quoted above. The law is true only for what is technically known as a "black body." The conception of such a body is due to Kirchhoff, who defined it as a body which would absorb all radiations falling on it and would neither reflect nor transmit any. Kirchhoff pointed out that the radiation from such a body is a function of the temperature alone, and hence may be

energy. Manifestly they are not at the same temperature, and hence the term, black body temperature violates the conception of equal temperatures which is based upon thermal equilibrium between the two bodies if brought into contact.

Nevertheless, when pyrometers are calibrated in terms of black body temperature, as all instruments based upon the Stefan-Boltzman law must necessarily be, the conception of equal black body temperatures is of great practical value.

It will be evident from the foregoing that the optical pyrometer cannot be depended upon to give the correct absolute temperature of all bodies. It will, however, repeat its indications upon the same body with unerring accuracy, and in a large number of industrial operations this is all that is required. After the proper temperature for any operation has been discovered, the optical pyrometer will make it possible to duplicate this temperature day after day with great accuracy.

There are a large number of operations where the radiation differs but slightly from that of a black body and hence the optical pyrometer will read the abso-

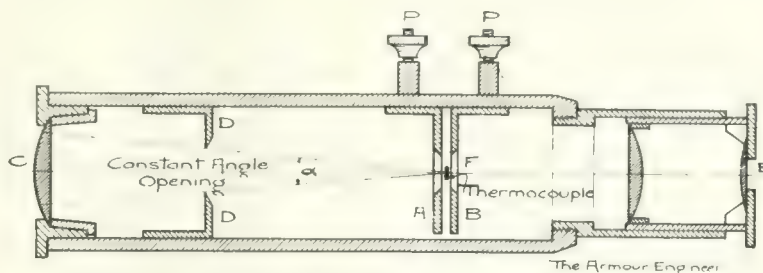


Fig. 4—Féry Thermo-electric Telescope.

used to measure the temperature. The first experimental realization of such a "black body" was made by Wein and Lummer, who heated the walls of a hollow opaque inclosure as uniformly as possible and observed the radiations coming from the inside through a very small opening in the walls of the inclosure.

It is evident that such a body will absorb all the radiations incident through the small opening, no matter what the material of the walls may be, for unless the walls are totally reflecting, all radiations must sooner or later be absorbed, except that portion which may again escape through the small opening. The presence of this small opening makes a slight departure from a theoretical black body.

No body is known whose surface radiation is that of a black body. The radiations from carbon and iron are very close to black body radiation, while the radiation from polished platinum and the white oxides departs very evidently from it. It follows, therefore, that a number of different bodies all heated to the same temperatures will radiate different amounts of energy, and hence the optical pyrometer would show varying temperatures for them all. In this connection the term "black body temperature" came into use. Two bodies are said to be at the same black body temperature when they radiate the same amount of en-

ergy. Manifestly they are not at the same temperature, and hence the term, black body temperature violates the conception of equal temperatures which is based upon thermal equilibrium between the two bodies if brought into contact.

One of the most convenient instruments, based upon the energy of total radiation, is Fery's Thermo-electric Telescope. This instrument combines the thermo-electric and optical principles in the measurement of temperature in that its indications depend upon the energy or radiation and are obtained from a thermocouple and galvanometer. The construction is shown in Fig. 4. Radiation from the incandescent body passes through the lens C and falls upon a very small and sensitive thermo-couple shown at F in the sketch. A diaphragm D D fixed in size and position, gives a cone of rays of constant angular aperture, independent of the distance from the incandescent body. These rays, falling on the thermo-couple, produce an increase in temperature proportional to the total energy of radiation, which in turn induces an electromotive force proportional to the energy of radiation. Thus a direct reading is obtained upon millivoltmeter which, according to the Stefan-Boltzman law, may be read in terms of temperature. The leads from the thermocouple are led to the binding posts shown at P P in the

sketch, to which the millivoltmeter leads are also connected. A and B in the sketch are screens placed on each side of the thermo-couple to exclude all light except that which comes from the incandescent source under observation. E is the eyepiece by means of which the image of the light source is focused on the thermo-couple.

An instrument of this kind is independent of the distance of the incandescent body, within certain limits, as will appear from the following considerations. If the instrument is sighted on an incandescent body of limited dimensions, the amount of radiation passing through the opening in the diaphragm D D will vary with the distance from the hot body, being inversely proportional to the square of the distance. If the thermo-couple were of such size as to receive all of the radiation converged upon it by the lens C, then the indications of the galvanometer would decrease as the distance from the incandescent source increases. But the thermo-couple, however, is not large enough to receive all of the radiation converged towards it. The image of the source of light, formed by the lens C is large enough to overlap the thermo-couple on all sides, so that when the observer sights the instrument the thermo-couple appears as a dark disc in the center of a bright field of light. When the instrument is brought nearer to the source of light, thus increasing the size of the image produced, the only effect is to increase the amount of this overlapping, while the thermo-couple receives no more radiation than before. On the other hand, however, if the instrument be withdrawn to such a distance from the source of light that the image formed is not large enough to completely cover the thermo-couple, the readings obtained will be too low, and will become less as the distance from the source of light is increased.

From the foregoing it will be evident that the instrument is independent of distance only within certain limits. The image of the incandescent source must always be large enough to completely cover the thermo-couple. In general, the diameter of the hot body should measure as many inches as the distance from the instrument to the hot body measures yards.

The Fery pyrometer is best adapted for use in the range from 1300° F. to 2800° F., where its indications are sufficiently accurate to answer all requirements of industrial work. It should not be forgotten, however, that an optical pyrometer depending for its reading upon a measurement of the energy of radiation reads black body temperatures, which in some cases may vary considerably from true temperatures. But when the same temperature is to be repeated time after time in the same process, the Fery pyrometer will repeat its readings with a degree of accuracy sufficient for all practical purposes.

A NEW METHOD OF METAL COATING.

The principle involved in the new method of coating various substances with metals, the invention of a Swiss engineer, and recently presented before the Academy of Sciences at Paris by a member of the institute, according to a consular report. The process consists in reducing molten tin, zinc, copper, lead, aluminum, or other metal or alloy to a state of pulverization by the pressure of an inert gas—nitrogen or hydrogen—and in that state driving it against the surface to be covered from a flexible tube with a tip like that of a large vaporizer for liquids. The apparatus is simple in construction and consists of a closed crucible, in which the covering metal is fused, with an orifice at the bottom which can be opened or closed by a stop valve worked from the outside. At the side of the apparatus stands the steel cylinder in which is supplied the condensed nitrogen or hydrogen gas, under high pressure, as such gases are used in various industrial processes. From this cylinder a tube leads through a small furnace heated by gas flames. The portion of the tube within the furnace is made spiral like the worm of a still, so as to multiply the heating surface, and the gas passing through it is raised to a temperature of 200° F. or more. Beyond the furnace the tube divides into two branches, one of which opens into the closed crucible containing the molten metal and exerts upon the molten surface the expansive pressure of the supply cylinder, augmented by heating as the gas passes through the spiral tube. The other branch of the pipe passes downward and terminates in a tip with a pointed orifice, so shaped as to promote spraying, as in an ordinary vaporizer. From the lower apex of the crucible another tube leads downward, terminating in a conical bulb, the point of which is pierced with an orifice and so held that the molten metal expelled through the orifice by gravity, aided by the pressure in the crucible, meets the stream of hot nitrogen or hydrogen gas ejected under high pressure from the first branch of the tube, as above described. The result of the mingling of the two jets of inert gas and molten metal under these conditions is to atomize or convert the latter into a cloud of metallic vapor, which, being projected against a surface of iron, steel, wood, or almost any other substance, forms upon it a metallic coating similar to the deposit of copper on an electrotype plate by the galvanic process, the difference being that by the new process, the method of coating with metal is mechanical and immediate, not dependent upon chemical action, and therefore applicable to a wide range of substances under any conditions which permit the apparatus to be used. For ordinary purposes the metalizing plant is stationary, but for metal coating large surfaces in or out of doors a portable apparatus has been devised which is said to give satisfactory results.

The output of oven coke in Canada in 1909 was 821,727, as against 852,200 tons in 1908.

METALLURGY.

USE OF MAGNESIUM IN DEOXIDIZING ALUMINUM ALLOYS.*

By H. M. LANE.†

Many foundrymen report a great deal of trouble in melting aluminum, and an examination of the castings turned out seems to indicate that the metal has been oxidized. We know that aluminum is oxidized with exceeding ease, on account of the fact that it is one of the metals in the manufacture of which much heat is absorbed, and consequently when it has an opportunity to oxidize, this heat is released.

The greater the amount of heat necessary to break down the oxide of any metal to form the pure metal, the greater will be the ease with which it will oxidize. In metallurgy advantage is taken of the fact that some metals oxidize easier than others, both in refining of metals and in the protection of certain metals against oxidation. For instance, in the case of the metallurgy of steel, metallic aluminum is frequently added to the molten steel to combine with the oxygen present and thus prevent its combination with the carbon, the oxide of aluminum being solid and hence taking up far less volume in the steel than that which the gas CO would occupy.

The difference in the volume between a solid oxide like aluminum oxide and a gaseous oxide like carbon monoxide (CO) is well illustrated by the statement made by Prof. J. W. Richards before the Electro-chemical Society recently, in which he called attention to the fact that if a steel contained .001 per cent by weight of oxygen and this combined with some of the carbon present to form CO gas, that the volume of the gas at atmospheric pressure would equal 60 per cent of the volume of the steel; and this clearly shows that the steel would be exceedingly porous.

For deoxidizing any of the more common metals a metal which deoxidizes with greater ease must be used.

The relative amount of heat liberated when a given amount of metal is oxidized is shown by Table I.

TABLE I.

	Calories.
MgO	145.5
CaO	145.0
SiO	131.0
Al ₂ O ₃	131.2
3 SiO ₂	99.9
2 Fe ₂ O ₃	65.7
3 NiO	61.5

The well-known iron thermit reaction which is caused by melting iron oxide and metallic aluminum and then igniting the mixture, is due to the fact that aluminum gives up more than twice as much heat in being deoxidized as iron does, hence as the aluminum is reduced to oxide approximately one-half of the heat is absorbed by reducing the iron, and the remainder is left to superheat the charge and hold the material in the molten state.

If the difficulties resulting from remelting aluminum are due to the oxide of aluminum present in the metal, and we wish to reduce this oxide, we must look for a metal that gives up more heat than aluminum when it is oxidized. As shown by Table I, calcium and magnesium would fulfill these conditions. Calcium cannot now be obtained pure, however, in commercial form at a low price, and consequently magnesium is the metal to which we would naturally turn.

Unfortunately, the oxides of both aluminum and magnesium are exceedingly refractory, and hence solidify immediately while being formed, and no efficient fluxes have been found for removing them from the metal in liquid form. Owing to its exceedingly low specific gravity, however, it is argued that magnesium oxide ought to free itself from aluminum fairly well.

In order to determine the efficiency of magnesium in deoxidizing aluminum, it was decided to make two series of test bars, in one of which metal would be remelted several times without adding any magnesium and in the other it was remelted several times with equal additions of magnesium for each remelting. As the loss in aluminum for each melting was generally over one per cent, it was decided to try an addition of one-half of one per cent of magnesium each time. It was argued that if loss of strength in aluminum was due to oxidation during remelting, that each succeeding cast made without magnesium should be weaker than the preceding, while if exactly a sufficient amount of magnesium had been added to the other pot to overcome oxidation, the test bars in this series should be of equal strength. If they increased in strength it would be evident that an excess of magnesium had been added.

In carrying out the experiments two new No. 40 crucibles were taken. In one there was introduced 25 lbs. of aluminum, and when this was melted one-half of one per cent of magnesium was added, stirred in, the pot allowed to stand about two minutes, and then the metal poured. The test bars were poured in No. 00 Albany sand, four bars being cast in a mold, and the molds poured in horizontal position. The bars were pulled without machining, on account of

*Paper read before the American Brass Founders' Association.

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the fact that the investigation sought to determine the effect of oxidation on castings just as they came from the sand rather than upon machined metal.

After pouring the bars the remainder of the metal was run into chills and allowed to cool. The metal from the chills and the gates and the sprues from the molds were then carefully weighed and returned to the pot. Each time the amount of magnesium added was carefully calculated upon the exact amount of metal returned to the pot, and the proceeding repeated until in the A series, which was the series with magnesium, eight sets of bars had been cast. In the case of the eighth melting the metal was not hot enough to run perfect bars, and so it was weighed and returned to the pot and the ninth melting made. Of course, between the A-7 and the A-9 meltings one per cent of magnesium had been added—that is, one-half per cent at each melting.

When the bars in the A series were pulled, it was found that each succeeding series of bars increased in strength, as shown by the accompanying table and curves.

For the B series of bars 50 lbs. of aluminum was taken, melted in a No. 40 crucible, skimmed carefully, and a series of four bars poured. In this case no magnesium was added for the first eight meltings. In the case of the fifth and seventh meltings only one bar from each series ran perfect.

Beginning with the ninth cast, one-half of one per cent of magnesium was added just before pouring, and this procedure repeated for eight consecutive meltings, making 16 in all for the B series. Great care was taken in all of this work, and a small amount of chloride of zinc placed on the metal when it became molten.

To our surprise, the bars from the B pot did not fall off in strength with the first eight meltings, but, on the other hand, showed a very slight increase in strength. This increase in strength must be due to some impurity introduced into the metal.

Careful analysis for zinc has been made to determine how much zinc had been taken up from the zinc chloride. Of course, there is also a possibility of carbon being absorbed from the crucible, or of metal from the stirring rod.

The results of the first nine sets of bars in the B series certainly seem to indicate that with careful work aluminum can be remelted repeatedly without material loss in strength, and without introducing serious trouble from blowholes. The metal used in the series was made by the Aluminum Company of America, and was what is known as their No. 1 metal, which is supposed to be 99 per cent pure aluminum.

With the addition of magnesium to the ninth bar in the B series, the strength of this set of bars jumped to approximately the same figure as that of the first bar in the A series, and the strength rose steadily with each succeeding melting, except that for some reason which we have not discovered the series of bars from

the B-16 melt are slightly weaker than those from the preceding pot.

The difference between the succeeding casts shows a marked difference in strength and fracture in some cases, as for instance, between the A-3 and the A-4 series, where the strength fell off slightly and the fracture showed a crystalline structure. The A-5 series increased in strength over the A-4 series, but showed a marked tendency toward crystallization in the fracture.

In the B series small flaws were found in several of the bars, though in some cases these did not seem to affect the strength much, as for instance in the B-14 series, flaws occurred in two of the bars, which only seemed to reduce the strength about 300 lbs. In another bar a flaw reduced the strength almost 2,000 lbs.

Careful determinations have been made to ascertain exactly the percentage of magnesium left in the bars after melting, and these results are given in Table III. It will be noticed that some of the magnesium was oxidized, and this undoubtedly reduced

TABLE III—PERCENTAGE OF MAGNESIUM IN TEST BARS AS CAST.

A-1.....	.66	B-9.....	.86
A-2.....	.87	B-10.....	.85
A-3.....	1.32	B-11.....	1.39
A-4.....	1.74	B-12.....	1.75
A-5.....	2.43	B-13.....	2.32
A-6.....	2.43	B-14.....	2.41
A-7.....	2.56	B-15.....	2.64
A-9.....	3.71	B-16.....	2.69

a certain amount of aluminum oxide, and thus probably assisted in strengthening the metal. It would be interesting to make another series of bars using approximately the percentage of magnesium lost in remelting, thus eliminating the increase strength due to increasing the proportions of magnesium in the metal.

It is interesting to note that less than three per cent of magnesium was sufficient to almost double the tensile strength of the aluminum test bars, as seen by comparing tests B-1 and B-15. An examination of the fracture of these bars, however, shows that increasing the percentage of magnesium increased the crystallization and decreased elongation, this latter fact being clearly shown in Table II and the accompanying curves.

When melting the metal it was noticed that the magnesium produced a much cleaner metal, although it had to be poured at a trifle higher temperature than that commonly used for aluminum castings. A pyrometer was not used on the pots, and it is barely possible that the cleaner surface of the metal deceived the operator as to the pouring temperature.

At intervals throughout the series eleven bars were cast in a graphite mold or chill made from a block of Acheson graphite. The results obtained from pulling these eleven bars are given in Table IV, and it will be noticed that the ultimate strength of the series is higher than that obtained by casting the bars in sand, it being evident that the graphite acted to some

extent as a chill. The relative strength of the bars one to another is approximately the same as in the case of the series cast in sand.

As already stated, the metal was melted in a No. 40 crucible, and after the magnesium was added the crucible was stirred with an iron bar, but it is prob-

able that the metal did not take up much if any metal from the iron bar, first, on account of the fact that when the bar was introduced into the metal a small amount of aluminum immediately hardened about it, and the stirring only continued a few seconds after this crust had melted off from the iron, so that very

TABLE II.—SHOWING RESULTS OF TESTS OF ALUMINUM TEST BARS, CAST IN SAND.

Mark.	Diam.	Area, sq. in.	Elastic Limit.		Ultimate tensile strength.		Elongation in 4 in.	Elongation in %	—Reduction in Area.—			Remarks.
			Total load.	Lbs. to sq. in.	Total load.	Lbs. to sq. in.			Diam.	Area.	Pct. reduction.	
A-1	.515	.2083	1800	8630	3325	15970	.44	11.0	.495	.1924	7.5	Coarse granular fracture.
	.520	.2124	1800	8460	3185	15000	.25	6.25	.500	.1963	7.6	Coarse granular fracture.
	.515	.2083	1700	8160	3175	15240	.30	7.5	.485	.1847	11.2	Coarse granular fracture.
	.520	.2124	1800	8160	3150	14810	.26	6.5	.498	.1948	8.2	Coarse granular fracture.
A-2	.520	.2124	1900	8940	3200	15050	.14	3.5	.509	.2035	4.1	Coarse granular fracture.
	.517	.2099	1950	9200	3350	15960	.14	3.5	.509	.2035	2.9	Coarse granular fracture.
	.520	.2124	2000	9340	3275	15400	.16	4.0	.508	.2027	4.5	Coarse granular fracture.
	.517	.2099	1950	9300	3350	15960	.19	4.75	.498	.1948	7.2	Coarse granular fracture.
A-3	.516	.2091	2200	10530	3695	17570	.23	5.75	.515	.2083	0.2	Coarse granular fracture.
	.525	.2165	2100	9700	3750	17320	.24	6.0	.513	.2067	4.4	Coarse granular fracture.
	.520	.2124	2000	9430	3365	15850	.17	4.25	.505	.2003	5.6	Coarse granular fracture.
	.515	.2083	2200	10550	3650	17520	.17*	4.25	.502	.1979	5.0	Coarse granular fracture.
A-4	.520	.2124	1700	8000	3420	16100	.15*	3.75	.520	.2124	0.0	Break at fillet.
	.524	.2156	1700	7900	3215	14920	.12*	3.00	.534	.2240	...	Break at fillet partly crystallized.
	.515	.2083	2000	9610	3400	16320	.20	5.00	.507	.2019	3.1	Break at fillet partly crystallized.
	.525	.2165	2100	9700	3250	15080	.08*	2.00	...	...	...	Break at fillet partly crystallized.
A-5	.520	.2124	2000	9430	3620	17010	.17*	4.25	.507	.2019	5.0	Break at fillet partly crystallized.
	.517	.2099	1900	9060	3490	16620	.12*	3.00	...	...	...	Break at fillet partly crystallized.
	.519	.2116	1900	8980	3425	16200	.14*	3.50	...	...	...	Break at fillet partly crystallized.
	.528	.2190	2100	9600	3275	14950	.08*	2.00	...	...	...	Break at fillet partly crystallized.
A-6	.520	.2124	1900	8950	3520	16550	.15*	3.75†	...	...	...	Break at fillet partly crystallized.
	.520	.2124	1700	8000	3375	15880	.14*	3.50†	...	...	...	Break at fillet partly crystallized.
	.517	.2099	1900	9060	3365	16040	.10*	2.50	...	...	...	Break at fillet, one-half crystallized.
	.515	.2083	2000	9610	3325	16000	.11*	2.75	...	...	...	Break at fillet, one-half crystallized.
A-7	.513	.2067	2300	11130	4825	23330	.31	7.75	.498	.1948	5.7	Blue-gray, finer structure at break.
	.525	.2165	2500	11530	4680	21620	.25	6.25	.502	.1979	8.5	Blue-gray, finer structure at break.
	.512	.2059	2350	11420	4340	21080	.19	4.75	.515	.2083	...	Blue-gray, finer structure at break.
A-8*	.525	.2165	2700	12470	5270	24320	.28	7.00	.493	.1909	11.8	Blue-gray, finer structure at break.
	.522	.2140	2700	12620	4900	22900	.22	5.50	.498	.1948	8.8	Blue-gray, finer structure at break.
	.525	.2165	2800	12920	5225	23200	.23	5.75	.505	.2003	7.4	Blue-gray, finer structure at break.
	.520	.2124	2500	11770	4965	23370	.22	5.50	.501	.1971	7.1	Small flaw.
B-1	.516	.2091	1300	6220	2035	9740	.32	8.00	.475	.1772	15.2	Break in fillet. Flaw 1/8 inch square.
	.513	.2067	1200	5810	2220	10730	.48	12.0	.445	.1555	24.7	
	.520	.2124	1100	5180	2110	9940	.27*	6.75	.455	.1624	23.5	
	.514	.2175	1300	6270	2135	10280	.29*	7.25	...	...	...	
B-2	.513	.2067	1200	5810	2165	10470	.39	9.25	.475	.1772	14.2	
	.522	.2140	1300	6080	2460	11480	.59	14.75	.436	.1493	30.2	
	.519	.2116	1200	5680	2535	11990	.85	21.25	.409	.1314	37.8	Slight flaw.
	.515	.2083	1200	5760	2475	11870	.63	15.75	.436	.1493	28.3	
B-3	.514	.2075	1150	5540	2310	11120	.40	10.0	.440	.1521	26.7	
	.515	.2083	1200	5760	2250	10800	.51	12.75	.445	.1555	25.4	
	.512	.2059	1100	5340	2295	11140	.48	12.00	.475	.1772	13.8	
	.517	.2099	1000	4765	2240	10670	.48	12.00	.478	.1794	14.4	
B-4	.519	.2116	1100	5200	2260	10680	.51	12.75	.463	.1684	20.3	
	.521	.2132	1300	6100	2780	13040	.65*	16.25	.465	.1698	20.3	Break in fillet.
	.515	.2083	1150	5520	2430	11660	.40	10.0	.452	.1605	23.0	
	.515	.2083	1000	4810	2520	12080	.56	14.0	.430	.1452	30.2	
B-5	.525	.2165	1100	5080	2800	12920	.80	20.0	.440	.1521	29.7	
	.525	.2165	1050	4850	2795	12900	.77	19.75	.430	.1452	32.9	
	.520	.2124	1100	5180	2575	12220	.55	13.75	.442	.1534	27.7	
	.530	.2206	1000	4545	2875	13030	.80	20.0	.410	.1370	40.2	
B-6	.514	.2075	950	4575	2630	12680	.71	17.75	.426	.1425	31.2	
	.518	.2107	1000	4750	2700	12800	.69	17.25	.432	.1466	30.3	
	.525	.2165	1000	4620	2715	12540	.86	21.5	.403	.1276	41.1	
	.521	.2132	1100	5170	2640	12380	.55*	13.75	.467	.1713	19.5	
B-7	.523	.2148	1000	4660	2610	12150	.60*	15.0	.454	.1619	24.6	
	.527	.2181	1100	5046	2740	12560	.94	23.5	.448	.1576	27.7	
	.530	.2206	1100	4980	2700	12230	.94	23.5	.395	.1225	44.4	
	.527	.2181	1800	8250	3400	15630	.32	8.0	.492	.1901	12.8	
B-8	.523	.2148	1700	7920	3200	15330	.31	7.75	.485	.1847	14.0	
	.520	.2124	1500	7070	3380	15880	.35	8.75	.487	.1863	12.2	
	.518	.2107	1800	8540	3410	16170	.35	8.75	.478	.1794	14.8	
	.520	.2124	1900	8950	3900	18360	.34	8.5	.498	.1948	8.2	
B-9	.525	.2165	2000	9240	3690	17030	.26	6.5	.512	.2059	4.9	
	.524	.2156	1900	8850	3955	18350	.32	8.0	.491	.1893	12.1	
	.528	.2190	2000	9140	3970	18110	.31	7.75	.495	.1924	12.1	
	.528	.2190	2000	9140	3515	16060	.16*	4.0	.525	.2165	5.8	Poorly filled mold.
B-10	.522	.2140	2000	9350	3625	16930	.20	5.00	.493	.1948	4.4	
	.538	.2273	1950	8580	3830	16850	.17*	4.25	...	...	...	Break in fillet one-half crystallized.
	.528	.2190	2100	9600	3615	16500	.16*	4.0	...	...	...	Break in fillet one-half crystallized.
	.527	.2181	2050	9400	4115	18870	.27	6.75	.505	.2003	8.0	
B-11	.537	.2265	2100	9280	3270	14430	.10*	2.5	.525	.2165	4.3	Blowhole 1/8 inch diameter.
	.520	.2124	2000	9430	4025	18920	.28	7.0	.495	.1924	9.4	
	.528	.2190	2000	9140	4085	18660	.28	7.0	.501	.1971	9.9	Slight flaw.
	.518	.2107	1850	8780	3900	18500	.24	6.0	.498	.1948	7.5	
B-12	.525	.2165	2000	9240	3920	18100	.26	6.5	.499	.1956	9.7	
	.534	.2240	2100	9390	3975	18570	.23	5.75	.524	.2156	3.7	
	.525	.2165	1800	8550	4040	18630	.26	6.5	.510	.2043	5.5	
	.528	.2190	2170	9820	4000	18270	.21*	5.28	...	...	...	Break in fillet. Slight flaw.
B-13	.528	.2190	2300	10520	4000	18270	.22	5.5	.517	.2099	4.1	1/8 inch diameter. Slag.
	.528	.2190	2250	10270	3590	16400	.13*	3.25	.514	.2075	5.2	1/8 inch diameter. Slag.
	.532	.2223	2000	9000	4150	18650	.28	7.00	.518	.2107	5.2	
	.530	.2206	2400	10870	4495	20350	.28	7.00	.520	.2124	3.5	
B-14	.530	.2206	2300	10420	4410	19950	.27	6.75	.505	.2003	9.1	
	.529	.2198	2300	10470	4435	20140	.26	6.5	.510	.2043	7.0	
	.534	.2240	2300	10260	4180	20000	.28	7.0	.507	.2019	9.2	
	.532	.2223	2300	10340	4490	18830	.19	4.75	.505	.2003	9.7	
B-15	.547	.2350	2300	9790	4380	19050	.16*	4.0	...	...	...	Break in fillet. Slight flaw.
	.532	.2223	2300	10340	4375	19360	.23	5.75	.510	.2043	8.0	
	.528	.2190	2400	10960	4365	19940	.18	4.5	.510	.2043	6.6	

*Indicates break was outside of the 4-inch length. †These bars did not run. ‡Yellow spots in fracture.

little time was given for the absorption of iron.

The tests were made as nearly under shop conditions as possible, but as they had to be completed at the end of the day's run, many of the melts were made after dark, and for this reason it was somewhat difficult to judge the pouring temperature accurately, and it is probable that the variations in strength were due as much to variations in pouring temperatures as to any other cause. It is always fairly difficult to judge the proper pouring temperature of aluminum alloys with which one is not thoroughly familiar, and

in this case, and hence has complicated the results. The experiments have certainly proved that very small percentages of magnesium greatly increase the strength of aluminum, and also improve its mechanical qualities.

More work should be done to test fully the various good properties which are imparted to aluminum by the use of magnesium, and it is to be hoped that any of the members of the Association who have had experience along this line will discuss the paper and give the results of their experience.

Mark.	Diam. Inches.	Area, Sq. in.	—Elastic Limit.—		—Ult. T. Strength—		Elongation in 2 in.		Reduction in Area.		Per cent.
			Load.	Lbs. per sq. in.	Load.	Lbs. per sq. in.	Inches.	Per cent.	Diam.	Area.	
A-1	.519	.2116	2300	10870	3900	18430	.46	23.0	.405	.1288	29.0
A-2	.513	.2067	2300	11070	4010	19380	.26	13.0	.459	.1655	29.0
A-3	.521	.2132	2500	11730	4360	20450	.27	13.5	.441	.1527	28.3
A-4	.525	.2165	2200	10160	3850	17780	.16	8.0	.495	.1924	11.1
A-5	.509	.2035	2500	13750	4300	21150	.10	5.0	.458	.1648	18.8
A-6	.506	.2011	2400	11930	3800	18890	.15	7.5	.456	.1633	18.7
A-9	.506	.2011	2900	14410	4050	20140	.09	4.5	.473	.1757	12.6
B-1	.513	.2067	1000	4840	2510	12140	.67	33.5	.347	.0946	54.25
B-2	.510	.2043	1200	5880	2535	12400	.59	29.5	.356	.0995	51.3
B-3	.506	.2011	1000	4980	2500	12420	.61	30.5	.355	.0990	50.8
B-5	.503	.1987	1000	5030	2585	13000	.40	20.0	.430	.1452	26.8

Note. Bars A-3, A-5 and B-5 broke outside of the punch marks.

for this reason the magnesium alloy was particularly difficult to judge.

It seems probable that when the zinc chloride is added to the surface of the metal that the chloride is broken up, thus giving free chlorine and nascent zinc. Had metallic zinc been added to the alloy, the atoms would all be in molecular combination, but by pre-

This paper has only been made possible by the co-operation of a number of parties. The magnesium was furnished by the C. W. Leavitt Co. of New York City, the aluminum was supplied by the Lumen Bearing Co. of Buffalo, New York, and they also did the melting and cast all of the bars. The test bars were all pulled by Hugh D. Pallister, Assistant Instructor

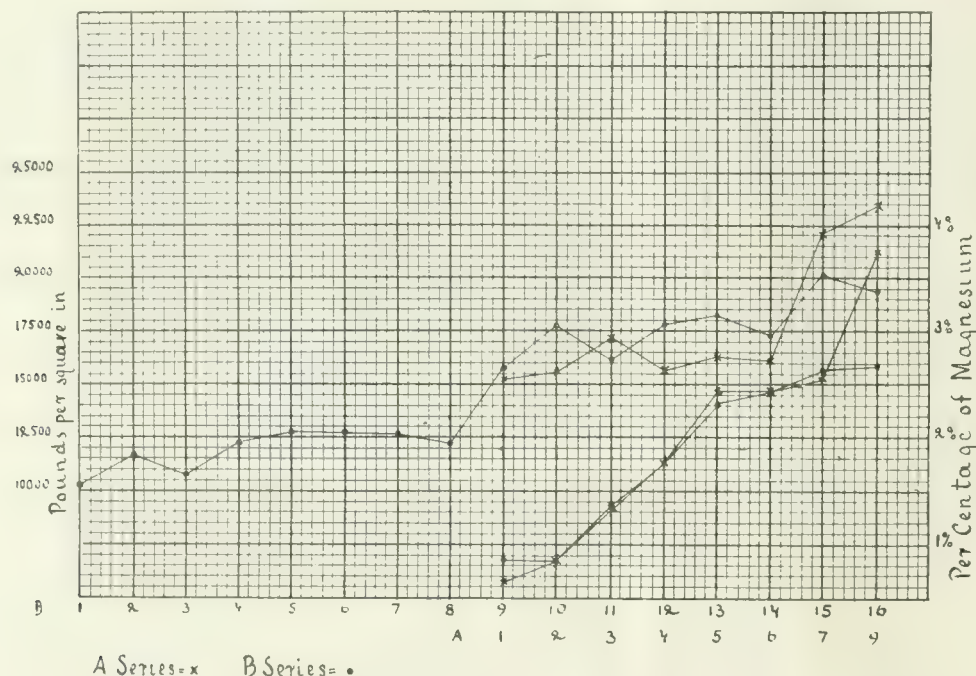


Fig. 1—Curves Showing Strength of Magnesium Test Bars and Percentage of Magnesium.

The figures at the left represent lbs. per square inch and those at right percentages of magnesium, while at the bottom are given the number of the bars of the two series.

senting them to the molten aluminum in the nascent or atomic state, it may be that they have the power of reducing aluminum oxide. It would take exceedingly delicate chemical work to prove this, but if it is true, it gives an explanation of zinc chloride as a purifier of aluminum. It was not the writer's intention that zinc chloride be used in the pots, but as it was the regular practice of the melter, it was followed

in Metallurgy at Case School of Applied Science, Cleveland, Ohio. Great care was taken in this work, and sufficient time given to the tests to insure reliability and accuracy. The analyses were made by the Detroit Testing Laboratory, Detroit, Mich. The determinations for magnesium were run with particular care, and it is very difficult indeed to account for the fact that in the first bar of each series containing mag-

nesium the analytical results showed more magnesium than was supposed to have been added. The only explanation is that there must have been some segregation of the magnesium, and it is possible that it tended to rise toward the top of the crucible, so that the bars that were cast from the first metal poured from the crucible would naturally receive more than their proportionate share of magnesium. The writer has seen this same result in other cases. It is a common experience in making alloys to find that at the first melting a metal which has been added will not be thoroughly incorporated with the alloy, and it is only after the second melting that the different ingredients blend together thoroughly. Then, too, the addition of a strange metal to a combination does not seem to form a homogeneous alloy at first, but if the main body of

zinc chloride, thus supporting the theory of the reduction of aluminum oxide by nascent zinc.

For sampling the bars they were placed in the milling machine and one-half of the thickness of the head of the bar milled from each end of the series of four bars. These chips were then thoroughly mixed and the sample taken from them. The sampling was done at the plant of the Russell Wheel & Foundry Co. through the courtesy of A. T. Waterfall.

The Canadian coal-mining industry was marked during 1909 by a decreased production in Nova Scotia and an increased production in the western provinces, resulting in an aggregate decrease for the whole of Canada of 3.5 per cent. The total production in 1909 was

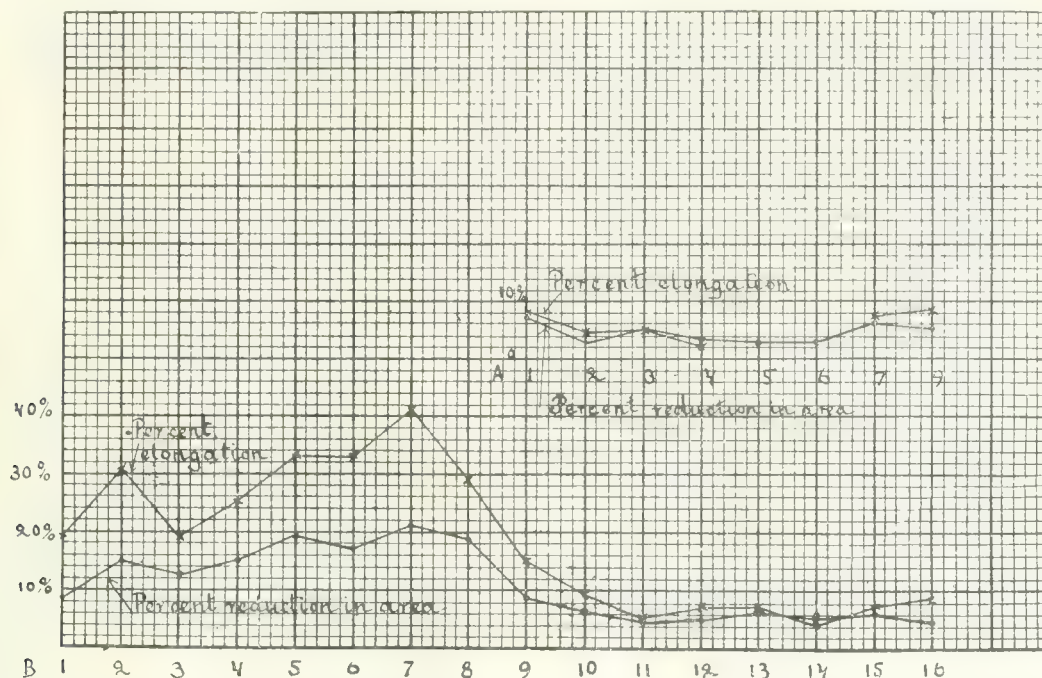


Fig. 2—Curves Showing Percentage in Reduction in Area and Elongation of Bars.

The lower diagram shows the results in B series. The figures at the left representing percentages and those at the bottom numbers on bars. The diagram in upper right hand corner represents the results in the A series in which case the percentage of reductions in area and elongation were almost identical. In the case of bars 5 and 6 a break occurred outside the punch marks so it was impossible to get a percentage of reduction in area.

the metal already contains some of the metal to be added, the blending seems to take place more rapidly and more uniformly. It is also barely possible that in the case of the first set of bars some of the magnesium oxide had not freed itself from the metal and passed into the bars. This would appear in the samples and be reported as magnesium even though it might in part represent metal which had been active in deoxidizing the aluminum in the lower part of the crucible.

As already stated, when it was discovered that in the B series of bars the strength increased slightly in spite of remelting without the addition of magnesium, careful determinations were made to ascertain the amount of zinc present in the aluminum. In the B-1 series of bars there was found .015 per cent of zinc, while in the B-8 series of bars there was found .085 per cent of zinc. Both of these are relatively small percentages, but the last is nearly six times the first, and both show that zinc passed into the metal from the

returned as 10,501,475 tons, valued at \$24,781,236, compared with 10,886,311 tons, valued at \$25,194,573, in 1908. Coal mining has been for a number of years the most important of Canada's mining industries, and in 1909 is credited with 27 per cent of the total mineral production of the country. The coal comprises anthracite, bituminous, and lignite, and 90 per cent of the production may be roughly classed as bituminous. In 1909 about 82.6 per cent of the output was placed directly on the market, 7.1 per cent made into coke, and 8.8 per cent used in colliery consumption and by workmen. The United States constitutes the main market for coal exported, 78 per cent of the shipments being sent to that country in 1909. The consumption of coal in 1909 amounted to 18,625,202 tons, a decrease of 3.76 per cent from the preceding year, and of this total 52.1 per cent was imported and 47.9 per cent domestic coal. The per capita consumption was approximately 2.599 tons.

THE PARAGON ELECTRIC FURNACE.*

By J. HARDEN.†

The electric steel furnace appears to be steadily gaining ground in this country and among others a 1½-ton Kjellin induction furnace has recently been erected in Sheffield, for crucible steel work, and has successfully completed its guarantee run. This furnace is especially adapted for producing the highest grade of tool steel. It is, however, well known that a furnace of this type is not adaptable for any extensive refining of steel, owing to the refining surface of the bath being small.

It has long been recognized by steel makers generally that, although the induction furnace is an excellent tool in the hands of the crucible steel maker, a furnace which would permit more extensive refining, and also the construction of large units, at the same time keeping the advantages embodied in the induction furnace system, is much to be desired. It is because of the comparatively low power factor of larger furnaces of the plain induction type that this class of furnace has not been constructed in large units to any great extent.

In order to overcome this, the Röchling-Rodenhauser furnace, which is partly a resistance furnace and partly an induction furnace, was designed. A considerable number of these furnaces have been erected and are in operation abroad, giving very good results. Many steel makers, however, are of the opinion that it would be an advantage if a furnace could allow superheating the slag for refining purposes, but at the same time enable the metallurgist to so regulate the heat of the charge that gases could be expelled without difficulty. At the same time, the perfect circulation of the bath should be maintained so as to secure a perfectly uniform material.

A new furnace on these lines, which the inventors call the "Paragon" furnace, has been designed and protected by patents, which appears to possess points in its favor. As is shown in Figs. 1, 2 and 3, this furnace is a combination of an arc and a resistance furnace, which makes it very much like an open hearth furnace, and it may indeed be worked exactly in the same manner. The slag blanket is kept at the desired temperature by means of arcs playing over the surface of the slag, while the remainder of the heating power is transmitted to the bath by means of terminal plates, such as are used in the Röchling-Rodenhauser furnace.

The result of this combination is that the slag may be heated to any desired temperature while dephosphorization and desulphurization take place, but the heating effect of the arcs need not be increased to such an extent as to keep the whole charge superheated, which, of course, would tend to overheat the steel im-

mediately beneath the arcs, but a carefully regulated amount of current can be sent at the same time through the resistance plates, thus also heating the charge from underneath.

When the dephosphorization and desulphurization are finished, it may be desirable to alter the gradient of heat in the furnace, so that the lower portion of the bath is hotter than its surface, in order to expel the gases more readily. This result is obtained by diminishing the power transmitted through the arcs and increasing the power transmitted to the side plates. In fact, it is even possible to extinguish the arcs for a time and keep the bath liquid by means of the side plates, which for this purpose are provided with a cooling device of enforced air. The metallurgist has, therefore, the power of controlling the gradient of heat in the bath at will.

This combination has another advantage, viz., that

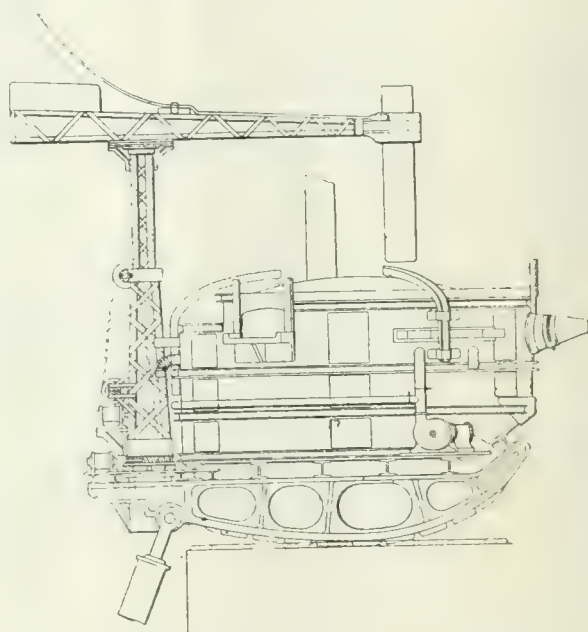


Fig. 1—Diagram of the Paragon Electric Furnace.

the electrodes do not require to be of such large section for a given capacity of furnace as would be the case if the total amount of power were transmitted by means of the arcs only, owing to the fact that normally about 50 per cent of the total power is transmitted through the arcs, while the other 50 per cent is transmitted through the side plates. The advantages claimed, therefore, are threefold:

1. The size of electrodes for a given furnace capacity is only about half that of an arc furnace, and consequently with a given electrode section available the melting capacity of the furnace can be nearly doubled.

2. Owing to the destructive effect of large arcs immediately under the roof of the furnace, the roof very quickly deteriorates, but in the case of this new furnace the roof is not subjected to such severe strain, owing to the reduced size of the electrodes and to the

*From the Iron and Coal Trades Review, London, England.

†Of the Gröndal Kjellin Co., London, England.

smaller amount of power transmitted through the arcs; therefore, the maintenance of the furnace costs less, and the roof lasts longer.

3. The control of the load factor of the furnace is better, because the current flowing through the resistance plates is a perfectly steady one, whereby the

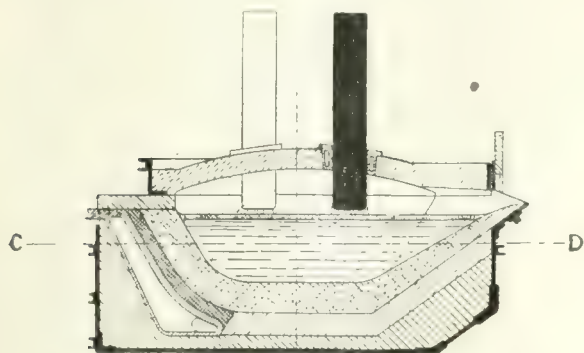


Fig. 2—Section of Furnace.

heavy current fluctuations otherwise caused by large arcs is very extensively compensated.

The result of this is that no special machinery is required either for compensating for the lower power factor of a true induction furnace or designed with a large fly wheel effect, as in the case of a simple arc furnace, on account of its fluctuating load factor.

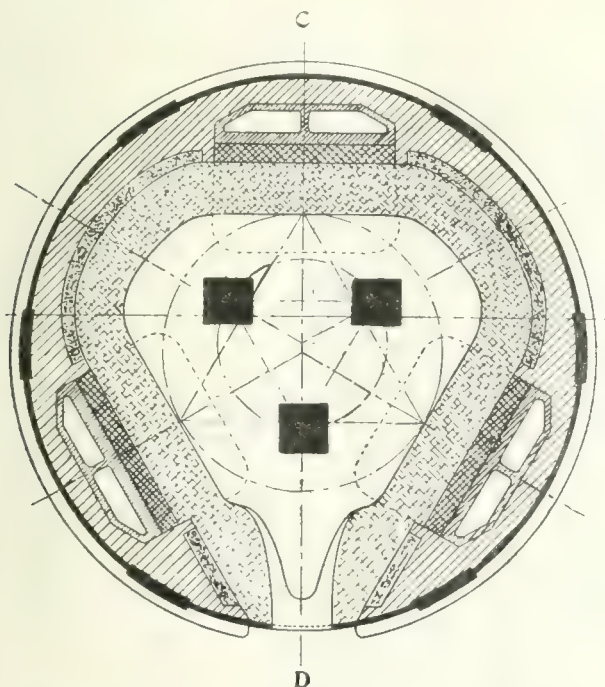


Fig. 3—Horizontal Section of Furnace.

Standard machinery and standard transformers only appear to be necessary, which, of course, decreases the first outlay for the plant, and under the circumstances may prove to be almost a deciding influence.

The power consumption per ton of steel should also prove more economical by means of this combination, as the high temperature of the steel is only required

during desulphurization, it being well known that dephosphorization does not require such a hot slag as desulphurization, and it follows as a logical sequence that the higher the temperature of the steel for any length of time, the greater also the heat losses during the process; in view of this, the total melting and refining process should be more economical with this combination. It therefore appears that this type of furnace includes several of the advantages of the induction furnace, viz., a good circulation of the charge, inasmuch as it is heated from below as well as from above, and a high load factor.

No electrical apparatus, in the ordinary sense of the word, seems to be an element of the design of this furnace, because the power supply transformers may be situated in an adjoining building protected from dust and injury, and entirely beyond the reach of the workmen. The low-voltage current only requires to be transmitted to the furnace by means of insulated busbars and cables, which is easily arranged.

The regulation appears to be simple, as if the furnace is operated direct from a generator, the field rheostat controlling the current to the generator is all that is required; or, if transformers are used in connection with an existing power supply, so-called regulating transformers with interchangeable terminal steps are employed. In fact, comparatively little regulation in this respect should be found necessary, because the current in the arcs ought to be easily regulated by altering the length of the arcs, while the current in the resistance plates is more or less automatically regulated by the condition of the charge.

The Stavanger Electro-Staalverk Co. is being organized in Norway, with a capital of \$120,600, to establish a plant for making high-grade steel electrically, using a process similar to that now used in Domnarfvet, Sweden, and elsewhere. The new company plans to take over the plant of the Stavanger Skibsofhugnings Co., and continue purchasing and breaking up old steel and iron vessels. The company will also secure raw materials from other sources and install modern machinery for making fine quality steel. The new plant is being located at the nearby village of Jorpeland, where electric power has been secured at \$6.70 per horsepower per year for the first 1,500 hp. and \$5.35 for 1,000 more horsepower delivered at the power shaft; 10,000 horsepower will be available to the company when needed. It is announced that the plant will be completed by August of this year. The full yearly capacity will be 1,400 tons high-grade steel billets, 600 tons steel castings, 300 tons hammered steel, and 700 tons refuse for resmelting. The machinery and buildings to be constructed will include smeltery, generator, furnace, ovens, crane, and building, costing about \$42,410; steel foundry, compressor, furnace, saws, pneumatic and other tools, and building, \$16,215; rolling works, with building, \$26,800; forge, with attachments, and building, \$3,215.

ELECTROLYTIC IRON.*By **CHARLES F. BURGESS.†**

During the past five years investigations have been carried on in the chemical engineering laboratories of the University of Wisconsin, dealing with electrolytic refining of iron and its use in the production of alloys. About 3 tons of iron have been refined and over 1,000 alloys produced and tested. While this experimental work has been on the "test tube" or laboratory scale, it is believed that some of the results indicate the feasibility of enlargement to commercial state, and a short discussion is submitted here bearing on the question, "Can electrolytic iron be made an industrial product?"

Many old textbooks on electro-chemistry give descriptions of methods of depositing iron electrolytically, implying, therefore, that electrolytic iron is not a new product. The only use suggested, however, is for facing engraved plates used for printing. By following any of these earlier processes it is seen that only very thin layers of iron can be deposited, the coating tending to become dark, rough and powdery. It is to overcome this limitation that the work at the University of Wisconsin was undertaken, and that this has been successful, in a measure, is indicated by an iron cathode nearly 2 in. in thickness.

The manner in which electrolytic refining has revolutionized the copper industry is now an old story, and electrolysis has likewise become an industrial agent in the metallurgy of silver, gold and lead. Now, that it has been found that iron can be refined electrolytically almost as easily as copper, it is pertinent to inquire whether there is a field of usefulness for such iron. This depends upon whether electrolytic iron has superior qualities due to purity or physical condition, and whether it can be procured at low cost.

From the standpoint of purity it must compete with some notable metallurgical developments which have resulted in product such as being made by the American Rolling Mill Co. in a basic open hearth process. For this material a purity of 99.94 per cent is claimed and justifiably so, as some of our analyses show. The electric furnace is another factor which is contributing largely to the control of the composition and purity of iron alloys, and the striking possibilities which are offered in this developing field cannot be overlooked in advocating the development of an electrolytic iron process.

The chief source of commercially pure iron has been and, perhaps, now is, the high grade Swedish and Norway iron, used largely as a base material for high grade crucible steel. The analyses of this material usually indicate a high purity, though frequently and erroneously calculated by difference, after determining carbon, sulphur, silicon, phosphorus and manganese. It

is not uncommon to find oxides and slag to the extent of 2 per cent in this material, and this impurity undoubtedly may have an influence on the resulting alloys made from it.

We have found that electrolytic iron can be produced with a purity as high as 99.97 per cent, and, perhaps, even better, using extraordinary precautions. This record has been made by using the best commercial grades of pure iron as anodes. A few hundredths of 1 per cent of purity must be sacrificed in using anodes of mild steel or other less pure materials. Electrolytic refining offers a means of reducing or screening out most of the impurities commonly found in iron and of producing a material not only of high purity, but of great uniformity. Even though it may be shown eventually that electrolytic iron may not have a higher purity than that attainable by other methods, the uniformity of its composition should make it a valuable material as a means of eliminating many of the variables with which the crucible steel maker has to contend.

Using an electrolyte containing 40 g. of iron per liter in the form of ferrous sulphate, together with 40 g. of ammonium chloride, it has been found possible to conduct a continuous refining operation for many months at a current density of 6 to 10 amperes per square foot of cathode surface, and at a potential difference of about 1 volt. The current efficiency is very close to 100 per cent, as it is in copper refining.

This leads to the calculation that 1 kw.-h. will refine 2 lbs. of iron; or a cost for power of $\frac{1}{2}$ ct. per pound is attainable. The costs for labor, solution maintenance and fixed charges are estimated to be not greater than the power costs, making a cost of refining of about \$20 per ton. Assuming the anode material to be a mild steel costing \$35 per ton, the cost of electrolytic iron would be in the neighborhood of \$55. These approximate calculations indicate that this material might well compete in price with high grade Swedish iron.

Among the properties of electrolytic iron which may give it some added usefulness is its content of hydrogen and the brittleness which results from this occluded gas. This hydrogen may be of some service in reducing oxides in a melt. The brittleness of the electrolytic iron before the hydrogen is driven off makes it easy to break it up into pieces suitable for introduction into the steel crucibles.

The United States consul at Bergen, Norway, reports that much progress is being made in that country in the smelting of iron ore by electricity. The new electric smelting works at Odda, Hardanger, recently started, will be greatly enlarged early this year; a new electric smelting factory with a capital of \$268,000 will soon be in operation at Tinfos; and sufficient capital has already been subscribed to insure the building of a third factory at Jorpeland, near Stavanger.

*Paper read before the Chicago Section of the American Electrochemical Society, Jan. 20, 1911.

†Professor of chemical engineering University of Wisconsin.

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NOTES AND COMMENTS.

Laboratory Efficiency.

The technical as well as the public press has been filled recently with descriptions of and comments on the work of Mr. Taylor and his associates in pointing out the waste of time and energy in many of the trades and professions and giving his methods for eliminating this waste. It seems to the writer that such methods of investigation applied to work in the chemical laboratory would yield very beneficial results. Any one who has observed such operations as the transfer of precipitates from beakers to filter paper, the filtration of said solutions, with washing and final ignition of precipitate, realizes the immense amount of time often wasted in performing these operations which in many cases are fundamental for chemical analysis.

The writer will only venture to suggest some of the points which have occurred to him by which time can be saved and will leave to the individual chemist the application of these suggestions.

The use of the rubber finger cot instead of the customary policeman will be found a great aid in removing an adhering precipitate from a beaker.

Much time can be saved in filtration by the proper selection of the filter paper to meet the demands of

the precipitate to be obtained. Thus use rapid paper for all precipitates not readily passing through and use the slower paper only when absolutely necessary. It is further suggested that a great saving of time can be made in filtering through any paper by properly fitting the moistened paper to the funnel, noting the flow of water through the paper. If the water does not pass through with sufficient rapidity it is cheaper to discard this paper and employ another one.

The customary method of washing a precipitate, using a wash bottle can be greatly improved by using a water supply located above the work table and connected to the washing tip by a piece of pure gum tubing of sufficient length to reach all of the precipitates to be washed.

The ignition of precipitates is a subject which will well repay some study. As an example the writer has often seen laboratory workers ignite carbonaceous materials over a gas flame where the gases of combustion completely surround the crucible containing the material. The suggestion that combustion requires not only initial heating but also oxygen is here quite in order. It is certain that much time on this operation can be saved by the proper use of such appliances as the Hempel furnace or a gas or electrically heated muffle.

The writer is sure that there are many other places where time can be saved other than those here enumerated. We will be very glad to publish at any time notes from our readers covering such mechanical appliances in use in their laboratories which are time savers or any methods of procedure in analytical work which are improvements on customary practice.

The Diamond Match Company Relinquishes Patents.

The action of the Diamond Match Company in returning their patent papers covering the use of phosphorous penta-sulphide in the manufacture of matches has recently been commented on very favorably. The way is thus opened for the passage of a bill now before Congress prohibiting the use of white phosphorus in match manufacture. The Outlook in a recent issue comments as follows on their action.

"The great corporations of this country have been subjected, often justly, sometimes unjustly, to severe criticism; it is all the more grateful, from time to time, to record instances of unselfish and humane action by such corporations. A notable instance of this has just occurred. The Diamond Match Company, which, we understand, is the largest manufacturer in the United States in this particular industry, has, of its own accord, abandoned legal patents which enable it exclusively to manufacture a kind of match the making of which is not an injury to the employe. To understand this matter, it should be known that there have been two kinds of phosphorus used in the making of matches. One is harmless; the other is a poison. A bulletin of the Bureau of Labor gives a really heart-rending account of the evils of white-phosphorus pois-

oning. In Great Britain and in other countries the subject was thoroughly investigated years ago; in the year 1899 alone there were reported in Great Britain one hundred and two cases of phosphorus poisoning, nineteen of which were known to have terminated fatally. The common term for the unique malady which afflicts men and women engaged in match manufacture is 'phossy jaw'; the scientific term is phosphorus necrosis. It is a peculiarly horrible and distressing disease. Other countries have protected their workmen against this danger, and it is earnestly to be hoped that the United States, by prohibiting the use of white phosphorus (as was recommended by President Taft in his last annual message, and as is proposed in a bill now before Congress), may make this loathsome disease a thing of the past. In the meantime the action of the Diamond Match Company, even although it may be in some measure dictated by enlightened self-interest, is, nevertheless, unselfish and to be praised cordially. The Company held a patent under which an acceptable substitute for the kind of match in which white phosphorus is used could be made. The Company at first assigned the patent to three trustees, empowered to grant licenses to all who wished practical use of it on fair and reasonable terms. The names of these trustees—Professor Seligman, of Columbia; Mr. Neill, the United States Commissioner of Labor; and Mr. Ralston, counsel for the American Federation of Labor—were a guarantee of fair dealing. But later, as this course endangered the bill before Congress, because Congress thought it ought not to compel the use of a privately owned patent (which would follow if the use of white phosphorus was forbidden and the patent for the other method was retained by the Company even nominally), the Company formally renounced the ownership of the patent and dedicated it to the free use of the people of the United States. Thus the way is made clear for humane legislation by Congress."

RECENT PATENTS.

The following patents relating to industrial and engineering chemistry are reported by C. L. Parker, solicitor of patents, McGill Building, Washington, D. C.:

974,131. (*Gelatin Composition.*) Eduard Fr. Felsing, Munich, Germany, assignor to Maria Wilhelmine Felsing. Patented Nov. 1, 1910.

974,134. (*Tire Heating Compound.*) Henry S. Griswold, Phoenix, Arizona. Patented Nov. 1, 1910.

974,138. (*Method of Facilitating and Accelerating the Hardening of Hydraulic Cements.*) Ludwig Hatschek, Vocklabruck, Austria-Hungary. Patented Nov. 1, 1910.

974,182. (*Process for Obtaining Olefin or Saturated Acids.*) Isaac King Phelps, New Haven, Conn. Patented Nov. 1, 1910.

974,257. (*Metal Reducing Process.*) Frederick W. Gordon, Philadelphia, Pa. Patented Nov. 1, 1910.

974,285. (*Cellulose for Pyroxylin Compound.*) Paul H. A. Leder, Baltimore, Md., assignor of one-fourth to Alexander E. Hanline, and one-fourth to Ernest E. Quandt, Baltimore, Md.

The process comprises dissolving cellulose of pyroxlyn in acetone and adding sulfur dissolved in chloroform and evaporating to dryness at normal temperature. Patented Nov. 1, 1910.

974,419. (*Process of Making Copper Alloys and the Product Thereof.*) James Naulty and John Scanlin, Philadelphia, Pa. Patented Nov. 1, 1910.

974,440. (*Process for the Destruction of Hairs and Other Extraneous Matter in Floss and Waste Silk.*) Claudius Seignol, Lyon, France. Patented Nov. 1, 1910.

974,608. (*Process for Smelting Ore.*) Frederick T. Snyder, Oak Park, Ill., assignor to Electric Metals Co., Chicago, Ill.

This is a process of making steel which consists in smelting iron ore with a minimum of carbon in an electric furnace having a refractory crucible surmounted by water jacketed walls and electrodes surrounded by material adapted to flux the earthy constituents of the ore, whereby steel low in carbon may be produced and collected out of contact with carbon, and the walls of the furnace saved from corrosion in the absence of a carbon lining. Patented Nov. 1, 1910.

974,633. (*Process of Making Ammonia, Alkyl Ammonia, or Ammonia Bases.*) Henry Spencer Blackmore, Mount Vernon, N. Y., assignor to Robert E. Robinson and Daniel C. Spruance, trustees, New York City, N. Y. Patented Nov. 1, 1910.

974,822. (*Method of Treating Manganese Steel.*) Winfield S. Potter, New York, N. Y. Patented Nov. 8, 1910. This is a method of treating manganese steel previously strengthened and toughened, which consists in reheating such metal to the temperature not above 550° C., then working the metal into its finished form, and finally cooling it.

974,836. (*Typewriting Machine.*) Arthur W. Smith, New York, assignor to Yost Writing Machine Co., Ilion, New York, a corporation of New York. Patented Nov. 8, 1910.

974,900. (*Explosive Compound.*) Hudson Maxim, New York, N. Y. Patented Nov. 8, 1910. The explosive consists of a gelatinated mixture of pyro-nitro-cellulose and tri-nitranisol.

974,921. (*Process of Making Barium Oxid.*) Charles Rollin, Newcastle-upon-Tyne, England, assignor to himself and Hedworth Barium Company, Limited, Newcastle-upon-Tyne, England. Patented Nov. 8, 1910.

974,962. (*Method of Preserving Wood.*) George Hartmann and Oscar Alwin Schwerdtner, Gross Schonau, near Zittau, Germany. The method consists in applying thereto a solution of sodium zincate and sodium chlorid. Patented Nov. 8, 1910.

974,993. (*Process of Making Anhydrous Barium Hydroxid.*) Charles Rollin, Newcastle-upon-Tyne, England, assignor to himself and Hetworth Barium Company, Limited, Newcastle-upon-Tyne, England. Patented Nov. 8, 1910.

975,040. (*Process for Removing Oxygen from Vessels.*) Robert Hopfelt, Cologne-Klettenberg, Germany. Patented Nov. 8, 1910.

975,076. (*Process of Carbonizing.*) Albert F. Rockwell, Bristol, Conn., assignor to the New Departure Mfg. Co., Bristol, Conn., a corporation of Connecticut. Patented Nov. 8, 1910.

975,106. (*Process of Extracting Copper from Ore.*) Wm. L. Austin, Riverside, Cal. Patented Nov. 8, 1910.

975,148. (*Process for Recovering Antimony from Ores and the Like.*) John Roy Masson, Melbourne, Victoria, Australia. This is a process for recovering antimony consisting in subjecting the material to an alkali solution and then precipitating the antimony on aluminum. Patented Nov. 8, 1910.

975,151. (*Formic Acid.*) Max Neumann, Wiesbaden, Germany, assignor to General Chemical Company, New York, N. Y., a corporation of New York. Patented Nov. 8, 1910.

975,217. (*Treatment of Zinc Ores by the Precipitation Process.*) Adolph Desgraz, Hanover, Germany, assignor to Imbert Process Company, New York, N. Y., a corporation of New York. Patented Nov. 8, 1910.

- 975,353. (Stable Mixture for Producing *Hydrogen Peroxide*.) Reinhold Gruter and Heinrich Pohl, Charlottenburg, Germany, assignor to Chemische Werke vorm. Dr. Heinrich Byk, Charlottenburg, Germany, a corporation of Germany. This is a mixture of perborates containing a less quantity of water of crystallization than completely hydrated perborates, solid acid substances and sugar. Patented Nov. 8, 1910.
- 975,370. (Process of Preparing *Manganese-Steel Ingots* for Rolling.) Winfield S. Potter, New York, N. Y. Patented Nov. 8, 1910.
- 975,387. (Method of Treating *Bagasse*.) Thomas J. Hutchinson, Manchester, England. Patented Nov. 8, 1910.
- 975,394. (*Liquid Wax Finish*.) Joseph D. Bryant, Columbus, Ohio. Patented Nov. 15, 1910.
- 975,447. (Process for *Emulsifying Oils, Fats, and the Like*.) Armand Mueller-Jacobs, Huntington, and Reinhard C. F. A. C. Bernhardt, New York, N. Y., assignor to the Arabol Mfg. Co., a corporation of New York. Patented Nov. 15, 1910.
- 975,625. (Process of *Extracting Iron* from Its Ores.) Stephen G. Martin, Chicago, Ill., assignor of one-third to Wm. O. Bartholomew, St. Louis, Mo., and one-third to Edward Schaaf, St. Marys, Mo. This is a process of smelting ores which consists in subjecting a mixture of fluxed ores to the action of heat and a gaseous medium consisting of pure air and nitrogen peroxide. Patented Nov. 15, 1910.
- 975,835. (Process of *Electrolytically Treating Tannic Infusions of Plants*.) Hermann Damkohler and Hugo Schwindt, Bremen, Germany. This is a process of treating tannic infusions of plants, such, for example, as the bark of the mangrove, which consists in subjecting the said infusions to the electrolysis within a cell, in which the tannic infusion forms a bath for cathode which is separated from the anode by a porous partition wall, and in which the baths for the electrodes contain solutions of metal salts, the metals of which are those of the electrodes. Patented Nov. 15, 1910.
- 975,866. (*Formic Acid*.) Henry Howard, Boston, Mass. Patented Nov. 15, 1910.
- 975,867. (Process of Manufacturing *Sublimed White Lead*.) Louis S. Hughes, Joplin, Mo., assignor to Picher Lead Co., Joplin, Mo., a corporation of Missouri. Patented Nov. 15, 1910.
- 975,980. (*Alkaline Battery*.) Wm. Morrison, Des Moines, Ia. Patented Nov. 15, 1910.
- 976,036. (Method of *Unhairing or Tanning Hides and Skins*.) George D. Burton, Boston, Mass. Patented Nov. 15, 1910. This is a method of treating hides or skins by electricity, which consists in first placing the hides or skins in a solution of sal soda, then passing an electric current through this solution and causing the development of a gas which acts upon the fat or oil globules of the hides or skins and causes them to open and discharge their contents in this solution.
- 976,044. (Treatment of Precious *Metalliferous Ores*.) John Collins Clancy, New York, N. Y. Patented Nov. 15, 1910.
- 976,166. (*Stationery Box*.) Wm. A. Gray, Newark, N. J. Patented Nov. 22, 1910.
- 976,319. (*Protective Coating for Carbon Bodies*.) Hermann Viertel, Lichtenburg, near Berlin, and Geo. Egly, Trep-tow, near Berlin, Germany, assignor to Gebrueder, Siemens & Co., Lichtenburg, near Germany. Patented Nov. 22, 1910.
- 976,337. (Method of Preparing *Pigments*.) Frederick M. Beckett, Niagara Falls, N. Y., assignor to Electro Metallurgical Co., New York, N. Y., a corporation of New York. Patented Nov. 22, 1910.
- 976,456. (Method of *Uniting Metals*.) Wm. G. Grey and Wm. Griffith, Pittsburg, Pa. This is a method of uniting metals, such as iron or steel, with copper, brass, bronze, aluminum, or other metal of alloy, which consists in first cleaning the harder metal, subjecting it to a solution of metallic salt and alum, and then applying in molten form the body of softer metal. Patented Nov. 22, 1910.
- 976,520. (Process of Making *Waterproof Portland Cement* and Product.) Maximilian Toch, New York, N. Y., assignor of one-half to Henry Toch, New York, N. Y. Patented Nov. 22, 1910.
- 976,525. (Treating *Sulphides or Sulphates*.) Utley Wedge, Ardmore, Pa., assignor to the Furnace Patent Co., Philadelphia, Pa., a corporation of Pennsylvania. Patented Nov. 22, 1910.
- 976,557. (Process of Producing *Metallic Zinc*.) Oliver D. Dawson, El Paso, Texas. Patented Nov. 22, 1910.
- 976,602. (Art of *Brewing Beer* and Other Liquors.) Frank Rogerson, Stratham, England, assignor to Ozonair, Limited, London, England. Patented Nov. 22, 1910.
- 976,760. (*Pyrophoric Mass*.) Carl Auer Von Welsbach, Vienna, Austria-Hungary, assignor to the firm of Treibacher Chemische Werke Gesellschaft m. b. H., Treibach, Austria-Hungary. Patented Nov. 22, 1910.
- 976,793. (*Fertilizer*.) C. Ellis, Larchmont, New York, N. Y. Patented Nov. 22, 1910.
- 976,975. (*Spring-Attaching Clip*.) Leonard A. Young, Detroit, Mich. Patented Nov. 29, 1910.
- 976,977. (Process for Obtaining *Silk Fabrics*.) Carl Rudolf Bauman, Gavirate, Italy, and Gottlieb Gottfried Diesser, Zurich, Switzerland. Patented Nov. 29, 1910.
- 976,996. (Process of Manufacturing *Refractory Material*.) Geo. Cannan Fludder, Geo. Eric Fludder and Albert Wm. Fludder, Arlesford, Colchester, Eng. Patented Nov. 29, 1910.
- 977,000. (Process for the Production of *Carbon-Black*, Together with Combustible Gas.) Warren H. Frost and Joshua J. Nix, Los Angeles, Cal., said Nix assignor to said Frost. This is a process of producing carbon-black, together with combustible gas, which consists in producing a condition of suction in a combustion chamber, continuously supplying petroleum to said chamber, admitting a restricted quantity of air to said chamber to cause partial combustion of the petroleum, the condition of suction in the chamber being such that the resulting temperature is sufficiently high to substantially break up the tarry matter and cause the unconsumed petroleum to be decomposed substantially into combustion gas and free carbon, drawing the combustible gas to maintain the condition of suction in the combustion chamber, and delivering such combustion gas. Patented Nov. 29, 1910.
- 977,076. (*Explosive*.) Maurice Delvigne, Namur, Belgium. Patented Nov. 29, 1910.
- 977,442. (Composition of Matter for and Method of Generating *Hydrogen*.) Hans Foersterling and Herbert Philipp, Perth Amboy, N. J., assignors to the Roessler & Hasslecher Chemical Co., a corporation of New York. Patented Dec. 6, 1910.
- 977,465. (Process of Treating *Cassava* and Producing Alcohol Therefrom.) Charles C. Moore, Jr., Lexington, Ky., assignor to the Tropical Products Co., Washington, D. C., a corporation of West Virginia. Patented Dec. 6, 1910.
- 977,545. (*Explosive*.) Gershom Moore Peters, Cincinnati, and Milton Fletcher Lindsey, Kings Mills, Ohio, assignor to the Kink Power Co., Cincinnati, Ohio, a corporation of Ohio. Renewed Oct. 20, 1910. Patented Dec. 6, 1910.
- 977,603. (Process of *Brewing Beer* of Low Alcoholic Contents.) Henry E. Deckerbach, Cincinnati, Ohio. Patented Dec. 6, 1910.
- 977,681. (Manufacture of *Artificial Stone* from Slag.) Wilhelm Schumacher, Osnabruck, Germany. Patented Dec. 6, 1910.
- 977,984. (*Graphite Article* and Method of Making the Same.) Frank J. Tone, Niagara Falls, N. Y. Patented Dec. 6, 1910.
- 977,992. (Process for the Recovery of the *Paraffine Ingredients from Cannel Coals* and Other Similarly Constituted Bituminous Materials Without Change in Their Chemical Composition.) Henry Wurtz, Newark, N. J., assignor to American Chemical Education Co., a corporation. Patented Dec. 6, 1910.

977,996. (*Method of Smelting and Refining Copper Ores and Compounds.*) Ralph Bagley, Pittsburg, Pa. The method consists in the following steps: Dissolving or melting ores that contain within themselves oxidizable elements, by submerging in a molten bath of matte and by the heat of oxidation produced by forcing air through the same, and eliminating the silica, alumina and lime; then separating the slag from the low grade matte; then transferring the matte into a converter, oxidizing it therein, adding value bearing silicious ore, and forming thereby silicate of iron slags, and then eliminating residual oxidizable impurities by forcing air into the bath. Patented Dec. 6, 1910.

978,138. (*Printing Paste.*) Joseph Deinet, Elberfeld, Germany, assignor to Farbmanfabriken vorm. Friedr. Bayer & Co., Elberfeld, Germany, a corporation of Germany. Patented Dec. 13, 1910.

978,142. (*Disinfecting Compound.*) Wilhem Dreyfus, New York, N. Y. Patented Dec. 13, 1910.

978,193. (*Process of Making Fertilizer.*) Spencer B. Newberry, Sandusky, Ohio, assignor of one-half to Geo. R. Fishburne, Charleston, S. C.

This is a process of conversion of insoluble phosphate of lime into citrate soluble form by calcination with an amount of lime approximately equal to the phosphoric acid, and an amount of sodium carbonate approximately equal to one-half the phosphoric acid, contained in the phosphate treated. Patented Dec. 13, 1910.

978,211. (*Art of Extracting Metals Electrolytically.*) James Hart Robertson, New York, N. Y. Patented Dec. 13, 1910.

978,239. (*Silver Cleaning and Polishing Paste.*) George Uhl and Wilhelm Schmidt, New York, N. Y. Patented Dec. 13, 1910.

978,282. (*Process of Fermentation for Use in Manufacture of Yeast and Spirit.*) Desider Foster and Philip Finitzer, Budapest, Austria-Hungary. Patented Dec. 13, 1910.

978,443. (*Method of Treating Glycerine.*) Samuel Henry Fleming, Camden, N. J., assignor to E. I. Du Pont de Nemours Powder Co., Wilmington, Del., a corporation of New Jersey.

This is a process of polymerizing or condensing glycerine, which consists in subjecting glycerine admixed with a condensing reagent to heat when under a pressure less than atmospheric. Patented Dec. 13, 1910.

978,679. (*Carborundum Abrasive Article.*) Frank J. Tone, Niagara Falls, N. Y., assignor to the Carborundum Company, Niagara Falls, N. Y., a corporation of Pennsylvania. Patented Dec. 13, 1910.

978,696. (*Process for Separating Rubber or Rubber-Like Substances and Resin from Materials Containing the Same.*) Leon Henri Chaunt, Aubervilliers, France, assignor by mesne assignments to Asia Rubber Company of America, a corporation of Maine. Patented Dec. 13, 1910.

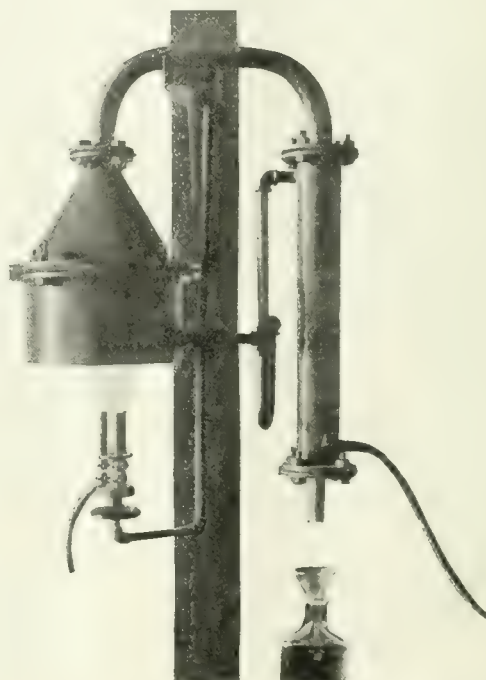
978,747. (*Abrasive Article.*) Francis W. Higgins, Niagara Falls, N. Y., assignor to the Carborundum Company of Niagara Falls, N. Y., a corporation of Pennsylvania. Patented Dec. 13, 1910.

978,814. (*Process of Embalming.*) Carl J. Barnes, Chicago, Ill. Patented Dec. 13, 1910.

978,848. (*Method of Making Portland Cement.*) Joseph Maxwell Carrere, Allentown, Pa., assignor to Blanc Stainless Cement Co., Allentown, Pa., a corporation of New Jersey.

The method consists in subjecting to a relatively low and gradually increasing heat a mix comprising limestone clay rich in alkalies, driving off substantially all the carbonic acid gas and a portion of the alkalies, thereafter subjecting the mix to a higher and clinkering heat produced by a short intense singeing flame and discharging the resulting clinker substantially as soon as formed, the gradual heating being due to the passage of products of combustion of the short flame through the upper part of the kiln.

AUTOMATIC WATER STILL



A serviceable gas distilling apparatus.

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THE BASIS OF QUALITY IN PAPER.*

By ARTHUR D. LITTLE.†

The Germans long ago, after extended investigation, established and have since developed and maintained under government regulations, a system of classification of papers based on their composition and physical properties. This classification or definition of co-called "Normal Papers" has since attained a considerable acceptance in countries other than Germany and has come to be regarded as constituting a series of standards of quality in paper. Valuable as this system of classification has proved itself to be to both users and makers of paper, and indispensable as the refined and accurate testing methods which it involves have become for the intelligent selection of papers, it remains true, nevertheless, that, taken by itself, or as applied to any particular papers without reference to their intended use, it is without significance as to quality. There is, in a word, no such thing as a general basis for quality in paper, and no other possible basis for quality in case of any particular paper than its suitability for its intended use. Good writing paper is bad blotting; good news print is poor wrapping; good book paper is an impossible cigarette paper; and so on. In no case can the properties of good paper be defined until the specific use for which the paper is intended has been stated or assumed. The properties which confer high quality upon a paper intended for one purpose are precisely those which condemn it for another. Stretch is a desirable property for a bag paper, it may be fatally objectionable in paper for lithography; opacity is a principal element in determining the quality of bible paper, transparency is equally important in pergamin and glassine papers. Certain grades of book papers base their claims to quality mainly upon the extent to which they bulk for weight; the buyer of wrapping paper should consider, on the other hand, area for weight in determining quality.

Keeping clearly in mind, then, the conclusion that the basis of quality in paper must always be found in the relation of the properties of particular papers to the intended purpose of these papers, let us consider the more important properties of the material and their relation to particular uses.

In case of a thing so apparently simple in its make-up as a sheet of paper, an extraordinary number of factors are involved in any consideration of its quality. Many of these factors are elusive, in the sense of evading exact definition or determination, but they are none the less important, or even at times decisive, on that account. Such, for example, are permanency, feel, texture, finish, rattle, tear, formation, fastness of color, distinctness and artistic quality of water-marking, dirt, softness, hardness, hairiness, or "whiskers," distinctive appearance, "color" of white papers, printing quality and ability to withstand erasure. As to most of these properties reliance must be placed upon expert judgment and comparison with standard or accepted samples of generally similar papers. Permanency, however, is largely a function of composition, and the composition of a paper can be determined with a considerable degree of accuracy. No paper, therefore, which is intended for permanent records should contain ground wood, unbleached fiber of any kind, acid, sulphur, soluble chlorides, or more than a moderate amount of ash. Preferably, it should be free from resin, and in the absence of conclusive proof of the permanent character of papers made from well prepared chemical wood fibers, preference should be given to all rag papers. Since over-bleaching hastens the deterioration of fibers, extreme whiteness is undesirable in the sheet.

In case of papers for ephemeral uses, permanency has, of course, little bearing upon quality. For printing upon fast perfecting presses, newspaper is better paper because of the large proportion of ground wood it contains, and to which it owes in great measure its special printing quality.

Fastness of color, while not permitting of exact quantitative determination, can, nevertheless, be tested experimentally with sufficient accuracy for most purposes, while very definite conclusions may often be drawn from the known degree of fastness of the dyes or pigments from which the paper derives its color. Fastness of color is of comparatively little importance as affecting the quality of papers intended to serve a temporary purpose. It becomes of prime importance in wall papers, etching, photograph mounts, plate papers, map and fine book paper. Similarly uniformity of color has slight influence on the quality of small lots of paper, especially if these are to be used as single sheets. It does bear directly upon the quality of paper for serial publications which are to be bound together. With the invention of the Ives Colorimeter the color of

*Professional paper No. 4 of the contributions to Engineering Chemistry by the members of the staff of Arthur D. Little, Inc., Chemists and Engineers.

†Official Chemist, American Paper and Pulp Association

paper has fortunately become a property permitting of exact measurement and definition.

For almost all purposes the presence of a noticeable amount of dirt is regarded as detrimental to the quality of paper. It not only detracts from the appearance of the paper, but its presence may be indicative of the use of inferior or carelessly prepared material. In some few cases, as with wrappers made from screenings, the dirt may be so considerable in amount, and so uniformly distributed as to become an integral and characteristic part of the paper, while as with certain Japanese papers dirt in the form of unreduced bark or vegetable tissue may be so disposed within the sheet as to actually enhance its quality from the artistic standpoint. While it is not easy to give numerical expression to the amount of dirt present in a sheet, the experienced paper maker or paper buyer has no difficulty in making close comparisons in this regard. It is by no means so easy to estimate exactly the bearing of relative amounts of dirt upon quality.

Fortunately, what may be called the fundamental properties of paper permit in nearly every case of exact measurement and numerical statement. These fundamental properties are thickness, weight per unit area, resistance to bursting strain, tensile strength in different directions, stretch, ability to resist wear, tendency to absorb ink or water, opacity. Furthermore, these properties are largely determined by the composition of the paper, and the care and skill with which its material has been prepared and manipulated during manufacture. Fortunately, again the composition can be accurately determined by chemical and microscopical examination while at the same time many direct and important inferences may be drawn from these examinations as to the course and nature of the processes of manufacture.

Of all the fundamental properties, thickness is perhaps most easily determined. Differences of a few ten-thousandths of an inch are instantly detected by trained fingers, and the exact measurement of thickness to ten-thousandths of an inch is easily made by means of several forms of micrometer employed in paper testing. Thickness bears directly upon quality in case of most papers, but the way in which it bears is determined always by the purpose for which the paper is intended. Bible papers, onion skin, kraft, pergamyn, condenser paper, tissues generally, and many other special sorts gain in quality, as represented by money value, with decrease of thickness, partly because of increased difficulty and cost of manufacture, but principally because they better meet the necessities of the customer. Other things being equal, a manifold paper which permits the making of ten copies is obviously a better paper for its purpose than one which cannot be used for more than five. The quality and value of the paper used for building up electrical condensers is enormously influenced by the relative thinness of the paper, since the efficiency of the condenser depends largely upon the closeness with which

the tin foil plates are brought together; but while thinness is thus important, quality in this case is finally determined by absence of pin holes, the presence of which entirely destroys the value of the paper for its purpose.

Weight per unit area is a quality factor of varying importance, although since paper is sold by the pound the lighter of two otherwise equally satisfactory papers is commonly to be preferred. This does not always hold, however, as in case of paper for conversion into celluloid. With wrapping papers weight per unit area becomes of the first importance, and when low and combined with strength commands the market, as evidenced by the rapidly extending popularity of kraft paper. With cover papers a small increase in weight may double the cost of mailing pamphlets, while obviously at equal prices per pound the cost of paper for printing a given number of pages is proportional to the weight per unit area.

Weight for bulk influences quality for most purposes, and is especially important in connection with book papers, blottings, matrix papers and box boards. With book papers, indeed, this factor is often a good exponent of general quality, since low weight for bulk implies a minimum of filler, the use of high-grade stock and skillful treatment in the beater. English book papers are notable for their bulking property and moderate weight. Where an edition involving a number of volumes of varying amount of text is to be made, the bulking property of the paper for the different volumes should be in inverse proportion to the number of pages in the volumes in order that the volumes themselves shall run uniform in size. Laid papers bulk more than wove papers of the same composition, and the bulking quality of different fibers varies over a wide range, esparto making an especially bulky paper. The use of mineral fillers, prolonged beating, hard calendering, coating and especially the addition of barytes to the coating mixture, all tend to make the paper thin for weight. They are justified only to the extent to which they may compensate by affecting other quality factors favorably.

Strength, whether measured as resistance to bursting strain, resistance to direct pull, or determined roughly by mere tearing, is commonly regarded as one of the most direct evidences of quality. There is a large measure of justification for this conclusion since high strength generally implies good stock, carefully prepared and skillfully manipulated. A certain minimum of strength, which, however, varies greatly with the class of paper, is an essential prerequisite of quality; while with special sorts of paper, as bag paper, kraft, paper for rags, twine, cartridge cases, etc., strength almost determines quality provided it is not associated with brittleness which makes the paper weak to sudden strain. With envelope papers, strength after folding is far more important than the strength of the flat paper. Curiously, rosin sizing reduces strength, and "wild" papers are commonly stronger than similar pa-

pers showing good formation. The "strength ratio" affords the most convenient way of reporting strength of papers. It is the quotient obtained by dividing the strength in pounds as determined by the Mullen tester, by the thickness in ten-thousandths of an inch. A paper 35-10,000 thick which tested 35 lbs. would, therefore, have a strength ratio of 1.0. The strongest commercial papers seldom or never show a strength ratio higher than 2.0, although we have recently prepared samples by special methods with the extraordinarily high ratio of 3.8.

The stretch of paper commonly ranges from 1 to 4 per cent. The amount of stretch shown by the paper affects quality favorably or otherwise according to the requirements of use. A large proportionate stretch improves the quality of bag papers, cartridge paper, drawing, matrix and paper for twine, artificial leather and embossing. It is very objectionable in lithograph, coated and many printing papers requiring close register, as also in paper for music rolls for mechanical organ and piano players.

In this country, up to the present time little or no attention has been paid to what in case of most papers is one of the most reliable and significant indications of general quality, namely the ability of the paper to withstand wear. Paper for money or for permanent records, loose leaf books, maps, school books, works of reference, blue prints, children's books, wrapping, card stock for catalog files, folders, and many other uses, should exhibit this property in the highest degree consistent with other requirements of the specification. Although in the past reliable methods and apparatus have not been available, in the United States at least, for determining wearing power, there is no longer any excuse for ignoring this essential element of quality, the numerical value of which may now be determined with the utmost accuracy with standard apparatus.

The capillary power of a paper, or its tendency to absorb ink or water, may be given sufficiently accurate expression by simple methods of testing. The bearing of the results upon quality depends entirely upon the class of paper tested. With due regard to other factors, the quality of writing papers rises as the capillary power falls, the quality of blotting paper is almost wholly determined by its capillary power; matrix papers and papers for celluloid manufacture or for conversion into parchment paper, vulcanized fiber or roofing must have high capillary power, that of printing papers should generally be only moderate if the best results in rapid printing are to be secured.

The determination of opacity is more difficult, although this property is often one upon which the quality of paper in large part depends as in case of bible paper, envelope paper, and book papers generally. A considerable degree of translucency upon the other hand is a mark of quality in bond papers, while for other writings opacity commonly suggests quality. In parchment paper, again, quality pre-supposes trans-

lucency, while pergamin papers of good quality must be almost transparent.

These physical properties of paper which, taken together, and interpreted with reference to the use for which the paper is intended, are almost conclusive as defining quality, are nevertheless themselves largely functions of composition of the sheet. For this reason any final estimate of quality must include consideration of the sorts of fiber present, their condition and proportion, the kind and amount of filler, the proportion and sorts of size, and the presence or absence of acid, bleach residues, and other chemicals likely to cause deterioration or otherwise impair quality.

Although paper of some sort may be made of almost any vegetable fiber, the number of fibrous raw materials economically available for paper making is, in view of the almost infinite variety of vegetation, surprisingly limited. Cotton, flax, jute, straw, esparto, hemp and a few woods almost exhaust the list. Nevertheless, not only does each of these fibers impart definite and often distinctive characteristics to the sheet into the composition of which it enters, but the range of variation is further enormously increased by admixture of several fibers, and the chemical and mechanical treatment which they have received by way of preparation. The best stock for any paper is easily spoiled by careless or improper methods. The stock that under skillful treatment normally yields a strong, tough, permanent sheet, may by improper beating or careless bleaching give only a weak, brittle and short-lived paper. The degree to which the fiber substance is hydrated during beating has a profound effect upon the properties of the paper, as also the manner in which the fibres themselves have been subdivided. All this, of course, to the papermaker himself is the mere A B C of his art, but it is commonly unknown to the paper consumer or ignored by him in his estimates of quality. Quality cannot be predicated upon stock alone. It is, nevertheless, true that for many papers, especially those intended for writing and the higher grades of printing, the properties representative of the highest quality are not obtainable except through the use of a large proportion of rag stock. For insulating papers such as cable paper, however, manila stock is distinctly better than rag, and, curiously enough, so is even a lime cooked straw pulp. Where great toughness is desirable, as in tag and certain wrapping papers, the highest quality is obtainable only through use of manila and similar bast fibers.

The relation of the wood fibers to quality has been a matter of controversy since the time of their introduction. As to ground wood, there is probably now no question of its unsuitability for practically every use involving permanency, since exposure for even a few hours to direct sunlight renders ground-wood papers weak and brittle. It is particularly undesirable in so-called manila papers intended for making envelopes, which, if containing much ground wood, quickly lose all strength on the fold under the influence of sunlight.

For wall papers, rather strangely, ground wood does not seem open to much objection, partly because no great strength is required of such papers, but principally because the surface coating with size and color protects the paper beneath. For newsprint paper, ground wood remains the only possible material in spite of some minor disadvantages for such use.

Sulphite and soda fiber, if thoroughly well cooked and carefully bleached, are probably nearly, and perhaps quite as permanent as rag stock. In fact, the examination of over 400 samples of paper from the library of the University of Berlin showed that papers composed wholly or in large part of chemical wood fiber gave slightly less evidence of deterioration than all rag papers. Since the better grades of rag stock are considerably more expensive than chemical wood fiber, endow the paper with more wearing power and better feel and texture, the presence of wood fibers is, in most cases, properly regarded as tending to lower quality. They are highly objectionable in papers for making parchment and vulcanized fiber, which should consist wholly of cotton rag stock.

The number of cases in which mineral filler improves the quality of paper is comparatively limited, although it is well recognized that for certain uses a moderate amount of filler reacts favorably upon quality, by increasing opacity, and improving feel and finish. Filler invariably, however, lowers strength and resistance to wear, diminishes bulk for weight and tends toward brittleness. In many special papers it is wholly inadmissible.

The number of factors which are concerned with the quality of paper in its multitudinous applications to special uses is so great as to prevent consideration or even enumeration of them all. A paper for wrapping hardware or a card for mounting silver jewelry may seem to possess every desirable property, and yet be worse than useless because of a trace of sulphur. A printing paper may develop "whiskers" or clog the type by mineral filler, a coated paper may pick or develop odor, a cigarette paper may burn badly, a writing paper may allow the ink to spread because the size has been converted into peptones by over-heating, a filter paper may fail to hold a fine precipitate or unduly retard the passage of liquid, and so on. Enough has been said to suggest to consumers of paper the complexity of the problems involved in the determination of quality, the importance of paper testing, and the advantages to both maker and consumer of carefully considered and intelligently drawn specifications defining quality as a function of intended use.

Japanese cotton-mill interests are embarking upon the enterprise of raising cotton in Siam. Considerable Siamese cotton has been shipped to Japan at times in recent years and it is said in a general way to equal Indian cotton. Considerable areas in Siam are said to be suitable to cotton growing.

EXCERPTS FROM ADDRESSES ON CERTAIN PROBLEMS CONNECTED WITH THE PRESENT-DAY RELATION BETWEEN CHEMISTRY AND MANUFACTURE IN AMERICA.*

ROBERT KENNEDY DUNCAN.

The problems to be discussed are in large measure the result of a lack of mutuality between the factory organizations on the one hand and the universities and technical institutions on the other. This lack of mutuality depends primarily upon a lack of understanding on the part of factory organizations of the advantages that are to be derived from the employment of sensible chemical research, and of a corresponding lack of understanding on the part of institutions of learning of the proper training and type of men that are requisite for the successful practice of chemical industrial research in these factories. A mutual understanding, in America at least, might speedily lead to co-operation, and co-operation to a sensibly functioning co-ordination. The American factories, taking them by and large, and making such distinguished exceptions as could be numbered on the fingers of both hands, certainly do not have co-ordinating relations with the universities, such, for example, as we find developed in Germany. Furthermore, in so far as industrial research is concerned, they do not have co-ordinating relations with one another. There exists no common consciousness among the corporation officials of this country of the proper methods of conducting factory research. In order to show how true this is, permit me to place before you the results of a preliminary inquiry which was initiated last spring as to the conduct of factory bureaus of research:

There was eventually provided a list of some seventy-five corporations possessing either bureaus of research or individual research chemists. From these corporations there were received some forty-five frank replies, given on the understanding that each reply was to be confidential and was to be used only in a summation of results. The results of this preliminary inquiry are illuminating of the chaotic conditions that pervade factory research.

It may be interesting to know that in very few instances are contracts made with chemists. In general the chemist is "hired" by the week or by the month.

Next, as to the salaries, or, as a term which is more fitting on the whole, "wages."

In most instances wages are paid by the week: translated into terms of monthly payment, the young graduate receives sums ranging from \$45 to \$100. This latter statement exemplifies the extraordinary

* From the Journal of Industrial and Engineering Chemistry.

diversity of conduct and the chaotic conditions that everywhere prevail in factory research. So far as one may strike an average among such qualitatively different factors, \$60 would perhaps be representative.

You will probably be surprised to know that there is practically no mechanism of promotion for chemists such as often obtains in factories for officials. One corporation does state that "each man is considered every six months, and if previous work warrants the action, he is recommended to the manager for a raise of salary." Another corporation "is governed only by results;" still another says that "promotion is entirely for merit; seniority counts for little with us." One corporation adds one dollar per week every six months up to a maximum of \$40 per week.

An interesting result of this inquiry appears in the reply to the question: "Do the members of the bureau sign a contract resigning all rights to any discoveries made while in your employ?" The prevailing answer is "Yes." It is a matter of surprise that contracts embodying resignation to rights of discovery were so prevalent. In reply to the question as to whether any additional remuneration was granted to successful researchers, the prevailing answer was "no," with an occasional "increased pay." Two corporations speak of "opportunity to acquire stock" or "a cash present," but there is no instance of royalties or a share in the profits.

Personal observation for several years leads me to state that presumably 95 per cent of so-called factory research is worse than loss, *worse than loss* because the failure of the individual instance places a *finale* on the possibility of that particular factory to understand the advantage of applied science. The normal failure that attends factory research is due to ignorance of the canons of judgment in choosing chemists, inexperience in dealing with them, and a generous lack of knowledge of the facilities with which it is necessary to furnish them—laboratory, library, and living facilities.

An inquiry recently instituted among the universities enables me to state that the young men pursuing the study of chemical engineering are few in number when compared with those in mining, electrical, civil and even sanitary engineering. The reason for this lies in large measure at the door of the manufacturer, for as stated above he does not understand the conditions he must meet to insure the chemist he employs successful and happy functioning. In some measure, too, it may lie at the door of the university, for it sometimes happens that the director of the courses in chemical engineering is an engineer, not a chemist, and in cases where its responsibility is under the chemical department the director is generally "academic" in quality; in either instance the course may lack the training and inspiration necessary to send good men into industrial chemistry.

The canons of judgment in the choice of an industrial chemist appear to be:

(1) Training (Scholarship).

(2) Creative Power.

(3) Masculine Qualities, which enable the chemist in the factory to deal with workmen and govern foremen.

(4) Personal Qualities, as they affect his relations with colleagues and officials.

(5) Personal Integrity.

(6) Practicality.

(7) Health.

In proceeding to nominate a candidate for an industrial position, it has too often been obvious to me that the professor of chemistry has nominated his man on the basis of scholarship alone.

A problem of extreme importance, and one to which I have never seen attention drawn, is the extent to which outside industrial or commercial work is permissible to the instructor in the chemical department of a university. Most professors or assistant professors of chemistry have from time to time offers of routine or research problems, generally attended with the offer of a microscopic remuneration, for industrialists are prone to believe that professors "love" to do such things. Now, we all know that the more the professor becomes involved in money-making outside the less value he is to his students; we know, too, that unless he is engaged in research of some kind, whether "pure" or "applied," he is possibly of still less value to his students, and, finally, we know that his university salary, whatever it may be, is inadequate to his needs. An extrinsic, but important consideration is the fact that carrying on outside work at the expense of the university plant and at a low remuneration, because of his salary, he is taking the bread out of the mouths of his professional brethren, who have to pay for *their* laboratories, apparatus, reagents, and books. I ask again, to what extent may the professional chemist do commercial work and what should be the scale of his remuneration?

Another matter of serious moment, particularly as it refers to commercial routine laboratories, has to do with academic instruction in chemical analysis. I may say advisedly, on the basis of numerous communications, that there is considerable dissatisfaction over the type of instruction that in many instances is offered young men who after graduation enter commercial laboratories in positional capacities—dissatisfaction with antiquated and cumbersome methods that are taught and with the neglect of the time element in making analyses—dissatisfaction, too, over the tardiness with which university laboratories adopt new valid methods and apparatus, and over their neglect to teach their students some sense of proportion in distinguishing between accuracy and *practical* accuracy. Now, it is within the knowledge of us all that young men can never become chemists until they are

first trained and disciplined into the exercise of the last fraction of accuracy that is in them. On the other hand, the university can no longer consider itself as apart from life, or as other than, in part, an institution for fitting young men with opportunities for life.

Finally, in this connection, and as a matter of personal opinion, it seems regrettable that chemical instruction should be so largely analytic in character; I mean by this that there seems to be not so much a disproportionate amount of chemical analysis taught in the academic courses, though certainly there is a great deal of it, but that the other courses partake to such an extent of the analytic character in their methods. One wanders through course after course—not formally analytical—only to find that they are taught analytically by analytic-minded men.

It is for this reason that I attach so much importance to the discipline and methods of organic chemistry, for organic chemistry is almost the sole subject in the chemical curriculum in which a student gains educational training in synthetic working and synthetic thinking. Now, in industrial chemistry the fields that most need research, and that in consequence offer the best opportunities, are in large measure organic—tanning, starch, glue, soap, essential oils, petroleum, and so on almost indefinitely. Furthermore, both in the organic and inorganic fields of industrial chemistry, and outside of the routine testing-laboratories, the problems that are perhaps most important to industrialists are *synthetic* problems. This is naturally the case, for the art of manufacture is fundamentally "making things." Yet the men whom the schools furnish to solve these problems are for the most part men so drilled in analytic habits of thought and work that they cannot naturally and efficiently act or think synthetically. Glass men want new glasses, the metallurgical people new alloys, the enamel workers new enamels, and so on; and yet too often the men appointed to the task are beaten before they begin by their very habits of thought.

At the conclusion of these remarks on the relation of chemistry to industry in America, some reference ought perhaps to be made to the progress and development of the scheme of industrial fellowships which I initiated at the University of Kansas.

Fellowship No. 1. On the Chemistry of Laundering.—Mr. F. W. Faragher's fellowship was extended for six months beyond its termination, at double the value, at the end of which time he entered the factories of his donating corporation, where he is doing capable work. On the basis of a part of his researches, the University of Kansas conferred upon him the degree of Ph. D.

No. 2. On the Study of Diastase.—Mr. R. C. Shuey had his fellowship extended by the donating corporation for one year beyond its termination. At the end of this time he had perfected a method for drying alfalfa, in such a way as to conserve its diastatic content, as well as its taste, odor, color, etc. If

he succeeds in proving the practicability of his process from the standpoint of cost, his work will amply satisfy his donors.

No. 3. On the Chemistry of Bread.—The donors of Mr. H. A. Kohman's fellowship, the National Association of Master Bakers, in recognition of the value of his other work in behalf of the association, at the termination of his fellowship conferred upon him all proprietary rights in his process of standardizing the large-scale manufacture of salt-rising bread. Mr. Kohman discovered the efficient bacillus for its manufacture, isolated it, grew it in large quantities, and through its use has been able to turn out salt-rising bread, of beautifully uniform quality, at the rate of a thousand loaves a day for over a week. He has been offered large considerations for the rights of this process, and on the basis of his general work and at the request of a certain corporation, he has been offered a new fellowship, yielding \$2,500 a year, to continue his researches. His original fellowship was \$500 a year.

No. 4. On the Utilization of the Constituents of Buttermilk.—Mr. E. L. Tague, on the termination of his fellowship, has succeeded in extracting from buttermilk veritable casein. This casein, in quantities of over a hundred pounds at a time, was forwarded to the paper-makers for actual commercial practice. Their reports afforded a complete vindication of his methods.

No. 5. On the Extraction of the Utilizable Constituents from Crude Petroleum.—Dr. F. W. Bushong, through two years' labor upon petroleum, has placed the matter of this fellowship upon such a footing that I have recently been offered a multiple fellowship, providing for the employment of at least six men, to take full advantage of his work. This fellowship I shall accept so soon as rooms are provided for its efficient working.

No. 6. On Enamel-lined Steel Tanks.—On the basis of the work accomplished by Mr. A. J. Weith and Mr. F. P. Brock, the donors of this fellowship insisted upon taking them over into the corporation six months before its expiration. They have succeeded in constructing an enamel which lies beautifully on steel, which is of extraordinary resistivity and which affords good promise of being adapted to large-scale application. In addition, they have developed a substance which is not glass or enamel at all and which also lies beautifully on steel and seems remarkably well adapted in its resistivity, etc., for use as a coating for steel tanks.

This completes the list of fellowships which have expired.

Of the fellowships which are now in operation:

No. 7. On the Relation between the Physical Properties of Glass and Its Chemical Constituents.—Dr. E. Ward Tillotson is making splendid progress in this fellowship, both from the standpoint of practical results and, as well, along lines of great academic interest. He has won the full regard and confidence of his

donors and it is expected that it will soon be possible to publish some of his more important results that do not have a direct practical application.

No. 8. On the Chemistry of Cement and Lime.—Dr. J. F. Mackey is carrying out through this research a comprehensive investigation into a certain phase of cement manufacture after a fashion such as has never before been attempted, and, in addition, seems well on the way to the solution of the large-scale manufacture of slaked lime—a problem which has been of serious annoyance to lime manufacturers.

No. 9. On the Extractive Principles from the Ductless Glands of Whales.—Mr. E. R. Weidlein was sent up to Newfoundland and Labrador early in June, for the purpose of collecting material. He returned in September, completely successful in his quest. He brought back with him over a hundred pounds of suprarenal glands, over a hundred pounds of thyroids, some thymus glands and a large quantity of other very interesting material. He has already discovered that the extractive principles from the suprarenal glands of whales differ in an interesting fashion from the extractive principles obtained from land animals.

No. 10. On the Chemical Treatment of Wood.—This fellowship began considerably less than a year ago, but Dr. L. V. Redman has already succeeded in devising and developing a remarkable and promising process for the improvement of the finish on wood. The finish he obtains, while brilliant and entirely resistive to all ordinary chemical reagents, is forty-three times harder than varnish. He already has patents pending.

No. 11. On Utilities for Borax.—This fellowship, unlike the others, was but of one year's duration. On the basis, however, of what Mr. B. C. Frichot has accomplished, I am just in receipt of a letter from his donors, expressing their cordial appreciation and eagerly requesting a continuance of the fellowship.

No. 12. On the Chemistry of Vegetable Ivory.—Mr. J. P. Trickey is taking full advantage of the opportunities for investigation which lie in this subject and, on the basis of results already achieved, has won from his donors their entire confidence and appreciation.

In the University of Pittsburgh, in which this work is to be carried on as well, I have already received several important fellowships:

No. 1. On the Chemistry of Baking, yielding \$750 a year for two years, with an additional cash bonus of \$2,000.

Nos. 2, 3 and 4. On the Abatement of the Smoke Nuisance, yielding \$2,000, \$1,500 and \$750 for three men per year respectively, for 2 years, with an additional consideration of not more than 10 per cent of the net profits.

No. 5. On the Relation of the Pots to the Glass-making and the Elimination of "Strea," yielding \$1,500 a year for 2 years, with an additional cash bonus of \$2,000.

Nos. 6, 7 and 8. On the Chemistry of Baking, wholly independent of, but with the acquiescence of, No. 1—for three men, yielding respectively \$2,500, \$1,500 and \$750 a year for two years, together with an additional consideration of not more than \$10,000 cash.

In addition to these fellowship, I have pending, as stated above, one providing for 6 men, in petroleum research and another multiple fellowship of \$10,000 a year for 2 years, for 3 men, yielding \$5,000, \$3,000 and \$2,000 per year respectively, together with a large additional consideration.

The scheme itself, as expressed in the agreement between the various corporations and the universities concerned, has undergone a remarkable development. These fellowships, beginning with \$500 a year, have gone progressively up through \$650, \$750, \$1,000, \$1,500, \$2,000, \$2,500, and now I am threatened with fellowships yielding \$5,000 a year and \$3,000 a year to the fellow appointed.

One remarkable phase of its development lies in the creation of multiple fellowships, designed for the intensive work of several men upon one problem.

M. C. WHITAKER.

Prof. Duncan's plans for the encouragement and development of industrial research opens up an enormous field, and one which will be of immense benefit to the manufacturers. The subjects undertaken should be selected with the greatest care, and should be limited to those branches which can be handled in properly equipped laboratories. Business men may be interested in supporting an industrial research when it can be shown that the laboratory equipment is of a character to offer them superior facilities to those found in their own works laboratories. But it would be a difficult matter to interest a keen business man in an industrial research where the laboratory equipment is no better, or, as in the case of many institutions, even less than may be found in his own works. Plenty of industrial problems may be found, and generous support will be given by industries as fast as it can be demonstrated that qualified men and satisfactory equipment may be enlisted for their solution.

One of the most important things to be considered in the education of the chemists for manufacturers is the fundamental education.

Equally important and essential to the young chemist in making himself indispensable to the manager is a knowledge of certain other subjects, such as works equipment, works organization, works labor, and works economics.

The margin between a profit and a loss in a chemical manufacture is often very small and no matter how perfectly the chemical reaction may be developed, a poorly equipped works may convert it into a losing venture. Again, a losing chemical process may be, and often is, converted into a dividend-payer by improving and developing the organization for its execution.

Then comes in the question of labor. Of what con-

sequence is the chemical reaction which depends upon labor to make it work, if the chemist does not know how to make a "Dago" work? Are the men ever taught the *applications* of these sciences to manufacturing problems with real live factory appliances? Do the students ever realize that the organization factor, the labor factor, the general process economics are all equal in importance to their chemical knowledge when it comes to producing the results for which the manager is held responsible and for which he employs the chemist to assist him?

I think that the factory managers are not altogether to blame for the state of affairs Dr. Duncan has described. The chemists are not altogether to blame either. The education of the men to solve the problems of chemical manufacture is a great undertaking, and one which is not to be considered on the same basis as in other engineering fields.

DR. L. H. BAEKELAND.

One of the main reasons why the work of the chemist is not properly understood nor properly appreciated is that the average public, even including educated men who are not chemists themselves, is not in a position to grasp the scope and the purposes of chemical investigation. The man of average intelligence can easily understand a mechanical proposition involving the work of the engineer, but it requires chemical knowledge, besides intelligence, to have some understanding of chemical problems and their difficulties. The average man has only the haziest conceptions of this subject.

Chemical research work, at its very best, is slow in practical results, and what is still worse, the real visible difficulties begin only after the initial research work is over, and the subject is to be reduced to practice. Therefore, to an uninformed person, the value of all the preliminary painstaking research work is not very apparent.

Research in industrial chemistry involves work of no less high order and no less difficulties than that required for purely scientific investigation. In fact, the man engaged in purely theoretical work has a great advantage over the industrial chemist because he has to take into consideration the cost of production, the market conditions, and the factor of superiority of quality. If the theoretical investigation of the subject becomes too difficult, or does not bring in results, or does not please him, he does not have to pass sleepless nights over it, but he can simply drop it then and there, and quietly select some other line of work more to his liking, and carry it out according to his own fancies and without submitting himself to the drastic criticisms of the consumer or the market at large.

Too few chemical inventors realize the enormous distance which lies between a laboratory conception and its successful application to practice. Many problems which are solved in the laboratory after great effort often require years of painstaking work, not to

speak of immense financial sacrifices, before they are acceptable in current practice and become a basis for a paying commercial operation. By the latter I do not mean, of course, the so-called "rake-off" of promoters, who frequently begin by smothering a good thing under too heavy charges.

The production of garnet for abrasive purposes in 1909 in the United States amounted to 2,972 short tons, valued at \$102,315, as compared with 1,996 short tons, valued at \$64,620, in 1908. This was a fair increase over the production of 1908, but is nevertheless somewhat disappointing. The industry does not yet seem to have recovered from the setback it received during 1908 from overproduction, a declining market, and the invasion of the abrasive industry by artificial abrasives. The average price of garnet per ton in 1909 was \$34.43, an increase of \$2.06 per ton as compared with \$32.37 in 1908. The garnet industry is practically confined to New York, but a small production was reported from North Carolina in 1909.

According to the annual report of A. Hirsch & Sons, well-known authorities, the consumption of metals in Germany showed a considerable increase in 1910 as compared with 1909. The fact that exports of copper manufactures amounted to 88,195 tons, 12 per cent more than in 1909, is illustrative of the general prosperous condition of the metallurgical and allied industries during the past year. The figures given in the following review of the situation are based on estimates of A. Hirsch & Sons. Germany purchased 207,419 tons of foreign copper in 1910, against 187,186 tons in 1909. About three-fourths of the imports come from the United States. In addition, German mines yield some 31,000 tons annually. The production of the United States, Canada, and Mexico during 1910 is estimated at 646,504 tons. Shipments of American copper to Europe amounted to 314,681 tons. Supplies at the ports of Hamburg and Rotterdam at the end of 1910 are placed at 16,300 tons, against 2,200 tons the preceding year. The consumption of lead in Germany was approximately 215,000 tons for 1910. Imports for the first 11 months of that year amounted to 71,625 tons, in addition to about 100,000 tons of lead-bearing ores. The prevailing opinion is that the price of this metal will advance during 1911. The market for zinc was very active during 1910, so that the stock on hand in zinc-producing works is very small. The German output in 1910 was 206,000 tons. Furthermore, from January 1 to November 30 of that year 35,197 tons of zinc and 218,867 tons of zinc-bearing ore were imported, chiefly from Australia. The price of tin showed considerable advance under the influence of a London combine which controls a large part of the production. During the first 11 months of 1910 Germany imported 13,039 tons of tin. The annual output of this country is estimated at 11,000 tons.

HISTORY OF ALUMINUM MANUFACTURE.*

It has been suggested to me to give an account of some of the more personal and unpublished facts in connection with my invention of the aluminus process, and of the work of putting it on a commercial basis.

My first knowledge of chemistry was gained as a schoolboy at Oberlin, Ohio, from reading a book on chemistry which my father studied in college in the forties. I still have the book, published in 1841. It is minus the cover and the title page, so I do not know the author. It may be interesting now to see what this book, published seventy years ago, says about aluminum: "The metal may be obtained by heating chloride of aluminum with potassium in a covered platinum or porcelain crucible and dissolving out the salt with water. As thus prepared it is a gray powder similar to platinum, but when rubbed in a mortar exhibits distinctly metallic lustre. It fuses at a higher temperature than cast iron and in this state is a conductor of electricity but a non-conductor when cold."

Later I read about Deville's work in France, and found the statement that every clay bank was a mine of aluminum, and that the metal was as costly as silver. I soon after began to think of processes for making aluminum cheaply. I remember my first experiment was to try to reduce aluminum from clay by means of carbon at a high temperature. I made a mixture of clay with carbon and ignited it in a mixture of charcoal with chlorate of potassium. It is needless to say that no aluminum was produced. I thought of cheapening the chloride of aluminum then used as the basis for aluminum manufacture, and tried to make it by heating chloride of calcium and chloride of magnesium with clay, following the analogy by which iron chloride is produced when common salt is thrown into a porcelain kiln. A little later I worked with pure alumina and tried to find some catalytic agent which would make it possible to reduce alumina with carbon at a high temperature. I tried mixtures of alumina and carbon with barium salts, with cryolite, and with carbonate of soda, hoping to get a double reaction by which the final result would be aluminum. I remember buying some metallic sodium and trying to reduce cryolite, but obtained very poor results. I made some aluminum sulphide but found it very unpromising as a source of aluminum then, as it has been ever since.

On a later occasion I tried to electrolyze a solution of aluminum salt in water, but found nothing but a deposit of hydroxide on the negative electrode. I did not give a great deal of time to these experiments, as I was then a student in college and was working on three or four other attempted inventions.

I had studied something of thermochemistry, and gradually the idea formed itself in my mind that if I could get a solution of alumina in something which

contained no water, and in a solvent which was chemically more stable than the alumina, this would probably give a bath from which aluminum could be obtained by electrolysis.

In February, 1886, I began to experiment on this plan. The first thing in which I tried to dissolve alumina for electrolysis was fluorspar, but I found that its fusing point was too high. I next made some magnesium fluoride, but found this also to have a rather high fusing point. I then took some cryolite, and found that it melted easily and in the molten condition dissolved alumina in large proportions. I rigged up a little electric battery—mostly borrowed from my professor of chemistry, Professor Jewett, of Oberlin College, where I had graduated the previous summer. I melted some cryolite in a clay crucible and dissolved alumina in it and passed an electric current through the molten mass for about two hours. When I poured out the melted mass I found no aluminum. It then occurred to me that the operation might be interfered with by impurities, principally silica, dissolved from the clay crucible. I next made a carbon crucible, enclosed it in a clay crucible, and repeated the experiment with better success. After passing the current for about two hours I poured out the material and found a number of small globules of aluminum. I was then quite sure that I had discovered the process that I was after.

I undertook to broaden and improve the method, and found that I could use, instead of cryolite, other double fluorides, particularly a double fluoride of potassium and aluminum. The most important change, however, which I made at this time, was in the material used as an anode. I wanted to get rid of the burning up of the carbon anodes. I tried a platinum anode and found that it seemed to work all right, but it was too expensive. I discovered that if I used a fusible bath of a potassium double fluoride with a sodium double fluoride, I could use a copper anode, which immediately became coated with a thin film of copper oxide and acted like a permanent platinum anode. This was not a step in advance as I had hoped, because more or less copper got into the reduced aluminum, and the use of a copper anode led me to use very fusible baths, which on the whole did not work as well as the less fusible baths. It is probable that this change delayed a successful result for a year or two.

When worked on a small scale, this process with any of the baths I have mentioned, and with either copper or carbon anodes, is not apparently promising. The ampere efficiency is low, sometimes zero, and the bath, whether composed of sodium or potassium salt, becomes filled with a black substance which accumulates and renders the process very difficult. I presume that my friend, Dr. Héroult, whom I have the pleasure of seeing here to-night, who invented the process independently in France about the same time, encountered the same difficulties. In spite of the difficulties mentioned, however, I had great faith in the theoretical

*Abstracted from remarks of Dr. Chas. M. Hall on the receipt of the Pekin Medal, awarded him for his inventions and discoveries in connection with the manufacture of aluminum.

possibilities of the process, and believed that the practical obstacles could be overcome, so I stuck to it from the start.

On the financial side I had, in the course of three years, three different sets of partners or backers, of whom two sets became discouraged and gave up. In the summer and fall of 1886, I worked with some people in Boston with whom my brother had made some financial arrangement. The results there obtained were not satisfactory to them and in October, 1886, my Boston friends declined to go further.

In December, 1886, I returned to my home in Oberlin, continued my study, and found that a bath composed of a very fusible double fluoride of aluminum and potassium, with copper anodes, worked much better than anything I had before tried. I have here a number of buttons of aluminum made by this method at that time. The larger one was made with current from a galvanic battery on December 7, 1886, and weighs about 8 grams.

After this I negotiated with the head of one of the large chemical manufacturing companies of the United States with headquarters at Cleveland, but we could not finally agree. I thought then, and have thought since, that this gentleman was the kind of man whom the proverbial inventor meets when he is deprived of the fruits of his labor. He was described to me as extremely careful and conservative, and was always wanting to take a few days more to think the matter over. He kept me on the string six months, and the kind of contract which he finally offered seemed to me entirely one-sided. The gentleman told a friend of mine afterwards that I was no business man (which was no doubt true), but I received what seemed to me much fairer treatment from all with whom I afterwards had business dealings, and found much more liberal associates and friends.

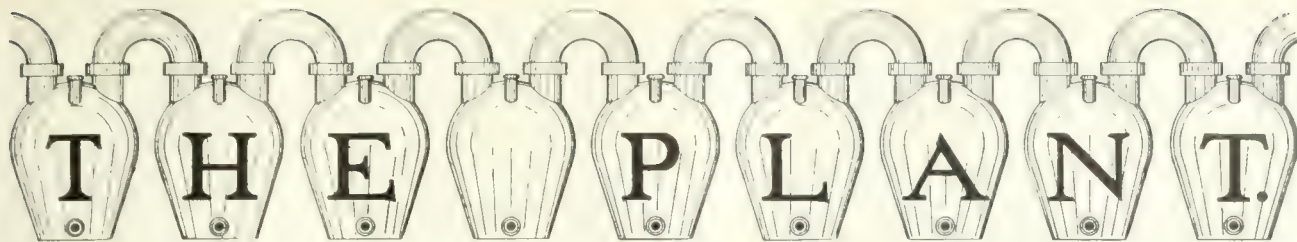
The Cowles Electric Smelting & Aluminum Co., who were then making aluminum alloys at Lockport, N. Y., were the second set of people who became interested in my invention. They took an option on it, and I spent a year with them, from July, 1887, to July, 1888. They finally gave up their option. The baths which I used at Lockport worked well for a few days, but after a time became less efficient. I finally worked out a system by which the difficulties were overcome. This was by making a bath consisting partly of calcium fluoride, or fluorspar, and adding 3 or 4 per cent of jectionable compound which spoiled the bath. After finally overcoming the difficulties which I have mentioned, I made several pounds of aluminum in small crucibles, which I showed to Mr. Alfred Cowles and gave him all the facts in relation to the same, but he was not interested.

I then sent a representative—Mr. Romaine Cole, now dead—to Pittsburg, to get together the gentlemen who formed the Pittsburg Reduction Company, now the Aluminum Company of America. We started in the summer of 1888 to build and operate a com-

mmercial plant on Smallman street in Pittsburg. We had at our disposal about fifty horse power in electrical current of 2,000 amperes. It took a few weeks after starting to get the dimensions of our baths just right, and then the difficulties which I have referred to disappeared as if by magic. The clogging and spoiling of the bath, which had caused trouble for the last three years, did not occur on a large scale. No calcium chloride was required. It seems that this is a process (unlike a good many others) which works badly on a small scale and well on a large scale. I accounted for this by the fact that on a large scale the electrodes are further separated and there is less circulation between the positive and negative electrodes, which lowers the efficiency and favors the formation of the clogging black compound. We also found, as I had predicted nearly three years before and had stated in a patent application filed two years before, that on a commercial scale no external heat was required to keep our baths in fusion. This was a great advantage, but I believe that it resulted from a law of nature and not from any invention, as we did not use any excess of current for maintaining fusion, but only the normal current and voltage for electrolytic purposes. The use of the electric current for fusing and heating in connection with electrolysis was a thing which had been disclosed and published almost a century before.

The maintaining of aluminum has now grown to a great commercial business. Many workers have contributed to it, and the credit is to be divided among many. Our financial people, particularly the gentlemen of the Mellen Bank, of Pittsburg, have given indispensable aid. Our patent attorneys and experts, our engineering and chemical staff, sales agents, superintendents, foremen and others have all done their share. In the commercial development of the business I think the greatest credit is due to our first president, Captain Alfred E. Hunt, who died in 1899, and to our present president, Mr. Arthur V. Davis, who has been identified with the business for twenty-two years, and who has been manager and general of our forces for the last eleven years.

A new isothermal valve has been invented by Alfred Baldwin, of Keighley, Yorkshire, England, and is now in operation, whereby when the temperature of a thermometer placed in the dye vessel reaches any fixed point it completes an electrical circuit which, by means of an electro magnet, automatically shuts off the steam. By this means the range of temperature can be kept at 205° to 206° F. Since the employment of steam the temperature in a dye vessel has varied from 208° to 210° on the side where the steam was admitted down to 202° on the opposite side; this difference of 6° to 8° possibly accounts for unevenness in dyeing. By this new control valve it is claimed that 95 per cent of the damage can be avoided and there would be a saving in steam of 50 per cent in open vessels to 85 per cent in closed vessels.



TRANSITE AND STEEL TOWER TOP CONSTRUCTION.

By WARREN H. MILLER.

One of the most annoying problems about a chemical plant is how to counteract the tendency of process tower top and still tower sheds to drop loose boards and trim, due to the acid eating through the nails in the minute cracks where two boards, or a board and a weather strip, are nailed together. Every violent wind storm or thunder squall is sure to bring in its train a shower of louvre shutter slats, weather strips and cornice fascia boards from the process and still tower tops, besides all sorts of light woodwork once put up with ordinary carpenter nails which are now all pretty well eaten through. It is no unusual thing for whole boards to come sailing down from the still tower tops, which are the most exposed. The resulting minor carpenter expenses, together with the generally acid-eaten condition of the boards and trim led us finally to investigate a construction that had no wood or nails in it at all. The idea was to have red "French A" or Celadon roof tile on $2\frac{1}{2}$ in. by 2 in. by $\frac{1}{4}$ in. angle iron purlins spaced $13\frac{3}{8}$ ins., the roof to be carried by light steel trusses; the walls would then be of sheets of "transite" bolted to suitable girts running around the tower tops. Transite is a non-inflammable composite board, made under hydraulic pressure of asbestos fiber and plaster bond. It has a smooth glassy face which will take paint nicely, but the back is rough so that any plate is non-reversible. It is made in thicknesses from $\frac{1}{8}$ in. up to $\frac{1}{2}$ in., and in two stock sizes, 40 ins. by 40 ins. and 42 ins. by 44 ins. The $\frac{1}{4}$ in. thickness weighs 2.3 lbs. per square foot. While it is not acid proof in the sense that it will resist the steady dripping of weak acid on it any more than red roofing tile will, yet it will outlast the best pine boards, and, as it is bolted with $\frac{1}{4}$ in. bolts, these will resist being eaten through much better than ordinary carpenter's finish nails. Any structure sheathed with it is practically fire-proof, though under great heat, when surrounded with flames, it is apt to spall apart if water is thrown upon it. It can be sawed, nailed, chiselled and drilled, though one always uses old tools for this purpose, as it is death on the fine edge of wood-working tools. No holes should be drilled nor nails driven nearer than an inch from the edge, as it is much more likely to chip out than to pass the nail if less than this is allowed.

The transite boards are put up in rows, each over-

lapping the row below, all joints meeting, so that a crack will run up a whole series of boards. These vertical cracks are then covered with 4-in. strips of transite, which also overlap the straps below, so that the sheathing is rain-proof, even against severe driving storms. As no two boards are exactly the same color, it is necessary to paint the work when done, and we found that black tar was the best and most durable paint. We first tried the graphite paint so successfully used about chemical plants on iron work, but in a few weeks the weather had washed it off, leaving a streaky, drab, and miserable appearance. But, with tar finish and red tile roof, our process and still-tower tops looked quite ornamental for a chemical plant.

The problem of the best steel frame to mount the transite board on was next given considerable study. The first scheme tried did not seem to the writer to be the best idea, but still one tower top was finished according to it. Briefly, it was this: The corner posts being 3 in. by 3 in. angles, the girts were selected of Z-bars (Carnegie Z-19), and each one of these Z bars was drilled with $\frac{3}{8}$ in. holes spaced 8 ins. apart in the upstanding leg of the Z. All the transite boards were to be piled up in sets of ten and drilled with a compressed air drill through a metal template. The holes were to be $\frac{1}{4}$ in. plus a fraction, so as to easily pass a $\frac{1}{4}$ -in. bolt.

The writer shook his head over this scheme. Suppose the drill was not held exactly plumb—how about the holes in the last few boards far below the template? Even if you got them square, would all the Z-bars be exactly straight, exactly plumb, all the holes in a row one below the other? Not as structural steel is usually made and put together for ordinary skeleton steel construction. As a matter of fact few or none of the Z-bars were exactly straight, nor were any of the measurements between holes on different girts the same, nor anywhere near enough for transite. In ordinary steel construction these matters make little difference, as you can pull almost any job of steel work into place with drift pins—but not with transite. The least attempt to spring a buckled or sagging Z-bar into place would end in the hole pulling out of the transite board. However, we tried out the scheme on one tower top, which proved to be a world-beater in point of erection cost. The labor on it alone went over 12 cts. per square foot of transite sheathing. The first ten sheets from the template were tried out one by one, but fitted—nowhere. While they might be coaxed to take the four corner holes, they usually managed to

miss all the others. We found that to make a job of it, each separate board would have to be held in its place and the holes scribed on it, drilled, and then fitted where it belonged. The tower top frame would also have to be plumbed to mechanical accuracy and held so with wooden shores until all the transite boards were on, otherwise no sets of holes would be square and the corners wouldn't match. As we had some 30,000 sq. ft. of process and still tower top steel framing to cover, this transite proposition bid fair to eat up a lot of money unless some simple scheme was devised to fasten on the boards without all this drilling and fitting. It seemed to the writer that some sort of a clip would be the best idea. Something that could be manufactured in quantities and put on the boards in the shop, so that no fitting would be necessary up on the steel work in the howling wind. As each sheet overlapped the one below it would only require a clip on the bottom, locking over the lip of the Z-bar. They were recessed 9/16 in. so that the clip would easily include both the leg of the Z-bar and the upper edge of the sheet below. Two clips were used to each sheet, being bolted on about four inches above the bottom edge of the sheet. To put them on you began with the bottom row, using small cast-iron C-clamps to hold the top edges of the sheets as you went along. As soon as the next row of sheets went on above, their overlaps held the tops of the sheets of the lowest row and released the C-clamps for duty on the top of the second row. Most of the tower sheds were four rows high. All the sheets could now be drilled with a template, as it made no difference whether there was a slight variation of 1/4 in. or so in the spacing of the Z-bars, as of course, this would be lost in the overlap. As to horizontal fit, the sheets could be slid along on their clips to take any position desired. Altogether, the work went along very smoothly and swiftly as soon as we began to use the clips and the 30,000 sq. ft. of transite board was put upon the steel tower top framing covering the Gay-Lussac, Glover and supplementary process towers in both works "A" and "B" and four still tower tops.

To make the transite clips we made a very inexpensive die which was planed out of a block of cast-iron, chilled, and the forms stamped out of pieces of 18 gauge sheet iron each with a single tap of the steam hammer. Each form made two transite clips as it simply needed cutting in half under the shears and punching for bolts to be ready for use. To put up the work you needed three sizes of 1/4-in. bolts; 3/4-in. for the clips, 1-in. for the straps and 1 1/2-in. for the lower ends of the straps where the bolt would have to pass through two straps and two sheets of transite. The straps themselves were 40-in. and 42-in. strips of transite, 4 ins. wide, and came already manufactured with the sheets. We staggered the bolt holes for them, and so got along with six bolts to the strap for each sheet. The transite clips took six and then there were six extra ones to be supplied for the top row of sheets which

had to be secured by a second pair of clips taking the top Z-bar. After it was all up, a painter in a bosun's chair tarred the whole job and it remains glossy and black to this day.

When we came to build more still houses with their towers the question of tower top covering came up again. This time we decided to eliminate the Z-bars entirely. To begin with, they were phenomenally weak in the direction they should be strongest, which was sidewise, to resist lateral wind pressure. Moreover, they were no longer rolled by the Carnegie company as a standard section, and when delivered were apt to be warped out of true, sometimes sagging as much as an inch from the true line of duty, so to speak. It seemed to the writer that the same weight of metal could be far better put into a stout angle held with the leg upstanding by reverse angle clips at the posts, and so, after making due representations through the proper channels, permission was obtained to go ahead and design something that would beat the Z-bars of which so many tons had been used. In Still Tower No. 5 the Zs were replaced by 3-in. by 2-in. by 1/4-in. angles with the 2-in. leg upstanding. It gave a far stiffer construction especially to resist lateral wind pressure, having about twice the moment of inertia of the Z-bar. The transite boards were put on precisely as before.

Throughout all this transite work we were forced to use metal corners, as it would be an impossibility to make up the corners square and true in transite or to keep them so if successful. The cheapest way out of it was to make up a lot of 22 gauge galvanized iron corners, 4 ins. on a side, and bolt them onto the end pieces of transite at each row, each corner overlapping the one below it, and passing under the transite of the sheets above it. These corners were, of course, tarred at the same time as the rest of the work.

The prospects for a reasonable window scheme for these transite tower sheds seemed bright for plenty of trouble ahead. We had 36 windows to provide for in the process tower tops and 13 in the still towers, so it would not do to cobble out a window for each particular case. A mill job would have to be designed and the 51 window frames and their sashes run out in a quantity. The job proved easier than it looked. In the first place, the distance between Z-bars was invariably within a small fraction of an inch, and all the transite boards were the same length, so that all the openings were pretty close to 37 7/8 ins. by 40 3/8 ins. A tower-top window must take the place of a single transite square, and it must also be opposite the distributors on each tower top, so that the foreman can see at a glance that all the overflows are free and all the pipes delivering to their respective holes. You all know how egg-blown weak acid is apt to clog up in some grooves and overflow in others, thus giving uneven distribution throughout the tower. In more than half the available window locations there was already an upstanding leg of an angle-iron taking the middle

of long Z-bar girt spans, such as in the 20 ft. sq. supplementary towers, so that the window sash would have to be sliding, not swinging, or else it would interfere with the angle. As the Z-bars formed the top and bottom of the window opening the sash had but one available way to slide, which was sideways. Having definitely fixed these limitations in the window design, it was then easy to get up a frame that would fit universally, whether there was an angle iron behind the window sash or not. I first made the section of the transite window shown in the illustration, Fig. 1. A yellow pine bottom sill was gotten up to fit the bottom Z-bar; next, the top sill was selected to fit under the top Z-bar. There were no side frames, as the window had to be free to slide either way, but the outside and inside stop frames were carried down on each side so as to permit the sash to slide through them. A slanting lower sill-board was then put on outside and a corresponding stool on the inner side

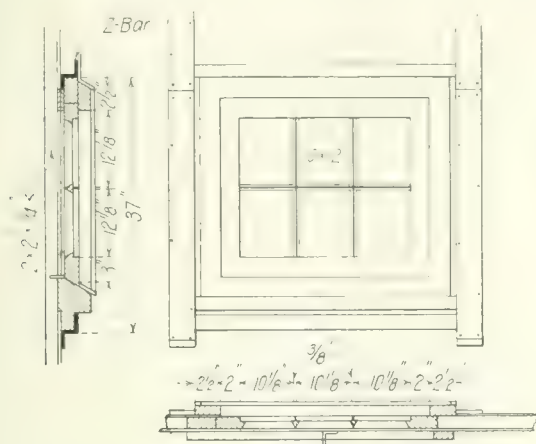


Fig. 1—Sketch of Transite Window.

of the lower sill, thus providing a substantial bottom groove for the sash. To get a water shed for the rain running down from the transite above, I took advantage of the fact that the lap of the transite would come down at least $\frac{3}{4}$ in. below the Z-bar. The upper sill would thus fit up snug under the bar behind the transite, and it sufficed to bevel the outer stop and put on a little white-pine eave to take the whole rain shed and drip it clear of the window. With these windows I was able to get in six 10-in. $\times$ 12-in. panes with $\frac{3}{8}$ -in. muntins and 3-in. $\times$ $1\frac{3}{8}$ -in. frames. The permissible thickness of sash under the Z-bar was $1\frac{3}{8}$ in. plus a trifle. I had all the sashes run out at the mill and all the lumber cut and finished to sections on the drawing so that all our carpenters had to do to put in a window was to cut off the right lengths of sill and stops and nail them up in place, notching for an angle iron and running the bottom side right or left, according to which way the sash ought to slide.

A word in conclusion as to terra cotta tile roofs. We found these quite as satisfactory on these light tower-top structures as on the heavier burner and still-house roofs. The plain glazed American designs proved more serviceable than the French, as the grace-

ful flutings of the latter made them more apt to accumulate dirt, ash and suphate. The latter were also unglazed or only partly so, so that under the steady drip of weak acid from some such matter as a leak in a still line they would crumble to a soft friable mud in a few weeks times. The American design cost 11 cts. per tile, with slightly more for gable tiles and less for half tile. To do roofing work with them you need, besides the standard tile, right and left gables, right and left half tile, ditto half gable, standard ridge tile and end ridge tile. The labor of putting them up is very little, as we found that a gang of the works Hungarians, five in number, sufficed to put on 2,000 sq. ft. a day, including wheeling the tile to the building and hoisting them up to the purlins. They lend themselves admirably to skylight work, since the glass tile, cast in exactly the same moulds as the terra cotta ones, are interchangeable with them and permit you to use one or a dozen, as the case may require, with perfect facility. Over the weak acid tanks on top of the process towers and still towers, a single glass tile in the center on each side is a very useful skylight, enabling the foreman to read scale and hydrometer measurements much later in the day than would be possible without such a skylight. These glass tile cost 50 cts. apiece in the standard mould, and about 27 cts. in half tile. The cost of these tile roofs is \$13 a square as laid. They are secured with copper wire passing around the purlins and fastening in an eyehole in the terra cotta. The steel work consists of $2\frac{1}{2}$ -in. by 2-in. by $\frac{1}{4}$ -in. angle-iron purlins, with the $2\frac{1}{2}$ -in. leg upstanding. They are spaced $13\frac{3}{8}$ ins., all except the bottom one, which is 12 ins. and has a 3-in. upstanding leg because there is no lap of any tile below it to hold the lower edge of the last row of tile in true alignment with the pitch of the roof. This is quite important, as, if neglected, the bottom row will not only sag down, but the lips of it, which go under the next row above, will lift them up out of the general plane of the roof and make an exceedingly unsightly eave of it. The first hole on each side of the peak is punched for the purlin $4\frac{1}{4}$ ins. below the peak. This will bring the top lips of the tile in correct position to receive the feet of the ridge tile. As to roof span with these purlins, we found that 12 ft. was about the limit, with 8 or 10 ft. spans giving stronger and stiffer roofs capable of carrying the heaviest snows. The minimum slope is $4\frac{3}{4}$ ins. by 12 ins., which will turn the heaviest thunder shower. The tile work very easily around points like still towers, octagonal process towers, etc. The lead flashing is burned on the tower and led out over the roof about a foot, overlapping the adjacent tile. This will deliver the water far enough out so that it will run down the grooves of the tile to the eaves, forthwith. We found that about once a year these roofs had to be re-tarred as to the steel work, and all the tile were then taken off, washed clean of accumulated sulphate and provided with new copper fastener wires.

As to the methods of holding these light tower top structures in place, the channel-iron bracket shown in the designs was amply strong for all sizes of towers, and could be secured to either wooden or steel tower posts. A set of these brackets was put around each tower, with an architect's level to get them all true, and on them went the channel-iron bottom frame of the house, bolting to the bracket by a hole in the bottom flange of the channels. On these the 3-in. by 3-in. corner posts were next erected and a few girts gotten on to hold them square. The posts were next plumbed and braced by temporary wooden shores and the rest of the girts bolted on, together with the trusses and purlins. The whole structure was next rivetted up without removing the wooden shores inside, as it is very important to keep the frame reasonably square until after rivetting. As there are no diagonals in the steel-work, this must be done with 4-in. by 4-in. hemlock shores. After the transite is on, the shores can be removed and the structure will then stay square.

The cost of the transite in $\frac{1}{4}$ -in. board, erected complete, was 15 cts. a square foot, and the steel work to hold it cost 7.2 cts. a pound erected and rivetted.

In 1909 the production of borax in the United States was 41,434 short tons, valued at \$1,534,365, an increase in quantity, as compared with 1908, of 16,434 tons, and in value of \$559,365. The quantity stated is the crude material mined and the value is fixed according to the percentage of anhydrous boric acid in the ore. All the output in 1909 was derived from two mines in Inyo and Los Angeles counties, Cal. Only colemanite or borate of lime is now being mined in California, and this varies so greatly in its content of anhydrous boric acid, not only in different mines, but in any single mine, that it is necessary to determine the percentage of this acid in the ore in order to fix the value of the output.

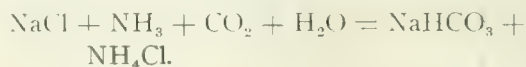
A primitive type of cotton gin is used in certain sections of China, according to a recent U. S. Consular Report. The gin consists of a corrugated iron roll about one-half inch in diameter and about 20 ins. long, set in close contact to a wooden roll about $\frac{3}{4}$ in. in diameter. The wooden roll is operated by a crank turned by hand. The iron roll is operated by a treadle wheel. The wheel is about 2 ft. in diameter with a crank pin about 4 ins. from the axle. A rod connects the crank pin with the treadle stick, the latter being a rough tree branch ending in three prongs. The operator stands with right foot on treadle stick, by means of which he turns the iron roll, and with his left hand turns the crank which revolves the wooden roll, while with his right hand he feeds the cotton pods into the rolls. The rolls pull all the fibers off the seeds and carry them through, while the seeds drop without having been carried through. The cotton seed is used for oil. Probably the entire cost of a cotton gin such as this is no more than \$5 United States currency.

THE MANUFACTURE OF AMMONIA-ALKALI IN ENGLAND.*

By WILLIAM MASON.

There is no branch of chemical industry in which the principles of engineering science have been so profitably applied as that of the manufacture of ammonia-alkali. The world's production of this article, which in 1883 was only 160,000 tons, has steadily increased until at the present time it is estimated to amount to upward of 2,000,000 tons per year, about nine-tenths of which is said to be produced by the Solvay system. The English Solvay works of Messrs. Brunner, Mond & Co., Ltd., in the Cheshire salt district, are the largest in the world and since the year 1881 have continued to show the most magnificent financial results.

It has been thought by some that the Solvay system was a secret method, but that is not so. This system works the method first made known by Schloesing and Roland in 1854, which is a modification of that practiced by earlier workers, such as Dyar and Hemming, who obtained the first patent for this process in 1838. It is represented by the following equation:



The ammonia-alkali process is so simple, so unlike the Leblanc process, that anyone with only a little experience in chemical manipulation can produce sodium carbonate by it.

Solvay's first patent in 1863 was for plant, and a few years afterward he established works at Couillet, in Belgium, making in the first year 179 tons of soda ash. He was the first to make the manufacture of ammonia-alkali a financial success, increasing the production each year, until in 1874 he manufactured upward of 4,678 tons, when it was introduced into England by the later Dr. Ludwig Mond with a success that is without a parallel in the history of chemical industry. The English Solvay works have paid an average dividend for the last 28 years of 31 per cent, the lowest in 1881 of 19 per cent, and the highest of 100 per cent in 1893.

Notwithstanding such success it can be shown that in the works of this successful company, when they were annually making a profit of nearly £200,000, in trying to improve the chemical side of this manufacture, they adopted and persistently worked a method which must have added at least 10 shillings per ton to the cost of production.

It has often been stated that in the carbonating part of this process a quantity of sodium bicarbonate remaining dissolved in the residual chloride liquors would always be lost, and to recover this bicarbonate Solvay suggests in one of his patents the placing of salt in the carbonating tower; but ordinary salt, as supplied to chemical works, contains impurities equal to

*From Metallurgical and Chemical Engineering.

0.8 per cent, which increases the percentage of insoluble matter in the finished soda ash. In 1878 Dr. Mond obtained a patent for a method for purifying the brine by adding to it a kind of water-softening mixture; by this method he obtained a very pure salt. This process was worked for some time and then discontinued, ordinary salt being afterward used.

Now there is no evidence whatever to show that any of the salt added in this way was ever decomposed, or was necessary to prevent the loss of bicarbonate in these liquors. For this purpose about 10 cwt. of salt per ton of soda ash produced was required; this salt contained 10 per cent of water, which diluted the liquors, the dilution being further increased by 50 per cent more water being required to wash out the excess of salt from the bicarbonate. This water carried away bicarbonate in solution, thus making more liquor to be passed through the ammonia still, which increased very considerably the cost of manufacture.

Below are given some analyses of the residual liquors: No. 1 from Solvay towers having had salt placed in them; No. 2, from another works which did not use salt in the carbonating apparatus, while No. 3 gives the analysis of liquors from Solvay towers after finding out that salt was not required to obtain a residual liquor from the carbonating tower, with little or no sodium bicarbonate in solution:

NO. 1. SOLVAY TOWER LIQUOR WITH SALT ADDED.

Percentage alkalinity calculated as NH_3	1.49	1.19	1.61	2.22
Ammonium chloride	16.53	17.12	16.12	11.29
Sodium chloride	13.11	12.98	12.29	14.98

NO. 2. CARBONATOR LIQUOR WITHOUT SALT.

Percentage alkalinity calculated as NH_3	1.65	1.87	1.56	1.63
Ammonium chloride	18.88	17.97	18.08	17.33
Sodium chloride	6.88	7.28	7.74	8.00

NO. 3. SOLVAY TOWER LIQUOR WITHOUT SALT.

Percentage alkalinity calculated as NH_3	2.02
Ammonium chloride	17.22
Sodium chloride	8.45

Since for every equivalent of ammonium chloride produced an equivalent of sodium carbonate is precipitated as sodium bicarbonate, analyses Nos. 2 and 3 clearly show that slightly better results were obtained than by the addition of salt.

Although this process is an English one, it was not worked in England until the year 1874, when works were built at Northwich, and a year later at Sandbach. In 1874 Solvay manufactured upward of 4,678 tons and it is evident that the works mentioned above were of the same capacity. They are very clearly described by him in the specifications of patents No. 999 and No. 1904, of 1876. The plant at Sandbach consists of lime kilns, four ammonia stills with heater and absorbers, five carbonating towers, four drying machines, besides other vessels. When working this plant showed a profitable return, but it was almost impossible to keep it continually working because of its extremely complicated parts, chiefly in the ammonia stills, the connecting pipes becoming blocked and the changing valve leaking.

Moreover, the lime, being charged unslaked, was very badly burned.

Through these causes it might safely be said that the plant did not work more than two-thirds of the time, yet with all its drawbacks this plant was working in the year 1881. This plant could never have competed favorably with the Leblanc process and the ammonia-alkali industry was in a most critical condition when the late Dr. Ludwig Mond, from the capacity of the ammonia distilling apparatus of this plant, designed the vertical still afterward patented by him in the year 1883.

This still is a fine example of chemical engineering, as it shows how the principles of engineering may be applied in the construction of chemical plant; it was such an improvement over the four-still arrangement that the daily output was increased by more than one-third, that is, from 13 tons to more than 21 tons, without increasing the other parts of the plant; in fact, this vertical still made the plant almost continuous, having only to be stopped a day in about six weeks.

This plant was duplicated and worked most successfully for many years. A vertical still had been used in distilling ammoniacal liquors long before this time, and it seems strange that it was not used at first, as it was much cheaper to construct. Solvay describes one in his specifications of patents for 1876.

The carbonating tower patented by Solvay in 1872 is also an apparatus which shows a considerable amount of engineering skill and may be considered his best contribution to this particular process and it still remains a part of the present-day plant with the addition only of an internal cooling coil.

Although the patent protecting this Solvay carbonating tower, which was so ingeniously designed and which produces the most perfect results in working, became void in 1886, another works was established at Runcorn which contained a plant much simpler in construction than that of Solvay; in fact, it consisted chiefly of large cylindrical vessels made of boiler plate to the number of about 30. It was worked as follows:

The ammoniacal gases from the stills, without cooling, passed into vessels called saturators which contained salt-brine and into which solid salt was also charged by means of a cage running down the length of the saturator, so that brine was made and saturated with ammonia in one operation. The ammoniated brine so produced was similar to that obtained in the Solvay system.

The great drawback of this method was that much salt became deposited in the saturators, sometimes to the depth of 12 ft. or more, which made it difficult to estimate the ammonia in them, a matter of the greatest importance in this process. The stills in this department on several occasions recovered upward of 620 lbs. NH_3 in the hour.

From the saturators the ammoniated brine, after cooling and settling, was passed into vessels called absorbers, two in number, so that while one was being

filled with ammoniated brine the other was working as a part of the carbonating plant. The carbonating plant consisted of three vessels, each containing about 600 cu. ft. of ammoniated brine. The carbonic acid gas from the lime kiln, after being washed and cooled and at an initial pressure of 38 lbs. per square inch, passed through these concentrators, through an absorber, then through a tank containing brine and lastly through a vessel containing acid.

Five carbonators were worked off during the day of 24 hours and produced about 10 tons of soda ash. The washing of the soda bicarbonate was so carried out that about 20 per cent of it was washed away and lost. Analyses of residual liquors from above carbonators have already been given above (Table No. 2).

By the Solvay towers the 600 cu. ft. of ammoniated brine could have been worked off in less than three hours, that is, in the state of carbonation as it passed into these carbonators, and at least 15 tons of soda ash might have been produced instead of 10 tons. The plant was supposed to contain a quantity of ammonia equal to 55,000 lbs. NH_3 , a quantity quite sufficient for a daily output of 35 tons of finished soda ash. Sixteen tons of limestone were also burnt per day. A particular feature in this plant was that the vessels were so placed that the liquor passed by gravity from vessel to vessel so that pumps were not required. This works was not a success.

At Middlewich, in 1889, a plant very similar to the one already described was erected, but on a much more extensive scale. It contained two large ammonia stills very similar in construction to the Mond still mentioned in the Solvay system, except that the superheater was connected by pipes instead of being built in one column. It also contained four sets of carbonators similarly worked as at Runcorn, and four lime kilns. With a little ingenuity this plant might have produced at least 600 tons of soda ash per week; with the whole plant working only 390 tons were made and with half the plant but 200 tons of soda ash were produced per week.

To illustrate this point two examples will be given. When the whole plant was working and contained a quantity of ammonia equal to 123,000 lbs. NH_3 , 104 tons of limestone were burnt, 16,000 cu. ft. of residual liquor distilled, 15,000 cu. ft. of brine used; 55.8 tons of soda ash were produced in the day of 24 hours. To effect this about 20 carbonators were worked, each containing 840 cu. ft. of ammoniated brine.

Another example when only half the plant was working and contained a quantity of ammonia equal to 57,000 lbs. NH_3 ; 6,600 cu. ft. of residual liquor were distilled, 6,600 cu. ft. of brine used; 23.68 tons of soda ash were produced; about 10 carbonators were worked off. The loss of ammonia in the working of this plant varied from 3 per cent to 15 per cent, that is, parts of ammonium sulphate per 100 parts of soda ash. This works in the year 1893 with a capital of some £180,000 lost £10,689 on the year's working.

Another works with about the same amount of capital paid only $7\frac{1}{2}$ per cent dividend. In fact, three works of very similar construction were bought up by Messrs. Brunner, Mond & Co., Ltd. If these works had used the Solvay tower or had used more engineering skill in the construction and working of their plants much more favorable returns would have been made.

Similar intermittent plants have also been worked in different parts of Germany about 30 years ago, all of which are said to have failed. None of these intermittent systems could ever succeed against the Solvay system and the price of alkali need not have been lowered to £2.175 6d per ton, as it must have cost more than £4 per ton to produce soda ash by them.

According to the census returns, the growth of the silk manufactures of the United States has been as follows for the series of years given: 1870, \$12,000,000; 1880, \$41,000,000; 1890, \$87,250,000; 1900, \$107,250,000; 1905, \$133,250,000, while it is estimated that the returns of production for 1910 will run considerably over \$150,000,000. The number of silk establishments has increased from 67 in 1850 to 624 in 1905.

During 1909 the quantity of gypsum mined in the United States was 2,252,785 short tons, an increase of nearly 31 per cent over the production of 1908, which was 1,721,829 short tons, and an increase of more than 28 per cent over that of 1907, which was 1,751,748 short tons. The gypsum sold without calcining and used principally as land plaster and as an ingredient in Portland cement and in paint showed a large increase in quantity, but a loss of about 11 cts. per ton in value; but the material calcined for plaster showed both a large increase in quantity and an increase of 30 cts. per ton in selling price at the mills. The total value of gypsum and gypsum products in 1909 was \$5,906,738, as compared with \$4,075,824 in 1908, an increase of \$1,830,914, or 44.9 per cent.

A recent consular report states that a series of prolonged investigations has led to the conclusion that the "peeling" of silver from plated articles is not due to insufficient plating, but to the high content of nickel in the basis metal; that ordinary commercial alloys of copper, zinc, and nickel containing more than 14 per cent of nickel are of doubtful utility in this respect; and that the tendency to strip increases with the thickness of the silver deposit. The properties considered in deciding upon the grade of German silver to be used as a basis metal for electroplating were strength, color, and malleability. However, where thick coatings of silver are concerned whiteness should not be considered as a matter of such prime necessity as the perfect adherence of the silver under conditions of wear, since strength is of more importance than color.

THE WHITING ELECTROLYTIC CELL.*

By JASPER WHITING.

The Whiting Electrolytic Cell is the result of an attempt to improve that type of apparatus which employs mercury as an agent in the electrolytic decomposition of alkali chlorides.

The chief stumbling block in the way of the suc-

cessful operation of mercury cells is the difficulty of removing the sodium amalgam in a uniform and continuous manner from the decomposing chamber, where it is formed, to the oxidizing chamber. If this amalgam is not so removed, it tends to accumulate in the decomposing chamber until, when a certain concentration is attained, violent secondary reactions are set up with consequent lowering of the efficiency of the cell. In the ordinary type of apparatus the mercury is

caused to flow in a wide and shallow channel from one compartment to another. Hydraulic engineers have long recognized the fact that when a stream of water flows through such a channel, the liquid travels much more rapidly in the center than on the sides, because the friction of the walls of the channel holds back the material adjacent thereto. If this is a serious difficulty with such a simple fluid as water, it is doubly true of a complex fluid such as mercury and

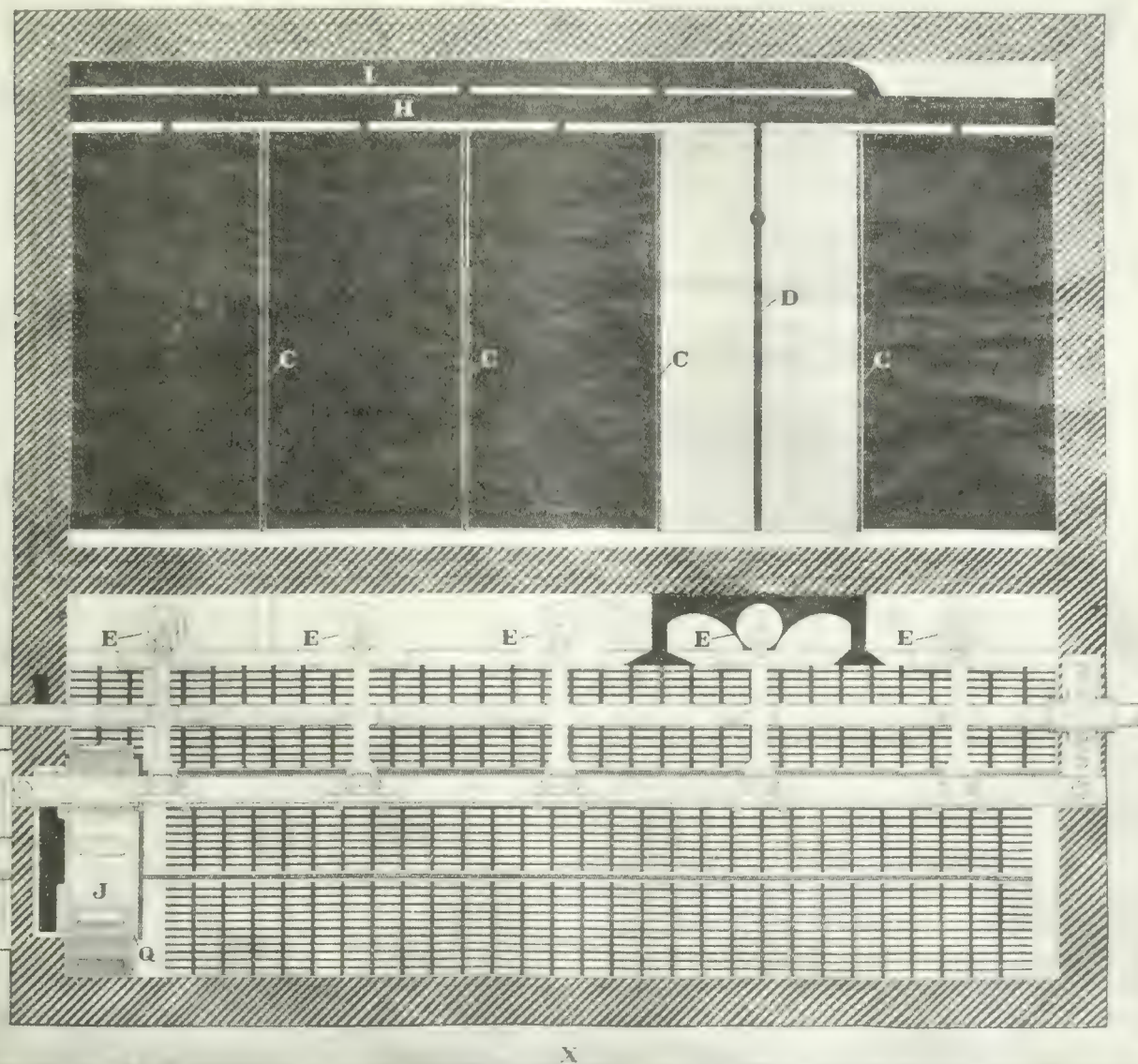


Fig. 1—Whiting Electrolytic Cell—Plan.

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its overlying stratum of sodium amalgam. As a result, the sodium amalgam tends to accumulate to an excessive degree, with consequent lowering of the current efficiency.

In the Whiting cell a principle new to this type of apparatus is employed for the purpose of overcoming the above difficulty. This cell operates intermittently. A body of mercury is placed in the decomposing compartment and held there in a stationary position as regards the anodes for a given length of time, during which sodium amalgam of the desired concentration

*From a paper read at the 17th general meeting of the American Electrochemical Society, in Pittsburg, Pa., May 7, 1910.

is produced. This amalgam is then completely drawn off from the decomposing compartment, and when removed its place is taken by another body of mercury free from sodium. By placing a number of these compartments in parallel, and operating them successively, the cell, while intermittent in principle, is continuous in its action.

The Whiting Cell is a massive concrete structure, supported on four concrete pedestals, but insulated therefrom. It consists of a shallow receptacle, or box, divided into two compartments, *A* and *B*, by a concrete partition. The bottom of the larger, or decomposing compartment *A*, is divided by low glass partitions *C* into a number of sections. The sections have V-shaped bottoms, sloping at a slight angle towards central slots *D*, these slots being carried through the dividing concrete partitions into the small or oxidizing compartment *B*, where they turn upward and are closed by poppet valves *E*, operated by cams *F*, attached to a slowly revolving shaft *G*. The other ends of these slots are connected one with another by means of a common channel *H*, called the distributing level, and this channel connects in turn with a secondary channel or run, *I*, which leads through one of the side walls of the cell to a pump *J*, at the extreme end of the oxidizing compartment. Mercury covers the bottom of the decomposing compartment, filling to a common level the several sections above described. The anodes *K*, consisting of slabs of Acheson granite, perforated to permit the free discharge of chlorine, rest upon ledges *L*, placed at the ends of the sections in such a way that the distance between the electrodes shall be a minimum. To these anodes are fastened graphite leads *M*, which extend upwards through the cover *N*, connecting with the busbar.

The oxidizing chamber is divided into three channels *P*, lined with graphite of special design, and sloping successively downwards to form a zig-zag path to the pump-pit *Q*, where a stoneware rotary pump *J*, also of special design, is placed, and operated by auxiliary means. A common body of electrolyte, brine, fills the decomposing chamber, and water, or caustic, the oxidizing compartment.

The action of the cell is as follows: The floor of the several sections of the decomposing chamber is covered with mercury, which is maintained at a common level by means of the distributing level. The current, supplied to the anodes from the busbars, flows through the brine solution to the mercury and out through the bottom of the cell through iron rods *R*, partly imbedded in the concrete and serving to connect the mercury in the cell with the cathode busbar.

The current passing through the electrolyte decomposes the brine, chlorine is given off at the anodes, escaping upwards, and sodium is separated at the surface of the mercury, amalgamating therewith. When this action has proceeded for a predetermined period of time, approximately two minutes, the poppet valve at the point of exit of one of the sections is opened

by the action of the cam, and the entire mass of sodium amalgam contained in that section sinks rapidly into the slot and down through the connecting pipe into the oxidizing chamber. The period required for the emptying of a section having been accurately determined, the design of the cam is such that when the mercury is all out of the section the valve closes. Mercury free from sodium then flows into the empty chamber by way of the distributing level, and continues to flow until the common level is established. In the meantime the sodium amalgam in the oxidizing compartment flows by gravity successively over the graphite plates, and in so doing is freed from the sodium it contained, only pure mercury reaching the pump-pit, this pure mercury being then raised by the rotary pump into the wall pipe of the decomposing compartment, through which it flows by gravity to the distributing level. Thus is the cycle completed.

By operating the sections successively, the action of the cell is, in effect, continuous, though the action of each individual section is intermittent.

It is easy to see the advantages of the electrical conditions in the decomposing compartment. A body of mercury is placed in a stationary position as regards the anodes, and at such a distance from them as is theoretically the best; therefore, the distribution of the current is uniform and the resistance of the electrolyte a minimum. Moreover, as there is no agitation of the mercury during the period of electrolysis, secondary reactions are not facilitated, and as the initial primary conditions are re-established at frequent intervals, uniformity of operation is maintained.

In order to operate the cell satisfactorily, it is necessary to feed brine of great strength uniformly between the electrodes. As the electrodes are very close together, and as the area of the cell is considerable, this appeared at first to be a serious difficulty. The problem was solved by utilizing the intermittent action of the mercury in the sections to control the operation of a device for feeding the brine. Cup-shaped receptacles *S*, equal in number to the sections of the decomposing chamber, are formed in the cover *N* of the decomposing compartment, these receptacles being connected by a common channel *T* at their upper extremities. Through the bottom of these cups, glass tubes *U* extend downward between the anodes, the end of each tube terminating below the surface of the mercury, at a point near the middle of each section.

Brine is fed into one of these cups from an outside source, and when the cup is full the brine overflows into the channel, filling each of the cups in turn. As long as the sections are filled with mercury, the bottom of the tubes are sealed, so that no brine can flow into the cell; but as soon as the mercury is withdrawn from one of the sections the seal is broken on the corresponding tube, and a definite quantity of concentrated brine, corresponding to the size of the cup, flows into that individual section. As this brine has a greater specific gravity than the partly electrolyzed

brine in the cell, it sinks to the bottom and spreads laterally, forming a layer about 1/16 in. (1.56 mm.) thick over the glass plates. When the purified mercury again fills the section, the new brine, free from chlorine, acts as a protective layer to it, and is in its proper position for being electrolyzed. Moreover, as the brine enters the cell at a time when the section is empty, it causes no agitation of the mercury, and consequently facilitates no secondary reactions.

The action of the oxidizing chamber of the cell could hardly be more simple, and it has the distinct ad-

We have overcome this trouble by means of a very simple device. Holes about 1/8 in. in diameter and 1/2 in. deep are bored at frequent intervals in the graphite slabs, and these holes are filled with pure mercury before starting to operate the cell. This pure mercury makes, of course, an effective contact with the graphite walls of the holes, especially as it is under a relatively high head, and, as it is also in contact with the sodium amalgam which flows above it, it forms a connecting link between the sodium amalgam and the graphite. Moreover, as sodium amalgam is

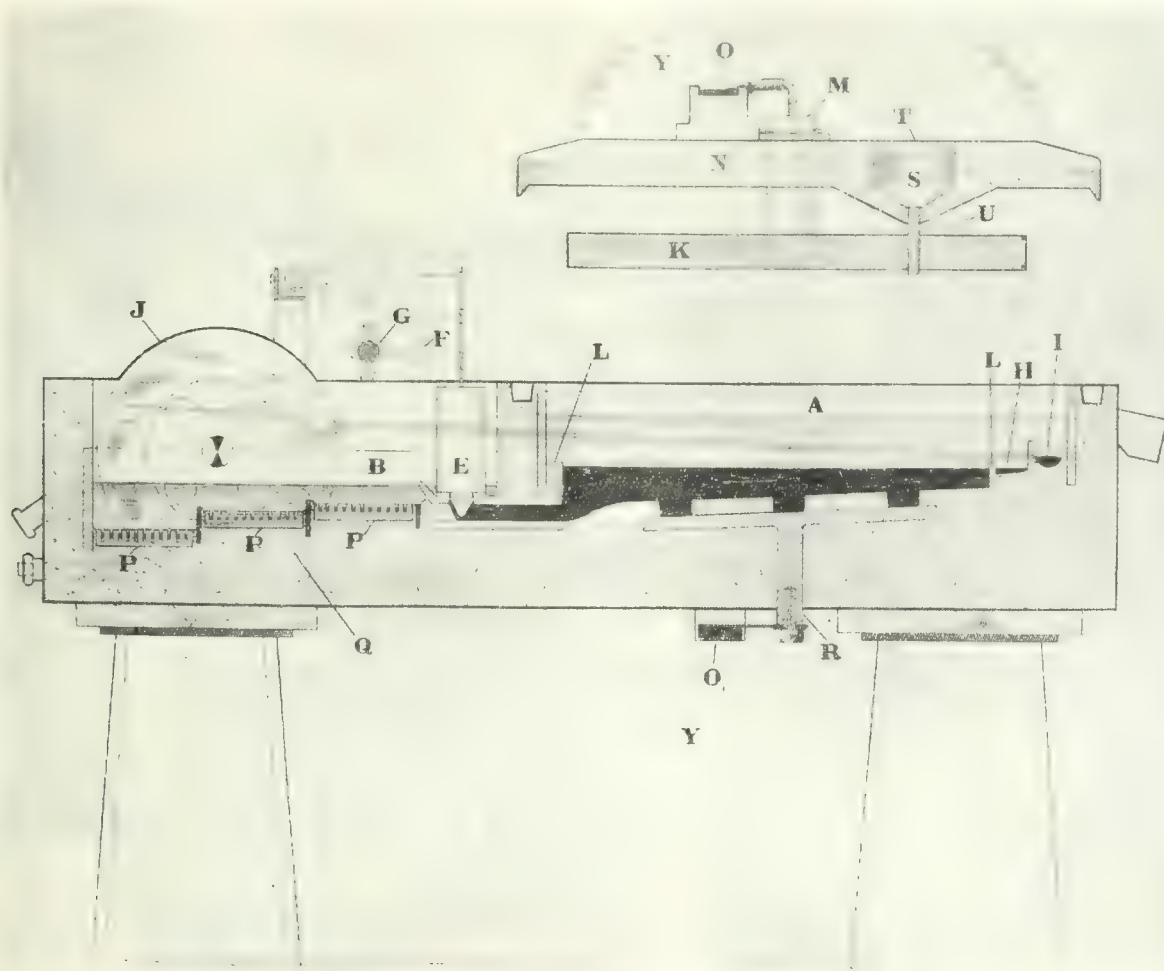


Fig. 2—Whiting Electrolytic Cell—Cross Section.

vantage of being automatic. The graphite slabs are so designed as to form a large number of channels through which the mercury flows. The sides of these channels extend upwards into the caustic solution, and form, with the amalgam, a galvanic couple, which serves to decompose the amalgam. The principle of this action is by no means new, but so far as I know it has never been successfully employed in an electrolytic cell until now. The difficulty lay in maintaining a proper contact between the amalgam and the graphite. It is easy to secure contact at first, but after the action has proceeded for a varying length of time, the graphite tends to become polarized by the hydrogen given off at its surface, until the necessary contact between it and the amalgam is destroyed.

lighter than mercury, and as the holes are narrow and deep, there is little tendency for the sodium to diffuse downwards; consequently the mercury in the holes remains pure, and, as there is no action between it and the graphite, there is no serious polarization of the walls of the holes, and, as a result, the original contact between the mercury and the graphite is not destroyed.

In practice, this galvanic action proceeds at a very vigorous and uniform rate until all the sodium in the amalgam is oxidized, when it automatically stops, pure mercury being delivered to the rotary pump to be returned to the decomposing chamber.

This method of oxidizing amalgam is superior, I think, to that employed in any other cell operating on

a commercial scale. It requires no attention, being at all times automatic, and is free from mercury losses. Caustic of almost any desired concentration may be produced. It is only necessary to regulate the amount of water fed into the chamber to produce the strength desired. The caustic formed is, as already stated, practically pure.

One of the features of the Whiting Cell is its accessibility. This is largely by reason of the construction of the cover of the decomposing compartment. This cover is made of reinforced concrete, with edges extending downward to loosely fit into channels made in the top of the wall of the decomposing compartment, thus forming a gas-tight, liquid seal. The anode leads extend upwards through the cover and, while removable, are so arranged that when the cover is raised, the anodes are raised also. To the cover, likewise, are fastened the busbar and the system for feeding the brine; consequently, when the cover is lifted, all the auxiliary apparatus in connection with the decomposing chamber of the cell is removed in one operation, permitting a rapid inspection and cleaning of the cell.

There is, of course, no fixed limit to the size of the Whiting Cell. As each section works independently, one from another, the sections may be multiplied in number as desired, until a cell of capacity suited to the conditions is produced. The cell which has been developed at the plant of the Oxford Paper Co., at Rumford, Me., is about 6 ft. (1.8 meter) square over all. It contains five sections, and is operated with a current of from 1,200 to 1,400 amperes, equivalent to a current density of over 100 amperes per square foot (11 A. per sq. dm.) of active anode surface. The voltage across the terminals is approximately 4, and the cells operate at a current efficiency of from 90 to 95 per cent.

The Whiting Cell is, I think, the only cell which can be shut down without noticeably affecting its efficiency, or without injury to the apparatus. To illustrate this in a practical way, during a test run of two weeks' duration in our preliminary experimental plant of four cells, the current was shut off for various reasons eleven different times, for periods varying from a few minutes to twelve hours, but, in spite of these many shut-downs, the plant maintained an average current efficiency of over 95 per cent. In case of shut-down it is only necessary to stop the operation of the pump, when all the mercury is automatically drawn off from the decomposing to the oxidizing compartment, where it remains until it is again desired to start the cell, which is accomplished by the starting of the pump. As the mercury is all out of the decomposing compartment while the current is off, there is no loss of mercury due to the corrosive action of the chlorinated brine, as is the case with other types of mercury cells.

Moreover, the Whiting Cell is singularly free from injury on account of accidents. Should there be an unexpected stoppage of the pump, the mercury leaves the decomposing chamber as above described. Should

one of the valves leak, the level of the mercury in the decomposing chamber will be slightly affected, with a corresponding increase in voltage, but no serious damage will be done. Should one of the valves fail to operate, sodium would accumulate in the mercury of that section, but the remainder of the cell would operate satisfactorily. Occasionally, the cell has to be washed, to free it from precipitated graphite dust, and from slime which forms on the glass plates. The operation of washing a cell is, however, an extremely simple one, and requires the labor of but one man for about half an hour.

As the cell is operated at a temperature below 40° C., the tendency to form chlorates is very small, and consequently the deterioration of the graphite anodes is reduced to a minimum. There is, of course, no depreciation of the graphite slabs in the oxidizing chamber of the cell.

As before stated, the mercury type of cell produces pure caustic. Consequently, our cell efficiency, as far as the caustic is concerned, is our plant efficiency. There are no losses incident to subsequent treatment. What flows out of the cell is our finished product, assuming, of course, that liquid caustic is sold. The chlorine gas escaping from the cell is 98 per cent pure, the remaining 2 per cent being hydrogen, and, as the chlorine leaves the cell at a relatively low temperature, it is unnecessary to cool it further before using it to form bleaching liquor.

The Whiting Electrolytic Cell is not an expensive cell to install. It is generally believed that mercury cells are very costly, for the reason that large quantities of mercury are required in their operation, and mercury is, of course, a high-priced material. When one considers, however, that the use of mercury eliminates the necessity of installing very expensive and rapidly deteriorating multiple-effect evaporating machinery, and auxiliary apparatus for the purification of the caustic produced in diaphragm cells, it is easy to see that one expense largely offsets the other. Moreover, mercury is a quick asset convertible into cash at a high proportion of its first cost, whereas secondhand machinery is of little value. It is my belief that a plant of a given capacity employing the Whiting Electrolytic Process would not cost materially more than a plant of similar capacity employing any one of the present available diaphragm processes.

The present plant, erected for the Oxford Paper Company, has a capacity in bleach liquor equivalent to about five tons of bleaching powder per day, and produces, in addition, about two tons of pure caustic soda. This plant has been operating since early in January, and the results obtained from it have been so satisfactory that its capacity is now being increased five times.

I desire, in ending this paper, to acknowledge the aid rendered in the development of the Whiting Process by Mr. William Hoopes, of the Aluminum Company of America; by Mr. W. R. Mott, who served for a few months as my assistant, at the time I first began ex-

perimenting; by Prof. C. F. Burgess, of the University of Wisconsin, who furnished valuable suggestions at a later stage in the cell's development; and by Mr. A. M. Hamblet, the present superintendent of the plant at Rumford, whose assistance during the past two years has been most valuable.

DISCUSSION.

PROF. C. F. BURGESS: I have observed Mr. Whiting's cell in its development, and have been impressed with the wide difference between a laboratory experiment and a commercial operation; also particularly with the "unlawful" behavior of sodium amalgam and chlorine. I hope some day that Mr. Whiting will give

cathode liquor obtained from the cell and how often do the cells have to be washed?

MR. JASPER WHITING: We use 350-375 pounds of mercury per cell, equivalent to 3,500-3,700 pounds per ton of bleach made per day. There is constant loss due to the solubility of mercury in the alkali and brine solutions, but this only amounts to approximately 2 per cent of the total per year. There are, of course, other losses, due to the mechanical handling, etc., which might bring the total up to 6 per cent per year. We have figured on 10 per cent in our estimates in order to be conservative.

The caustic solution as produced at present contains 20 per cent sodium hydrate. We have made it up to

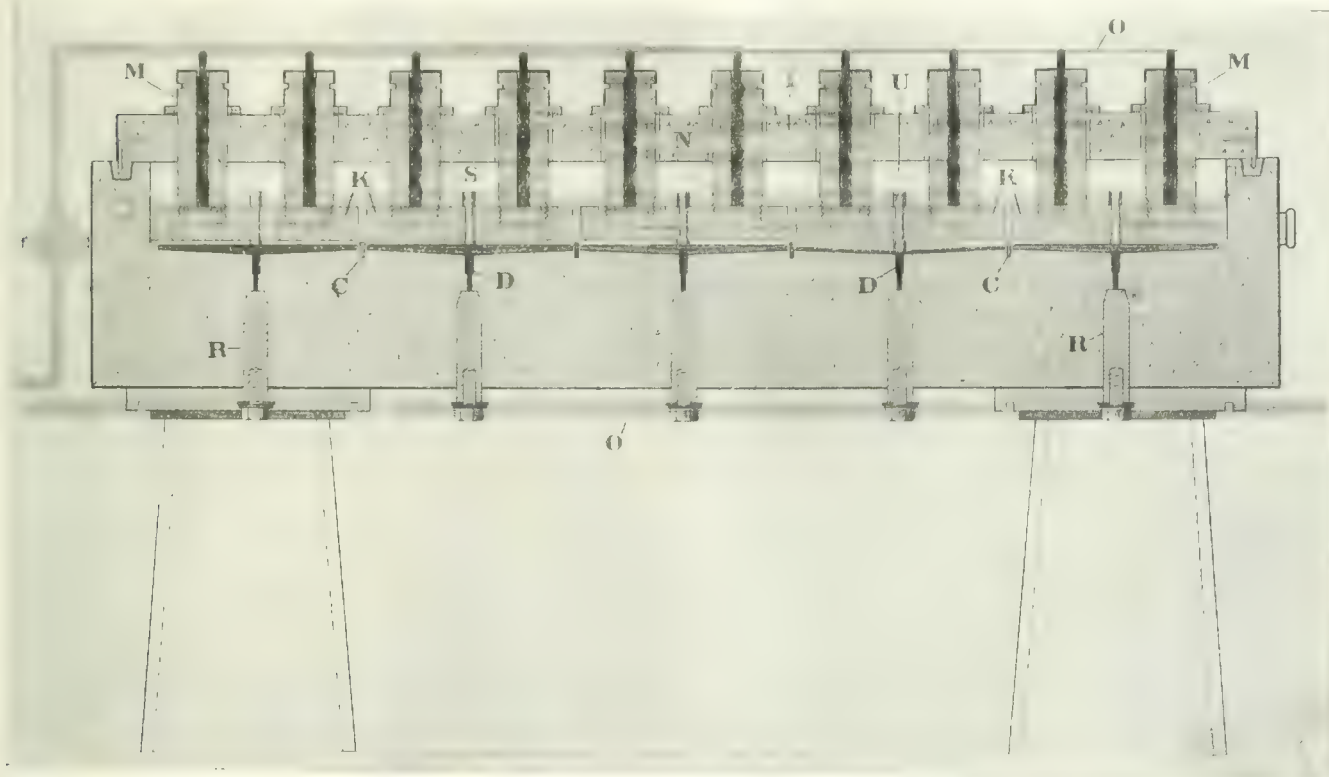


Fig. 3—Whiting Electrolytic Cell—Longitudinal Section.

us a paper on "Some Troubles I Have Met and How I Overcame Them."

CHAIRMAN BAEKELAND: All electrolytic cells work better in a laboratory than in practice; one can easily change the details of operation in a laboratory apparatus, but it may cost thousands of dollars to interrupt operation in a factory plant. I believe almost every one of us at some time or another of his life, has invented some electrolytic cell. Fortunately, the development of most of these inventions has been arrested before hundreds of thousands of dollars have been spent upon it. The fact that the cell described by Mr. Whiting is used in practice, increases considerably the value of his communication.

I would like to ask Mr. Whiting how many pounds of mercury he uses to a cell of 2,000-2,500 amperes, and also whether any losses of mercury occur during prolonged operation? What is the strength of the

40 per cent, and even more concentrated solutions can be made, we believe, if desirable. The frequency of washing varies with the conditions of operation. I should say that the cover of the cell is raised once a month on the average.

CHAIRMAN BAEKELAND: A rich cathode liquor is quite desirable, and if Mr. Whiting can produce easily 40 per cent solutions, this is a decided advantage for many purposes where no further concentration is necessary. The frequency of washing his cell corresponds about to that of the better class of the diaphragm cells.

DR. J. W. RICHARDS: Is there any loss of mercury by vaporization or by carrying off as vapor by the evolved hydrogen. Have any effects of salivation been noted on the workmen?

MR. WHITING: The mercury in our cells is always covered with solution and therefore cannot vaporize. We have experienced no trouble whatever with the

possible evil effects of mercury vapor upon our workmen.

CHAIRMAN BAEKELAND: How much hypochlorite or chlorate is formed in your anode compartment?

MR. WHITING: We have determined the amount, but I have no figures relating to this matter with me. We make a strong brine and use it in a closed system until the hypochlorites, sulphates, etc., become too concentrated, when we then throw it away and start anew. We have not yet determined how often this is necessary, but this varies a good deal with the purity of the brine and with other conditions. The impure liquor is thrown away, as it costs more to purify it than it is worth.

CHAIRMAN BAEKELAND: If chlorates and hypochlorites occur in considerable quantities in the anode liquor, they are continually electrolyzed in their turn, and form oxygen at the anode; after a certain percentage of these oxysalts has been reached, they remain constant. The loss of current in electrolyzing these compounds is small, but their chief drawback is the corrosion of the graphite anodes.

The anode question is a very serious one, because anodes form a large item in the cost of these processes. Dr. Acheson rendered a great service in furnishing the electrolytic industry with graphite electrodes, which are more lasting than the older carbon electrodes. Platinum is too expensive to use. Graphite anodes are less expensive, but they corrode; in laboratory experiments this corrosion may be small, but in practice, where large groups of cells are run by ordinary workmen instead of by experts, if ONE cell runs badly the circulating electrolyte vitiates the whole brine system and lowers the efficiencies in the cells and causes corrosion of the anodes. At the end of the year the books show high anode costs. The corrosion of the anodes not only causes a loss of anodes but increases the internal resistance by disarranging the cell conditions, especially as far as distance between electrodes is concerned. For instance, if the cells have been designed for a working distance of $\frac{3}{8}$ in. between the anode and cathode, after corrosion the ends and faces of the anodes will become very irregular and cause very different current densities at different parts; with the results of producing very different electrolytic conditions from those for which the cell was designed. Such small details as these are of the greatest importance in commercial work, and frequently decide the practical value of a cell.

DR. RICHARDS: Would a paraffined graphite electrode, such as Mr. Turrentine uses in his electrical analyses, decrease the corrosion?

CHAIRMAN BAEKELAND: We have tried paraffin on graphite anodes but it is quickly attacked by chlorine. In Germany other anodes have been tried, such as cast iron oxide electrodes. These resist chlorine, but there seems to exist some difficulty in producing this kind of anode.

MR. WHITING: We do not paraffin our anodes. We have tried paraffining them, but without success.

CHAIRMAN BAEKELAND: In commercial operations such small details as proper connections for the anodes may give much trouble, and often devices which look clumsy on first sight give better results in practice than connections designed with much care and forethought and which, notwithstanding their apparent superiority, cause bad contacts and increased voltage. It has happened that a cell however carefully planned in principle lost its advantages by some such seemingly trivial detail.

A general plan of calcining plaster—the size and weight of machinery depending upon the capacity desired—is given as follows in a report of the U. S. Geological Survey on the gypsum industry in 1909: The gypsum rock is crushed first in a jaw crusher; second, in a pot crusher; and then it goes to a rotary kiln drier. This drier is erected in brickwork like a boiler, and is equipped with an automatic feeder. If soft coal or wood is used as fuel, care must be taken that the products of combustion do not come in contact with the materials being dried, on account of the danger of discoloration. Fuel of any kind—oil, gas, coke, wood, or coal—is suitable. This drying process eliminates 10 per cent of moisture. Next, the crushed rock is sieved in a trommel, generally to 24 mesh. The material that does not pass the sieve is ground in burr mills, and this product, with the screenings from the trommel, is ready for boiling. The boiling is done in a large kettle with wrought-steel sides and cast-iron or very heavy steel convex bottom. Flues pass through the kettle near the bottom and distribute the heat, which is applied below the kettle and passes around the lower part of the sides, through the flues, and then around the upper part of the sides and out at the stack. Inside the kettle is a shaft, which propels stirrers below the flues and mixing paddles above. The kettles are heavy and rest on brickwork. The ground gypsum is fed from bins into the kettle, and is constantly stirred and boiled until the remainder of the free moisture is expelled. The temperature of this preliminary boiling should not exceed 265° F., for at a higher temperature the water of crystallization, or combined water, begins to separate, and then the separation must be completed or the calcination will be a failure. To remove the necessary three-fourths of the combined water the material is then heated steadily to a temperature of 390° to 395° F. Care must be taken not to allow the temperature of this second boiling to exceed 400°, or all the combined water will be expelled and the plaster will lose its setting properties. When properly boiled the gypsum settles and may then be discharged through a gate on the side near the bottom of the kettle. After boiling, the plaster should be screened again through 40-mesh wire cloth, and the oversize should be reduced in a finishing burr mill.

THE PURCHASE OF COAL ON AN EFFICIENCY BASIS.*

The most economical coal to buy is the one that will make steam for the least cost, all things considered. This does not mean necessarily the cheapest coal, or even the coal which will evaporate water at the least cost, considering the price of coal alone. These facts were the basis of a paper on the purchase of coal on an efficiency basis, which was read before the New England Water Works Association on February 9 by Mr. A. O. Doane, division engineer of the Metropolitan Water and Sewerage Board. The first step in formulating specifications, he said, is to find out what kind of coal will give the best results in the particular plant where it is to be used, and to make such requirements as will insure the delivery of the desired quality.

The principal points which should be covered by the specifications are as follows: The total amount of coal desired, the rate of delivery, the approximate proportions of lump and fine coal, a statement of the approximate composition of the coal desired, including the percentages of ash, sulphur and volatile matter; the heat units per pound desired, and the maximum limits of ash, moisture, sulphur and volatile matter which will be accepted. There should be stated the method of determining the amount to be paid for, whether by bill of lading or by delivery weights, the methods of taking samples, for conducting tests, for making corrections in contract price for quality, and there should be clearly defined a provision for settlement of differences which may arise between dealer and consumer. Provision should also be made for purchasing a limited amount of coal from other dealers for test purposes, if so desired, for procedure in case the contractor fails to deliver coal when required, and for increasing or diminishing the contract amount by a certain percentage if conditions of operation make such a course advisable.

Some of the considerations which ordinarily govern a purchaser in selecting the proper kind of fuel, and the effect of the impurities and other constituents of the coal, were outlined by Mr. Doane substantially as follows:

The location of the plant is an important matter, for it determines the freight rate and cost of whatever teaming may be necessary. As the transportation charges are generally the same per ton for all grades of coal, it follows that the larger the proportion of this expense is to the total cost of the coal, the more advantageous it is to use the higher grades of fuel. It is important to select a kind of coal which will not be subject to annoying and expensive interruptions in delivery.

If the transportation rates are such that the purchase of lower grade coal would apparently be profitable, the plant equipment and conditions of operation

should be carefully considered to see if there is sufficient boiler capacity when burning fuel of low-heating value, and if the grate and draft conditions are suitable. If the plant is not suitably arranged, a study should be made to determine if the probable saving resulting from using the low-grade coal will warrant the expense of any changes in equipment which may be necessary. A study of the labor conditions is also important, as the poorer the quality of coal the greater the amount that has to be handled, resulting in an increase in the work of keeping the fires in order and of removing the ashes.

The size of coal is also of considerable importance, uniformity of size, as a rule, giving the best results. Some of the best steam coal is of a very friable nature, and, if subjected to several handlings before reaching the consumers' bins, it may be delivered in a powdered condition. Coal in this form has a tendency to clog the air passages in the fuel bed, causing imperfect combustion, and to fall through the grate into the ash pit, thereby increasing the waste. The effect of fine coal is particularly bad when the smaller sizes of anthracite are used with it. On the other hand, some semi-anthracite coal is exceedingly hard and is delivered in large lumps, which break the coal-handling equipment and cause extra labor in the fire room, as the lumps have to be broken up with a heavy hammer before they can be burned.

The amount and composition of the ash have much to do with the value of a coal, for freight and handling charges must be paid on the incombustible material, and, in many cases, the ashes have to be carted away at considerable expense. A coal with a large percentage of ash clogs the fire, thus checking the draft and causing incomplete combustion and requiring more frequent cleaning of the fire, which causes a waste of coal, as a considerable amount of combustible material is carried into the ashpit with the clinkers. In such a case, moreover, the fire doors have to be opened more frequently, thus cooling off the gases and causing unequal expansion in the boiler plates, and the ash is heated to the furnace temperature and much of this heat is wasted. The fusible constituents of such a coal form clinkers, which increase the labor of firing and interfere with proper combustion. The percentage of ash may be considerably reduced by careful preparation at the mine.

There is some moisture, generally about 1 per cent, in the coal when mined, and as coal is transported in open cars, the amount of moisture may be either diminished or increased, according to weather conditions prevailing during the time the coal is exposed in the cars. Ordinary bituminous coal will hold about 5 per cent of moisture without dripping. This water is, of course, worse than useless to the consumer, as he has to pay all charges on it, and, in addition, loses the heat required to evaporate it and superheat the steam in the furnace. Coal containing a large amount of moisture, if delivered in cold weather, is exceed-

*From the Engineering Record.

ingly hard to unload, as the contents of the cars are frozen into a solid mass, requiring hard labor with the pick to break.

Sulphur generally occurs in coal as iron pyrites, sulphide of iron, sometimes in cubical crystals resembling glass or in combination with lime as sulphate of calcium or gypsum. It also sometimes occurs in a free state.

While sulphur is combustible, its heating value is less than one-third that of carbon. A considerable amount of the sulphur in coal does not burn, but remains in combination with the other constituents of the ash, causing the worst kind of clinkers, particularly if the coal is high in sulphur and iron; with a small percentage of ash, it forms a tough slag, which is almost impossible to break up, which in some cases unites with the cast-iron grate bars and quickly ruins them. The sulphur which is burned forms a gas which, in the presence of moisture, generates an acid which quickly corrodes the iron work of flues and stacks, while the ashes and clinkers which gather on metal surfaces also have a powerful corrosive action if they become moist.

It was formerly thought that sulphur was the cause of the so-called spontaneous combustion of coal, but, although it is a fact that coals containing a high percentage of sulphur often give trouble from heating, there are coals which contain but a small amount of sulphur in coal and is far too small to account for the rise in temperature of the pile, even if it were all oxidized, which is never the case. While the cause of heating is not fully understood, it appears to be due to the structure and mechanical condition of the coal rather than to its chemical composition. Some coals seem to have the property of absorbing oxygen, with consequent rise in temperature, and as the heat increases, the affinity for oxygen seems to increase until, under favorable circumstances, the coal becomes ignited.

Finely divided, freshly mined coal having a considerable amount of volatile material and moisture and stored in large masses, is apt to heat. As a rule, the heating is not uniform in the pile where favorable conditions exist, such as near partitions or around pillars, where a supply of oxygen may be had without sufficient radiation to remove the heat as fast as formed. The small ventilating pipes which are sometimes installed with a view to preventing heating furnish ideal spots for starting fires, in Mr. Doane's opinion.

The effect of volatile matter in coal depends largely on the type of furnace and boiler in which it is to be used, as with proper equipment coal containing very large percentages of volatile matter may be burned with economy and practically without smoke. With the ordinary steam plant, however, coal having a volatile content below 22 per cent will prove more efficient and give less trouble on account of smoke.

The combustible volatile matter consists principally of hydrocarbons, methane or marsh gas (CH_4) being the most important. Ethylene olefiant gas (C_2H_4) is

also a constituent. A considerable proportion of the volatile matter is noncombustible, consisting of water of composition which is not driven off at the temperature of the drying oven. The inert matter varies from about 4 per cent of the total weight in eastern coals to 14 per cent in western coals. There is also from 1 to $1\frac{1}{2}$ per cent of nitrogen and small quantities of carbonic acid gas (CO_2). The combustible portion of the volatile matter has a high heating value, owing to the hydrocarbons. Methane has a calorific capacity of 23,600 while carbon has 14,544 and sulphur 4,050 b. t. u. per pound. Consequently, a high volatile coal will often give a high result when tested in the calorimeter, but when burned in the furnace the evaporation results may be disappointing unless the equipment is suitable. The volatile matter distills off rapidly from the green coal, tending to lower the furnace temperature below the ignition point; it is also difficult to get the gases mixed with the proper proportion of highly heated air and provide space enough for them to be completely burned before coming into contact with the water-heating surfaces of the boiler.

For this reason it is important to limit the percentage of volatile matter to an amount which will give satisfaction in the particular plant where the coal is to be used.

The Linde Air Products Co., which has two plants in operation in this country—one at Buffalo and one at East Chicago, Ill.—has recently increased its capital from \$500,000 to \$1,000,000 and has acquired sites at North Trafford, Pa., and at South Elizabeth, N. J., for two new plants. The two plants will be completed and in operation by June of this year. Contracts also have been placed for the equipment. Part of the machinery, however, will be built at the company's own shops in Buffalo. Demand for oxy-acetylene apparatus has greatly increased in Pittsburg district in the past 12 months, and has necessitated the establishing of a plant at Pittsburg. The company, which is an English corporation, makes liquid air, which is supplied to steel casting plants, steel works and foundries. The Trafford plant will permit of the construction of pipe lines leading from the factory to the important consumers. This proposed line will lead to all of the East Pittsburg and Wilmerding plants of the Westinghouse companies, and also to McKeesport, East McKeesport and Homestead. The new plant will be within convenient distance of the new Westinghouse foundries at Trafford city. The first of the buildings of the new works will be of steel and brick, 152 ft. wide and 117 ft. deep. Other structures will follow as the business develops.

An official report estimates the number of Para rubber trees under cultivation in Cochin China at over 1,000,000, of which about 15,000 are being tapped. Present plans contemplate the planting of 4,000,000 trees additional as soon as the work can be done.

DIRECT MANUFACTURE OF SULPHATE OF AMMONIA—NOTES ON SOME EXPERIMENTS.*

By G. STANLEY COOPER, B. Sc.

For some months the writer and a colleague have been engaged in experimental work on the manufacture of sulphate of ammonia by the direct method. Various workers have devised schemes on similar lines to those published by us, and many investigators still find the same old trouble in dealing with the neutralization part of the process. If the acid in the saturator is too strong, deposition of sulphur and tar takes place. A strong acid is desirable, as it conduces to the production of solid sulphate of ammonia with a minimum of waste heat. If the acid is weak, sulphur and tar do not tend to deposit very much, but a solution only of sulphate is produced, and no solid salt. This entails a separate evaporation, in order to procure crystals, and this, of course, means additional expense both in heat and in wear and tear of the vessel. The introduction of a steam coil for evaporation purposes by the Chemical Engineering Company is a good move, but here, still, there is the cost of steam, which is not negligible by any means.

There has been introduced on the Continent various plants worked on the system of Herr Karl Burkheiser—a truly direct process, for there the sulphur in the gas is converted first into sulphur dioxide, and then into sulphate.

The writer's efforts have been directed towards the working of a plant in which a fairly strong acid could be used, thus reducing the necessity for much evaporation. It was necessary, therefore, at the same time to devote attention to the extraction of as much tar as possible, and also to prevent the deposition of sulphur in the acid by the action of the sulphuretted hydrogen. This gas acts on the acid (if the latter be too strong), and sulphur is deposited: Thus: $\text{H}_2\text{S} + \text{H}_2\text{SO}_4 = 2\text{H}_2\text{O} + \text{SO}_2 + \text{S}$.

Any sulphur dioxide so produced is also acted on by the sulphuretted hydrogen, and more sulphur is formed. Thus: $2\text{H}_2\text{S} + \text{SO}_2 = 2\text{H}_2\text{O} + 3\text{S}$. If these reactions take place in the saturator, the sulphate becomes contaminated with the sulphur, which is, of course, far from being desirable.

The arrangement of apparatus used is shown in the diagram. The hot gas, as it issues from the hydraulic main, is led directly through a long coil into the saturator. The coil is surrounded by the acid, and hence the latter gets heated considerably. At the bottom of the coil is an overflow pipe leading into a seal pot, where the products of condensation are collected. A considerable amount of tar is here removed. From the saturator the gas passes to a box or washer containing hot tar, such as is now used on many gas works as a naphthalene preventive. The gas issues through a nozzle or spray into the hot tar, and is thor-

oughly broken up thereby. The small tar bubbles or vesicles in the gas are also broken up and the tar removed. The removal of this tar fog at some part of the process is essential, as otherwise it is deposited in the saturator during the neutralizing process, and the sulphate is consequently discolored.

In order, partially, to overcome the difficulty of sulphur deposition in the saturator, a second washer was introduced. This washer was charged first with a strong solution of ammonium sulphate and in later experiments ammoniacal liquor. Precisely what the chemistry of the process is we are not at present able to say, but it proved to be a fairly successful method of overcoming the sulphur trouble. Beyond this second washer was an exhauster and a small tar extractor. This latter was used as an extra precaution, to guard against deposition of tar in the saturator. The gas then passed into the saturator, which was charged with pyrites acid. The acid was obtained at 144° Twaddell, and made up in the saturator at strengths varying from 55° to 100° Twaddell.

The arrangement was found to work fairly satisfac-

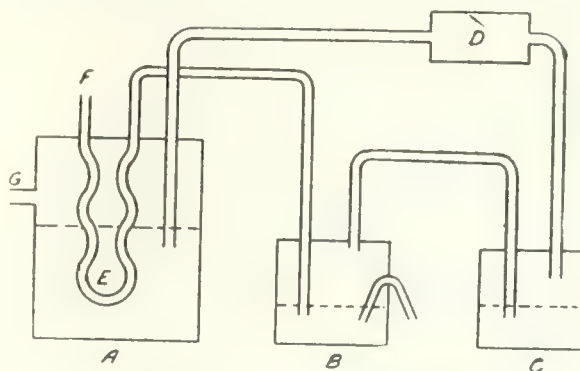


Diagram Showing Arrangement of Apparatus.

A, saturator; B, tar washer; C, liquor washer; D, exhauster and tar extractor; E, coil for heating acid; F, gas entrance from hydraulic main; G, gas exit to purifiers.

torily, and the heat given out by the coil in the saturator was sufficient to do away with any extraneous evaporation. The saturator had two compartments, one open and one closed and fitted with an acid seal, so as to enable the salt to be fished out without stopping the flow of gas at all. A good white salt was produced, which showed no trace of sulphur and which yielded on analysis 20.75 per cent of nitrogen.

Increased activity in mining in countries along the lower Asiatic coast is marked, not only in the Chinese Provinces of Yunnan and Kwangsi, but in Indo-China. The *Courier d'Haiphong* calls attention to the steady increase in mining generally in Tonkin and particularly notes that there was an output of over 449,000 metric tons of coal in 1909, more than half of which is exported, mainly to Hongkong. Over 14,000 metric tons of zinc were exported in the same year. Among other items in export returns are noted tin, copper, and wolfram. The introduction of modern machinery and mining appliances is not rapid, but fairly steady.

*From American Gas Light Journal.



THE VALUE OF CERTAIN PAINT OILS.*

By HENRY A. GARDENER.

The recent activity of the paint chemist in searching for oils to partly replace linseed oil, has come not only as a result of the present scarcity of the latter commodity, but through a desire to develop a vehicle that will be for certain technical paints even more satisfactory, if possible, than linseed oil.

It is not the writer's intention, however, to encourage the use of so-called substitutes for linseed oil. The market is flooded with them at the present time, and a great danger will lurk in their use until their composition is understood or their value determined in a practical way.

The real object of this paper is to present a summary of the results obtained by the writer from a series of experiments on paint vehicles, which may serve to guide other investigators along similar lines of research. The tests outlined herein, although of interest in pointing out the possibilities of the use of many oils other than linseed, are, nevertheless, tentative in nature and subject to confirmation through practical field exposure tests before they are to be accepted commercially.

It is well known that a most desirable feature of paint oils is their ability to set up in a short period to a hard surface that will not take dust, that will stand abrasion and offer resistance to moisture and gases.

This drying property is dependent upon the chemical nature of the oil. If it is an unsaturated compound, like linseed oil, rapid absorption of oxygen will cause the film to dry rapidly and become hard. If the oil be of a fully satisfied nature, like mineral oil, for example, oxygen cannot be taken up to any great extent and drying will not take place. The various animal and vegetable oils differ in their power of oxygen absorption to a lesser or greater extent. This difference is referred to by the chemist in terms of the iodine value. The iodine value of linseed oil is approximately 190, meaning that one gram of the oil will take up 190 centigrams of iodine.

Soya bean oil which has presented itself as a claimant for distinction in the list of paint oils, is now being imported into this country in large quantities. It is obtained from the seeds of *Soya Hispida*, a plant indigenous to Manchuria, but raised extensively in our own Southern states as a cattle food.

The writer has examined a number of representative samples of the oil that have come into this country, and the results obtained can be seen in Table I.

It is evident that the iodine value of soya bean oil is the only chemical characteristic that markedly differentiates it from linseed oil. Therefore, in the detection of soya bean oil and its estimation, the iodine values of several samples of mixed oils are given, as being of interest in this connection (Table II).

A series of tests were conducted to determine what drier was the most efficient to use with soya bean oil,

TABLE I.—CHEMICAL CHARACTERISTICS OF SOYA BEAN OIL.

Sample No.	Specific gravity.	Acid No.	Saponification No.	Iodine No.	Per cent of foots.
1	0.9233	1.87	188.4	127.8	3.81
2	0.9240	1.92	188.3	127.2	..
3	0.9231	1.90	187.8	131.7	..
4	0.9233	1.91	188.4	129.8	..
5	..	..	..	130.0	..
6	..	..	..	132.0	..
7	..	..	..	136.0	..
Average	0.9234	1.90	188.2	130.7	..

and the results of these tests indicated that the percentage of lead or manganese, or of lead and manganese, which gave the maximum efficiency in the drying of linseed oil, gave also to soya bean oil the best results. The methods used in carrying out the tests were the same as on linseed oil, referred to later in this paper.

A definite quantity of oil was placed in weighed friction-top can covers, which were re-weighed after

TABLE II.—IODINE VALUES OF LINSEED OIL AND MIXED OILS.

Sample No.	Straight linseed.	Soya, 25 per cent; linseed, 75 per cent.	Soya, 50 per cent; linseed, 50 per cent.	Soya, 75 per cent; linseed, 25 per cent.
1	190.3	175.2	160.7	140.4
2	189.5	175.9	161.7	140.8
3	188.0	175.4	160.3	139.0
Average	189.3	175.5	160.9	140.4

receiving the oil, to determine the amount used for the test in each case. The tin covers containing the oil were then placed in a large box under definite temperature control and humidity, for a certain period. Weighings were made at different periods, and the increases in weight, due to absorption of oxygen, was calculated to percentages.

Table III gives the results obtained. The time of drying was somewhat longer than with linseed oil.

Several test panels were painted with zinc and lead, alone and in combination, using linseed oil, soya bean oil, and mixtures of the two. No driers were used in these paints, as the object of the tests were to determine the drying action of the oils, without driers. The

*From the Journal of the Franklin Institute

drying of the linseed oil paints was very good in less than two days, while the straight soya bean oil required nearly four days to set up, and was then tacky for a long period. Laboratory tests of a preliminary nature made with mixtures of the two oils, ground with pigments, so far have indicated that a moderate percentage of soya bean oil is not detrimental when used with linseed oil.

The writer has found that a few drops of soya bean oil placed in a porcelain dish will give with a drop of strong sulphuric acid, faint fluorescent yellow and green colors, forming a pattern distinct from the dark-brown begonia-shaped leaf formed by sulphuric acid with pure linseed oil. Many other qualitative tests have been tried to detect the presence of soya oil in admixture with linseed oil. Soya oil seems, however, not to possess the chromogenetic properties of oils, such as cottonseed oil.

Tung oil, or Chinese wood oil, as it is often termed, has been considered heretofore as an oil most valuable in the manufacture of varnishes, but seldom if ever to be used as a paint oil. As a matter of fact, however, in manufacturing, certain technical paints for the protection of concrete and steel, the waterproofing of structures, and for other purposes, the manufacturer may find this oil, or its products, most useful, and to be prized very highly. Toch, as well as Lewkowitsch, have shown that metallic tungates are most excellent driers, and the former paint technologist has indicated their value for certain special purposes. The high iodine number of tung oil makes it a most rapid drier, even faster than linseed oil, but the soft and opaque film it leaves upon oxidation, makes essential its treatment by special processes when it is used either for varnishes or for paints, except in special cases.

The writer has prepared the tungate of lead by precipitating saponified tung oil with lead acetate. This tungate after it has been freed from water, may be fused with a small percentage of colophony and borate to produce a rapid drier that gives a perfectly clear film. Experiments with this drier are referred to later.

Menhaden oil, although of marine-animal origin, has a high iodine value, as the following recent tests will show:

Sample number 1.....	Iodine, 149
Sample number 2.....	Iodine, 162
Sample number 3.....	Iodine, 164
Sample number 4.....	Iodine, 158
Sample number 5.....	Iodine, 161

The oil obtained from fresh fish is extremely light in color, but the odor is objectionable, when the oil is heated. This oil is a fairly rapid drying oil, and its composition would indicate that it might possess considerable merit. Toch claims that it is of great value for smokestack paints, and for zinc paints exposed at the seashore. Its contents of stearine might account for certain waterproofing properties claimed by producers and would suggest its use in paints for the protection of iron and steel.

Some of the other oils used in these experiments will be considered briefly. Cottonseed oil and corn oil are both light in color, and although greatly inferior to even soya bean oil in their drying action, should merit further investigation.

Rosin oil, the product of the dry distillation of rosin, is dark and viscous. It has a low iodine value, and a high acid number that unfits it for general use as a painting oil.

Perilla oil is very high in iodine number, even higher than linseed oil, but it also is very scarce. It is obtained from the Perilla nuts, grown in Manchuria and the East Indies. Its iodine number is 206, but Lewkowitsch claims that its drying action is accompanied by a peculiar "dropping" of film. This condition might be overcome by special treatment. The writer has not made any experiments on this oil.

Waste oil, or the mixture of fatty acids, and glycerides obtained from the garbage in large cities, is being produced and used in great quantities as soap oils, but the investigations of the writer on several samples have shown this product to be of small value as a paint oil.

TABLE III.—SOYA BEAN OIL AND LEAD DRYER.

Per cent PbO.	0.05	0.10	0.30	0.50	0.70	1.00	1.30	1.60
1 day	..	0.07	0.63	1.34	1.05	1.53	0.93	1.35
3 days	..	0.07	3.52	4.31	2.75	4.86	4.82	4.12
5 days	..	0.09	5.04	6.06	6.09	6.75	6.75	6.66
12 days	..	..	6.88	7.54	7.43	7.76	7.32	6.47
15 days	..	..	8.84	8.93	8.59	8.81	8.44	7.46
20 days	0.05	0.20	9.02	9.08	8.90	9.03	8.65	7.83

TABLE III.—SOYA BEAN OIL AND MANGANESE.

Per cent MnO ₂	0.01	0.05	0.15	0.26	0.30
1 day	..	..	..	0.02	0.01
Per cent gain 10 days	..	5.06	6.48	6.10	5.97
20 days	0.05	9.07	8.80	6.78	6.51

TABLE III.—SOYA BEAN OIL, MANGANESE AND LEAD DRYER.

Per cent PbO.	0.20	0.30	0.50
MnO ₂	0.05	0.15	0.25
1 day	3.04	3.77	3.74
Per cent gain 8 days	5.96	6.43	6.47
12 days	6.33	6.78	6.67

Its use in combination with linseed oil to produce a soft, elastic, water repellant film for paints to protect iron and steel submerged in sea water should be studied.

There are several blown and oxidized linseed and other oils upon the market that are so thick that 50 per cent of benzine or other diluent is required to reduce them to painting consistency. There are also

TABLE IV.—CONSTANTS OF VARIOUS OILS.

	Specific gravity.	Iodine number.	Acid number.	Saponification number.
Linseed oil.....	0.934	188	3	191
China wood oil....	0.942	172	5	190
Soya bean oil.....	0.922	134	2	188
Menhaden oil.....	0.935	160	5	189
Sunflower oil.....	0.929	128	4	188
Cottonseed oil....	0.923	110	4	..
Corn oil.....	0.924	117	8	190
Rosin oil.....	0.989	28	..	..
Waste oil.....	..	60	34	..

several debloomed mineral oils which resemble linseed oil only in viscosity and their value should be looked into with great care before they are used.

There are on the market many so-called Japan oils. The base of many of these oils is melted rosin, thinned with some linseed oil and large quantities of benzene.

Their use is not to be recommended until practical tests of their value have been made.

A table (Table IV) of constants of various oils was made from recent experiments, the oils being subsequently used for the tests referred to in Table V.

The following drying tests (Table V) were made by painting thin films of oil on glass plates, exposing to the atmosphere, and carefully observing the time of the drying. These tests do not indicate the wearing value of the oils tested, but are simply criterions of their relative drying values. Although sample No. 2, which is untreated tung oil, dried with Drier X more rapidly than untreated linseed oil with Drier X, it is the writer's opinion that the linseed oil would be the more serviceable. The pine oil is not that of commerce, but a product of the refining of wood turpentine. "Drier X" referred to was strong lead and manganese linoleate drier.

TABLE V. DRYING TEST OF OILS

Sample No.		Hours
No. 1.	*Pine oil with 10 per cent Drier X.....	2½
No. 2.	Tung oil with 10 per cent Drier X.....	2¾
No. 3.	Linseed oil with 10 per cent Drier X.....	3
	{ Linseed oil...25 per cent } 4 { Soya bean oil...25 per cent } { Menhaden oil...25 per cent } { Tung oil...15 per cent } { Lead tungate...10 per cent }	3
No. 5.	{ Linseed oil...75 per cent } { Chinese w'd oil...25 per cent }	with 10% Drier X 3
No. 6.	{ Linseed oil...75 per cent } { *Pine oil...25 per cent }	with 10% Drier X 3
No. 7.	{ Linseed oil...50 per cent } { Menhaden oil...25 per cent } { Soya bean oil...15 per cent } { Lead tungate...10 per cent }	3¾
No. 8.	Menhaden oil with 10 per cent Drier X.....	3½
No. 9.	Soya bean oil with 10 per cent Drier X.....	3½
No. 10.	{ Linseed oil...80 per cent } { Waste oil...10 per cent } { Lead tungate...10 per cent }	with 10% Drier X 3½
No. 11.	{ Linseed oil...75 per cent } { Soya bean oil...25 per cent }	with 10% Drier X 4
No. 12.	{ Linseed oil...75 per cent } { Corn oil...25 per cent }	with 10% Drier X 5½
No. 13.	{ Linseed oil...75 per cent } { Cottonseed oil...25 per cent }	with 10% Drier X 6½
No. 14.	{ Linseed oil...75 per cent } { Rosin oil...25 per cent }	with 10% Drier X 10
No. 15.	Corn oil with 10 per cent Drier X.....	8
No. 16.	Cottonseed oil with 10 per cent Drier X.....	9
No. 17.	Rosin oil with 10 per cent Drier X.....	{ Not dry } { in 2 days }

*Pine oil referred to is not rosin oil, but the high boiling point distillate of wood turpentine.

The only practical field tests made by the writer thus far on the value of oils, other than linseed oil, are those at Nashville, Tenn., where a fence has been erected, consisting of 42 tests of white paints. In these tests pure linseed oil was used, except in formulas 31 to 42.

On these tests there have been placed one formula which has given satisfaction on the various other

test fences erected, and this formula has been made up with combinations of linseed oil and other oils.

One of the oils tested is called pine oil, a high-boiling-point product obtained from the manufacture of wood turpentine from sawdust. This has a boiling point of over 210° C. as against 150°, the boiling point of ordinary gum spirits. It is almost water white and has the same penetrating qualities as the pure gum spirits, and is almost free from that objectionable odor characteristic of so many low-boiling-point products. When mixed with 50 per cent linseed oil this product forms a paint oil of extremely light color, and most excellent properties, producing a semi-flat paint of great whiteness. If exposure tests prove this oil as worthy as the tests thus far show, it would appear to be of great importance to the Southern manufacturers to produce this material in quantity, by fractional distillation of the crude spirits. Its iodine number is high and its oxygen carrying properties are good. Its evaporation leaves a durable film.

The tests referred to on the Southern Test Fence are as follows:

COMPOSITION OF FORMULAS
Per Cent.

No. 31.	{ Corroded white lead...45 } { Zinc oxide...45 } { Asbestine...5 } { Calcium carbonate...5 }	{ Ground in pure linseed oil.
100		
No. 38.	Same as No. 31, but ground in 50 per cent raw linseed oil, 50 per cent soya bean oil.	
No. 39.	Same as No. 31, but ground in 50 per cent raw linseed oil, 50 per cent corn oil.	
No. 40.	Same as No. 31, but ground in 50 per cent raw linseed oil, 50 per cent cottonseed oil.	
No. 41.	Same as No. 31, but ground in 50 per cent raw linseed oil, 50 per cent rosin oil.	
No. 42.	Same as No. 31, but ground in 50 per cent raw linseed oil, 50 per cent pine oil.	

Driers.—The problem of drying of oils and their behavior with various siccatives in varying quantity is an interesting one, and, obviously, of considerable importance from a practical standpoint. Unfortunately there is a decided scarcity of reliable literature dealing with the subject for the guidance of those concerned in the manufacture or application of siccative products. Furthermore, when the problem is investigated, it is not difficult to see why this is so.

At a glance, it is evident that a decided obstacle in experimentation on the drying properties of oils, is the difficulty in obtaining identical conditions for comparative purposes. Inasmuch as a multitude of factors, such as uniformity and homogeneity of the driers and the oils themselves, intensity and source of light, temperature, uniformity of application, and many others, play a decisive part in the siccative tendencies of oils, the resources and ingenuity of the chemist engaged in the research are severely taxed.

It is a well-known fact that linseed oil, when applied to a clean surface, such as a glass plate, will undergo oxidation and take up oxygen to the extent of about 16 per cent, forming a hard, elastic, non-sticky product which has been called linoxyn. This material,

unlike the oil from which it has been formed, is insoluble in most solvents. Other oils, such as cottonseed, hemp, rape, olive, etc., are more fully satisfied in nature and have not the power to absorb the amount of oxygen taken up by linseed oil. It is said that perilla oil, however, will absorb as high as 20 per cent of oxygen.

In carrying out the following tests, on the drying of oils, a quantity of pure linseed oil of the following analysis was secured:

Specific gravity at 15 degrees C.	0.934
Acid number	5
Saponification number	191.2
Iodine number	188

This oil was distributed into a number of 8-oz. oil sample bottles, and to a series of these bottles was added varying quantities of a very concentrated drier made by boiling oil to 400° C. in an open kettle, with the subsequent addition of lead oxide. The amount of drier added to each bottle varied according to the percentage desired; being calculated on the lead content of the drier, which was very accurately determined by analysis.

There was secured in this manner a series of oils containing varying amounts of lead oxide, and from this lot was selected a certain number of samples which would be representative and typical of paint vehicles now found in the market.

Another series of tests were made by combining with a large number of samples of pure linseed oil as used above, various percentages of a manganese drier made by boiling oil at 400 degrees and incorporating therewith manganese dioxide.

Still another series of tests were made upon a number of oils into which were incorporated various small quantities of lead oxide and manganese oxide together, using the standard driers made in the above manner, all of which were carefully analyzed to determine their contents.

In view of the errors in manipulation that could occur where so many tests were made, it was not deemed advisable, in carrying out the tests, to use glass plates, on which only a minute quantity of oil could be maintained. A much better solution of the difficulty presented itself in using a series of small, round, crimped-edge tin plates about three inches in diameter, such as are used for lids of friction-top cans.

With paints it is impossible to secure films as thin as those presented by layers of oil on glass, nor would it be desirable to secure films of this same relative thickness. For this reason, an endeavor was made to conduct the following tests with films of the same relative thickness as that possessed by the average coating of paint. The drying of the films did not take place in the same short period, nor in the same ratio as with the thin layer that is secured by flowing oil upon glass. The results, however, are more practical, and of greater value to the manufacturer.

The cans were carefully numbered in consecutive order, corresponding to the numbers on the various

samples of oil. A very small quantity of oil was placed in each of the can covers, which were previously weighed, and allowed to distribute itself over the bottom surface thereof. Re-weighing of the covers gave the amount of oil which was taken for each test. The test samples in the covers were all placed in a large box with glass sides, having a series of perforated shelves. In the side of this box is an opening through which a tube was passed, carrying a continual current of air washed and dried in sulphuric acid.

Oxidation of the oil films commenced at once, and the amount of oxygen absorbed was determined at suit-

TABLE VI.—LINSEED OIL AND MnO_2 (MANGANESE) DRIER—TEST NO. 1.

Per cent MnO_2	0.02	0.05	0.15	0.25	0.35	0.45	0.55	0.70	1.00
1 day	0.08	0.11	0.16	...	3.21	3.46	3.27	3.01	2.76
2 days	0.16	5.88	4.48	...	3.63	4.01	2.70	3.51	3.18
3 days	0.21	6.79	4.61	...	3.83	4.31	...	3.91	3.18
4 days	...	...	4.64	...	...	...	...	...	...
5 days	3.01	6.84	...	...	4.13	4.68	4.19	3.91	3.99
6 days	8.00	...	4.88	...	4.37	...	4.51	4.32	4.13
7 days	8.58	6.92	4.90	...	4.48	...	4.61	4.52	4.23
Per c't gain 8 days	9.06	...	5.03	...	4.55	5.23	4.77	4.62	4.44
9 days	...	...	5.12	...	4.63	5.40	4.04	4.70	4.51
10 days	9.07	6.89	5.18	...	4.81	5.47	...	4.98	4.73
11 days	9.15	7.03	...	...	...	...	...	...	...
12 days	...	...	...	...	...	...	...	...	...
13 days	9.22	7.17	...	...	5.25	6.00	5.60	5.42	5.33
14 days	9.25	7.18	5.55	...	...	...	...	...	...
20 days	...	7.21	5.81	...	5.84	6.70	5.94	5.84	5.77

Above percentages of drier reckoned to metallic content.

able periods, by weighing, the increase in weight giving this factor. This test was kept up for a period of 20 days.

A test was also made in the same manner with a current of damp air passing into the box, to observe the relative oxidation under such conditions.

The experiments consumed considerable time and effort. A chart of the results obtained have been made (Table VI), to show the effect of the various driers.

The following outline will present to the mind of the reader the most salient points which have been

TABLE VII.—LINSEED OIL AND MnO_2 (MANGANESE) DRIER—TEST NO. 2 (CHECK).

Per cent MnO_2	0.02	0.05	0.15	0.25	0.35	0.45	0.55	0.70	1.00
1 day	...	3.12	4.42	3.86	...	3.19	2.98	3.27	2.56
2 days	...	6.15	4.73	...	...	3.51	3.28	3.70	2.96
3 days	0.28	8.29	...	4.12	3.72	...	3.39	3.71	3.15
4 days	3.83	6.32	4.75	4.21	3.87	3.61	3.58	4.05	3.43
Per c't gain 5 days	6.64	...	4.84	4.23	3.94	3.73	3.65	4.21	3.56
6 days	8.61	...	4.87	...	4.08	3.81	3.78	4.35	3.73
7 days	29.07	6.35	5.00	4.41	4.18	3.91	3.85	4.54	3.87
9 days	9.25	6.39	5.16	...	4.44	4.11	4.21	4.63	4.26
11 days	...	...	...	...	4.63	4.59	4.36	4.31	5.07
16 days	...	6.43	5.30	4.91	4.83	4.72	4.71	5.40	4.87

gleaned from these experiments, and which should give the manufacturer definite knowledge as to the best percentage of oxides to use either in boiled oil, paints or varnishes.

In the case of lead oxide, an increase in the percentage of lead oxide in the oil causes a relative increase in the oxygen absorption, but when a very large percentage of lead has been added, the film of oil dries to a leathery skin.

In the case of manganese oxide, the increase in oxygen absorption on the first day is much more pronounced than is the case with lead oxides. Further-

more, the oxidation of manganese oils seems to be relative to the increase in manganese up to a certain period, when the reverse of this law seems to take place, and beyond a certain definite percentage of manganese, added percentages seem to be of no value. It was furthermore observed that the films dry to a more brittle and harder skin than is the case with lead oxide.

The oxygen absorption with oils high in manganese has been noticed to be excessive, and the film of oil becomes surface-coated, drying beneath in a very slow manner, a condition that often leads to checking. The critical percentage where the amount of manganese appears to be most propitious and renders the greatest efficiency seems to be 0.02 per cent. This

box in which the samples of oil were drying, were discolored and turned brown after several days, showing that the acid had taken up some material of a volatile nature that was a product of the oxidation.

Another curious feature of these tests was the development of a peculiar aromatic odor which was given off by the oils upon their drying in dry air. When the oils were dried in moist air, a rank odor resembling propionic acid was observed, and this led the observer to believe that a reaction was effected by the absorbed oxygen, that caused the glycerin combined with the linoleic acid as linoleon to split up into evil-smelling compounds.

It has been suggested that the oxygen first attacks the glycerin, transforming it into carbonic acid, water

Per cent PbO.	0.00	0.05	0.10	0.30	0.50	0.70	1.00	1.30	1.60	1.90	1.60
1 day	0.042	0.049	0.092	0.058	0.066	0.062	0.062	0.079	0.039	0.14	0.72
2 days	0.098	0.104	0.153	0.116	0.158	0.194	0.194	4.83	4.79	5.27	6.11
3 days	0.128	0.159	0.170	0.137	0.279	0.185	7.11	8.60	5.35	7.89	8.28
4 days	0.164	0.214	0.206	0.178	...	4.07	7.39	9.55	8.53	7.93	8.68
5 days	0.176	...	0.306	...	0.340	7.60	7.47	9.87	8.78	8.18	...
6 days	0.188	0.231	...	0.243	0.472	9.36	7.64	10.01	9.00	8.24	9.09
7 days	0.206	0.251	...	0.253	1.080	10.06	...	10.14	...	...	...
8 days	0.212	0.253	...	0.280	4.80	10.38	7.70	10.22	9.05	...	...
9 days	0.226	0.291	0.306	0.331	7.36	10.41	7.73	10.23	9.07	...	...
13 days	0.327	0.428	0.610	0.674	11.01	10.67	7.91	10.48	9.29	8.62	...
15 days	0.466	0.455	0.650	2.41	11.05	...	7.92	10.50	9.30	...	...
20 days	0.521	1.08	1.78	8.76	11.25	10.67	7.98	10.52	9.36	...	...

critical percentage, as it may be termed, should not be exceeded, and any added amount of manganese has the effect of making the film much more brittle and causes the so-called "burning up" of the paint. The loading of paint with drier and the bad result therefrom, may be explained to some extent from the above results.

In the same way with lead driers, excessive amounts of lead oxide seems to have no beneficial effect on the drying of an oil, and when the percentage which seems to be the most beneficial, namely 0.5 per cent lead oxide is exceeded, the film is apt to become brittle.

TABLE IX.—LINSEED OIL AND PBO (LEAD AND MNO₂) (MANGANESE) COMBINATION DRYER.

Per cent PbO.	0.1	0.3	0.5	0.7	0.9	1.1	1.4
Per cent MnO ₂ .	.005	.015	0.025	0.35	0.45	0.55	0.7
1 day	0.026	0.061	0.055	0.022	0.16	0.11	3.06
2 days	0.094	0.087	0.143	0.16	5.21	6.28	3.37
3 days	0.118	...	0.17	4.23	7.63	8.31	3.74
4 days	...	0.11	0.23	7.36	8.87	9.20	4.02
5 days	0.120	0.12	0.29	9.04	9.13	9.37	4.17
6 days	0.17	0.13	1.44	9.88	9.26	9.51	4.34
7 days	0.21	0.18	4.65	10.11	9.28	...	4.45
11 days	0.30	0.26	10.03	10.35	9.61	9.85	5.11
12 days	...	...	...	10.45	9.66	...	...
13 days	0.35	0.54	10.37	10.51	9.67	10.03	5.33
18 days	0.49	3.43	10.38	10.62	9.62	...	5.73

Oils containing lead oxide driers are less influenced in their drying tendencies by conditions of moisture in the atmosphere than oils containing manganese, but frequently, however, the former dry much better in a dry atmosphere. As a general rule, varnishes rich in manganese dry more quickly in a dry atmosphere, while those containing small quantities dry more quickly in a damp atmosphere.

It was furthermore noticed in these tests that sulphuric acid, placed in dishes on the bottom of the large

and other volatile compounds, which are eliminated before the oil is dried to linoxyn. Toch, however, has shown that the drying of linseed oil gives off only very small percentages of carbon dioxide. Mulder has observed that in the process of linseed oil being oxidized, glycerin is set free, which becomes oxidized to formic, acetic and other acids, while the acid radicals are set free and are converted by oxygen into the anhydrides from which they pass by further oxidation into linoxyn.

The theory of auto-oxidation of linseed oil has been very ably treated by Blackler, whose experiments indicated that during the drying process the slow absorption of oxygen was, at a critical period, followed by a rapid absorption which he attributes to the presence of peroxides which accelerate the formation of more peroxide. Lead and manganese oxides may also be oxidized to peroxides by the action of oxygen, and in this event might act as very active catalyzing agents or carriers of oxygen. Blackler's statement that the presence of driers do not increase, but have a tendency to decrease the initial velocity of oxygen absorption, has been confirmed by these experiments, but it has been noticed throughout the tests that the driers have an accelerative action at a later period.

Some most interesting results were secured by dipping extremely fine copper gauze into linseed oil, and then suspending the gauze in the air. The adhesion of the oil to the copper caused the formation of films between the network, and remarkable drying action was observed. The copper or any superficial coating of copper oxide which may have been present on the metal, undoubtedly affected the result to some extent.

It has been found that metallic lead is even more efficient than copper in this respect, but this may be due to the action of free acid in the linseed oil, forming lead linoleates, products that greatly accelerate drying. Another interesting experiment was made by taking pieces of gauze cloth and immersing in linseed oil. After the excess oil had been removed, by pressing, the cloth was again weighed to determine the amount of oil used for the experiment. The increase in oxygen absorption in this case was very rapid and the result obtained confirmed the results in the other experiments.

In order to secure a more evenly distributed state of the oil, tests were conducted by saturating pieces of stiff blotting papers and after exposure, weighing as usual.

The influence of light on the drying of oils is unquestionably a potent one. The practical painter knows that a certain varnish will dry quicker when exposed to the light than when in the dark.

Chevreul was one of the first pioneers in this field of research to observe the effects of colored lights on drying, and he claimed that oil exposed under white glass dried more rapidly than when exposed under red glass which eliminates all light of short wave lengths.

Genthe obtained interesting results in the drying of oil submitted to the effect of the mercury lamp. Oxidation without driers was effected probably through the formation of peroxides.

In commenting on this subject Blackler gives a description of the use of the Uveol Lamp, which is similar to the mercury lamp, but has instead of a glass casing which cuts off the valuable rays, a fused quartz casing which allows their passage.

In the boiling of linseed oil, by certain processes the oil is heated to 250° and manganese resinate incorporated therein. It goes into solution quite rapidly. In other processes the oil is heated to 400° or over, and manganese as an oxide is boiled into the oil. Although it is unsafe to say that a small percentage of rosin such as would be introduced by the use of resinate driers, is not harmful, yet it appears that this process should give a good oil, inasmuch as it has been found that no matter whether the manganese is added to the oil, as resinate, borate or oxide, practically the same drying effect is noticed in every case where the percentage of manganese is the same. It is the opinion of some, however, that the resinate driers are not as well suited for durability as oxide driers. However, if a boiled oil is found to contain on analysis a small percentage of rosin (less than 0.5 per cent) it should not be suspected of adulteration. Practical tests, however, should be made with such oil along with an oil made with an oxide drier, before pronouncing on their relative values. Inasmuch as the addition of certain driers to linseed oil lessens the durability of the film, it is more practical to use the smallest amount of drier that will serve the purpose desired, that is, set

the oil up in a paint to a hard condition which will not take dust and which will stand abrasion.

The results of this investigation would indicate that when lead or manganese linoleates are used, the most efficient results may be obtained with 0.5 per cent lead or with 0.05 per cent manganese or with a combination of 0.5 per cent lead and 0.02 per cent manganese.

Until more definite results have been obtained with the tungates, which will probably prove of exceptional interest as driers, the above driers will probably be used to the greatest extent.

Carborundum is manufactured by fusing a mixture of pure granulated coke, very pure glass sand, and sawdust. The coke is the carbonaceous residue from the distillation of petroleum; the sand used is the purest glass sand. The sawdust is added mainly to make the mixture porous and thus to avoid explosions of the carbon monoxide produced during the reaction. The fundamental reaction takes place between the sand (silica) and the coke (carbon), resulting in the production of carbide of silicon, or carborundum. During a run which lasts 36 hours each furnace consumer 2,000 h. p. The voltage, starting at about 250, is lowered as the resistance decreases until it comes down to a voltage of about 185. The carborundum crystals are crushed under manganese steel rollers in a circular pan of the same material. The crushed product is then treated in a bath of sulphuric acid to dissolve the minute particles of steel that have been cut from the rolls and the pan. This method of removing the steel has been found more satisfactory than the method of removing it by magnets. After the carborundum has been washed to free it from acid it is screened into different grades and is then ready for manufacture.

Soapy water, according to the Electrical Review of England, will instantly wet coal dust and soak its way into it up to the full extent. Moreover, when the water has dried out the dust does not again become dust, for the soap remains disseminated throughout the mass and the dust has become a more or less cloggy mass, and is as effectively laid as is the dust of a country road by tar or oil. Experiments have been made which are said to show that the treatment of a dusty mine will be both effective and cheap, and there is not likely to be any offensive result from the use of soap, for if necessary the water can be made a vehicle for some form of antiseptic which will go far to improve the general sanitation of a mine.

Profit sharing in the chemical works of an important firm in Hoechst, Germany, is said by Consul General Frank D. Hill, of Frankfort-on-Main, to take the form of a yearly "gratification" which varies with the dividend paid by the company, the amount received by each employe being dependent upon the nature of his services.

M E T A L L U R G Y.

EXPERIENCES WITH THE 15-TON HER- OULT FURNACE AT THE SOUTH WORKS OF THE ILLINOIS STEEL CO., SOUTH CHICAGO.*

The 15-ton Heroult arc furnace at South Chicago has now been in more or less continuous operation since May 7, 1908, a period of over a year and a half, and, therefore, a few experiences of those who have been interested in these operations will perhaps be of interest.

DESCRIPTION OF FURNACE.

Upon a solid foundation about 5 ft. above the ground level, a stationary rack 8 ft. 9 ins. long is fastened. Upon this rack the furnace proper rests on a floating pinion, fastened to its shell by rivets. The arc of this floating pinion has a radius of 10 ft. and aims to give the furnace an angle of approximately 29° when tilted over to its full extent. The position of the furnace when tilted is shown in Fig. 1.

Attached to the extreme back of the furnace is an 18-in. plunger with a 4-ft. stroke working in a cylinder attached to a hydraulic line of 500 lbs. pressure to the square inch. This gives a lifting power, approximately, of 45 tons. The balance of the furnace is so arranged that the equilibrium is never upset, and therefore to return to a horizontal position merely requires the releasing of the pressure and the furnace returns of its own weight. It will be readily seen that the floating pinion and rack work requires some provision for the forward motion of the furnace when tipping. This is taken care of by having a movable cylinder pivoted at both the top and bottom which allows the cylinder to follow the motion of the furnace.

THE FURNACE PROPER.

The furnace shell is of plate steel 1 in. in thickness, riveted together. The outside horizontal cross-section plan is, approximately, that of a complete circle of $13\frac{1}{2}$ ft. in diameter, with two flattened portions situated at the front and back respectively.

On the bottom of the furnace, within the 1-in. plate, and next to it, one row of magnesite brick laid the $4\frac{1}{2}$ -in. way is placed across the flat portion.

The side walls of the furnace are vertical and consist of two rows of magnesite brick laid the 9-in. way, giving a thickness of 18 ins. of magnesite brick. These solid magnesite brick walls extend up to the furnace roof.

The bottom "proper" of the furnace consists of dead-burned Spaeter magnesite to a depth of 12 ins. at its thinnest point, which is, of course, at the extreme center. From this thinnest point the bottom slopes gradually upwards so as to form a portion of a sphere 7 ft. 2 ins. in radius.



Fig. 1—View Showing 15-Ton Heroult Electric Furnace Tilted by 18-in. Plunger.

To diverge a moment from the construction of the furnace, this bottom was put in in the following manner: Dead-burned and carefully ground Spaeter magnesite was mixed with basic open-hearth slag in the proportion of four of magnesite to one of open-hearth slag. To this mixture sufficient tar was added to make the mass sufficiently plastic to enable it to be tamped into the furnace in the usual manner. The entire depth of the bottom was tamped in this way. Next the furnace was filled with wood, dried out for about 48 hours and then filled with coke and the electrodes lowered and the current turned on. In this way the bottom was fluxed into place.

The furnace roof is of silica brick 12 ins. in thickness. The roof is made up on a movable ring. This ring is fitted with a top and bottom angle iron to take a skew back brick, and then from this the arch is spanned across the 10-ft. interior of the furnace with an 8-in. rise. The bricks are set in circles parallel to the steel ring. The usual wooden wedges being placed

*From an illustrated lecture by C. G. Osborne, Superintendent of Special Steels, Illinois Steel Co., delivered before the Chicago Section of the American Electrochemical Society on Jan. 20, 1911.

here and there to take care of the subsequent expansion of the brick. Holes for the electrodes are left in the roof by means of templates placed in position. The bricks are held in position around these holes by their lateral pressure.

There are five doors, two on each side of the furnace, and one in front over the pouring spout. These doors are of cast-iron lined with clay brick laid the $4\frac{1}{2}$ -in. way. They work in the usual groove arrangement and are operated by steam pressure of about 150 lbs. The front door over the pouring spout is an exception to this, being operated by hand with a counterbalance.

THE ELECTRODES AND THEIR HANDLING DEVICE.

The three electrodes are let down through the roof in the form of an equilateral triangle, each side of which is 5 ft. 2 ins. in length. The apex of this triangle pointing directly east, that is, towards the back of the furnace. The center of this triangle coincides with the center of the furnace roof. The overhead structure which supports these electrodes may be seen in Fig. 2.

There are, of course, three separate holders, one for each electrode. Each holder is constructed of a

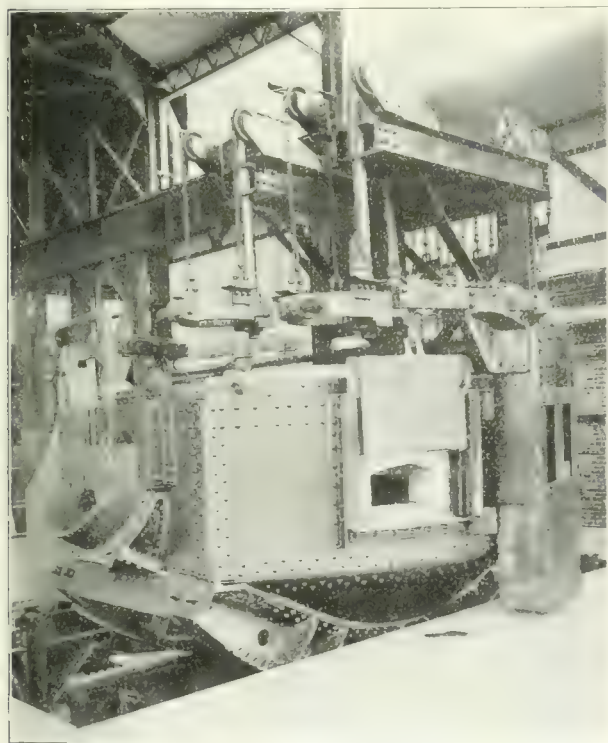


Fig. 2—Side View of 15-Ton Heroult Electric Furnace.

solid copper casting bolted directly to the bus-bar. In front these holders are split and joined with a right and left screw which enables the holder to be opened or closed at will. The holders are designed to carry a 24-in. electrode, but by means of contact blocks to fill, any smaller sized electrode can be employed.

The weight of the electrode is supported by the chains. These chains run back over pulleys to the drums at the back of the furnace. These electrodes are kept in alignment by vertical guides. The elec-

trodes are regulated by individual motors placed at the back of the furnace. These motors are attached by gears to the drums. The regulation is by hand, by controllers, or by an automatic device. Each electrode is, of course, regulated separately (see Fig. 3), and its regulation is as follows:

A stationary magnet coil, the current through which can be adjusted at will, is opposed by a floating magnet coil whose position is determined by a spring and the amount of current passing through it. The current in this floating magnet is obtained from a current transformer attached to the bus-bars. When the current in the bus-bars and the proportional current, therefore, of the floating magnet, is that for which the regulator of the stationary magnet is set, the magnetic opposition of the two magnet coils is exactly counterbalanced by the strength of the spring, and the apparatus is at rest. When, on the other hand, the strength of the current in the bus-bars, and, consequently, the proportional current in the floating magnet, is either greater or less than that at which the stationary magnet is set, then the magnetic opposition of the two magnet coils either overcome or is overcome by the strength of the spring and the two magnet coils either retreat from or approach towards one another. This motion of the floating coil is transmitted by a lever which in turn causes a dog to drop into a slot in a constantly moving rocker wheel, and in this way a right or left contact is made, and the motors controlling the electrodes, which, of course, work on D. C. current, are started, either clockwise or anti-clockwise, and the electrodes in turn are either raised or lowered. This is repeated with each oscillation of the rocker until equilibrium is restored. It might be of interest to state here that each contact made as spoken of above serves to raise or lower an electrode about $\frac{1}{8}$ in. It has already been stated that the regulation can be by hand-controllers. These three hand regulators are situated at the back of the furnace at a distance of about 4 ft. It is, of course, understood that the use of the hand regulators simultaneously disconnects the automatic regulators.

The operating platform of the furnace is about 9 ft. from the ground level. Around the furnace on this platform, at convenient points, bins are placed for the miscellaneous materials used in furnace operation.

The front part of the furnace platform opens up to allow a ladle to be hung in position when the furnace is tapped and for any miscellaneous work in the pit.

The power for the furnace is generated by dynamos having as prime movers reciprocating gas engines, reciprocating steam engines, high pressure and low pressure turbines. It is three-phase in character, 2,200 volts, and 25-cycle.

At the electric furnace it is stepped down by means of three 750-kw. transformers to the voltage of the furnace. These transformers are so arranged with switches that the primary turns may be altered to give a secondary voltage of 80, 90, 100 or 110 volts as desired. Ordinarily 90 volts is used.

The entire building is spanned by a 50-ton crane. The pouring platform is situated to the south of the furnace. It is 30 ft. long and enables eight molds to be placed in position for pouring.

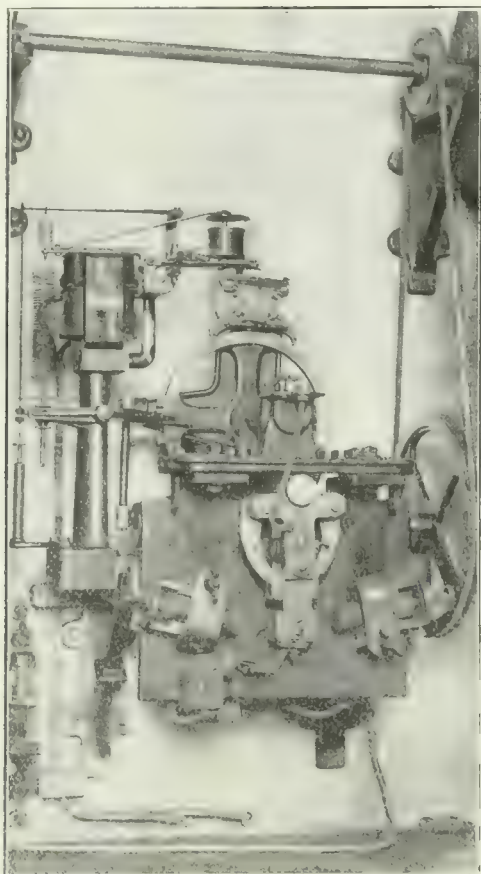


Fig. 3—Mechanism for Automatic Regulator of Electrodes.

THE OPERATION OF THE FURNACE.

The normal operation of the furnace is as follows:

Ordinary Bessemer pig iron is full blown in a 15-ton Bessemer converter, in from eight to twelve minutes. It is then poured directly from the Bessemer vessel to an electric furnace transfer ladle, and drawn by locomotive to the electric furnace building, a distance of about one-quarter of a mile, in about five minutes. As a precaution against the possible formation of a skull in the ladle, the Bessemer charge is blown about 1,500 lbs. of scrap hotter than ordinary Bessemer practice.

Immediately the ladle is received at the electric furnace it is picked up by the crane, slightly tilted and the silicious slag is completely cleaned off by hand-rabbling. The metal is now ready for charging. (See Fig. 4.) To do this the ladle is merely turned over on its trunnions and the metal poured into a spout through which it rushes to the furnace. This operation of cleaning off the slag and charging occupies from 5 to 10 minutes.

The electrodes are lowered and the current turned on. As the metal is being poured into the furnace the helpers shovel iron oxide and lime into the furnace through the working doors. In this way a basic ox-

dizing slag is produced which serves to remove the phosphorus. In about 30 minutes this slag has served its purpose and the furnace is tilted slightly forward and the slag removed in from 5 to 10 minutes by hand-rabbling.

The recarburizer is added at this point. On the bare surface of the oxidized metal, lime is quickly added with sufficient fluorspar to keep the mass fluid. In about 15 minutes this lime is melted, and finely divided coke dust is now thrown onto the top of the slag beneath each of the three electrodes. Under the influence of the arc, calcium carbide is produced in gradually increasing quantities.

As soon as this state of affairs is reached a neutral if not actually reducing atmosphere has been obtained. From here to the finish there is practically a dead-melt in a reducing atmosphere. The slag at this stage of the process is fluid and highly basic. If a sample should be taken and water added to it the resultant acetylene gas, from the well-known calcium carbide water reaction, is of sufficient quantity to light and burn for half a minute.

Tests are now taken to show the condition of the steel. A small cylindrical test piece is poured and



Fig. 4—Charging Converter Metal into Furnace.

forged to a round pancake shaped object under a steam hammer located at the furnace. If this forged sample shows by its appearance a satisfactory condition of the metal, the bath is tapped. If not, further refining is necessary.

To tap the furnace the electrodes are raised from the bath and the ladle swung by a crane under the pouring spout and the tilting lever pulled forward.

The pouring is done through a 1½-in. nozzle to molds of varying sizes.

A typical furnace charge sheet is shown in Table I. It will be seen from this that it takes us an hour and a half to two hours to heat, according to the grade of steel produced.

What is actually done in the electric furnace at

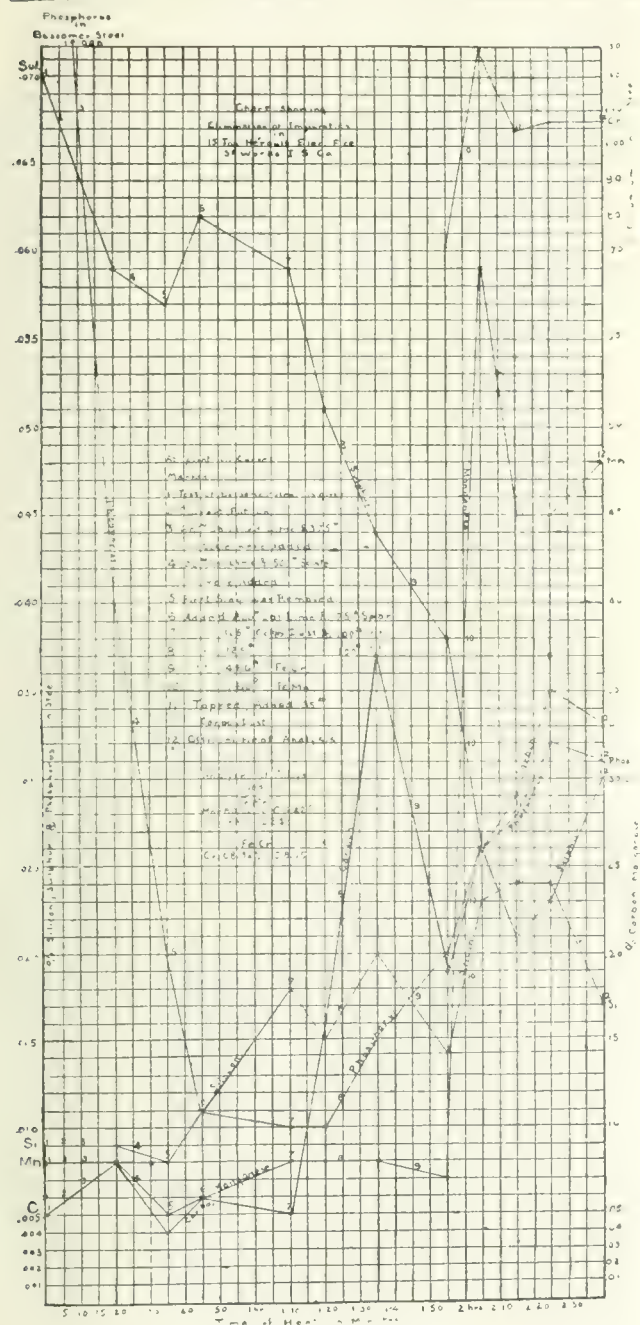


Fig. 5—Diagram Showing Elimination of Impurities in Electric Furnace.

South Works is to take oxidized blown metal of an approximate analysis of:

C.	S.	P.	Mn.	Si.
.05—.10	.035—.070	.095	.05—.10	.005—.015

and produce deoxidized steel low in sulphur and phosphorus and within reasonable limits of practically any analysis required by the consumer.

Fig. 5 shows the elimination of the impurities in the electric furnace during the process of making an alloy heat and Fig. 6 shows the fluctuation in the slag analysis.

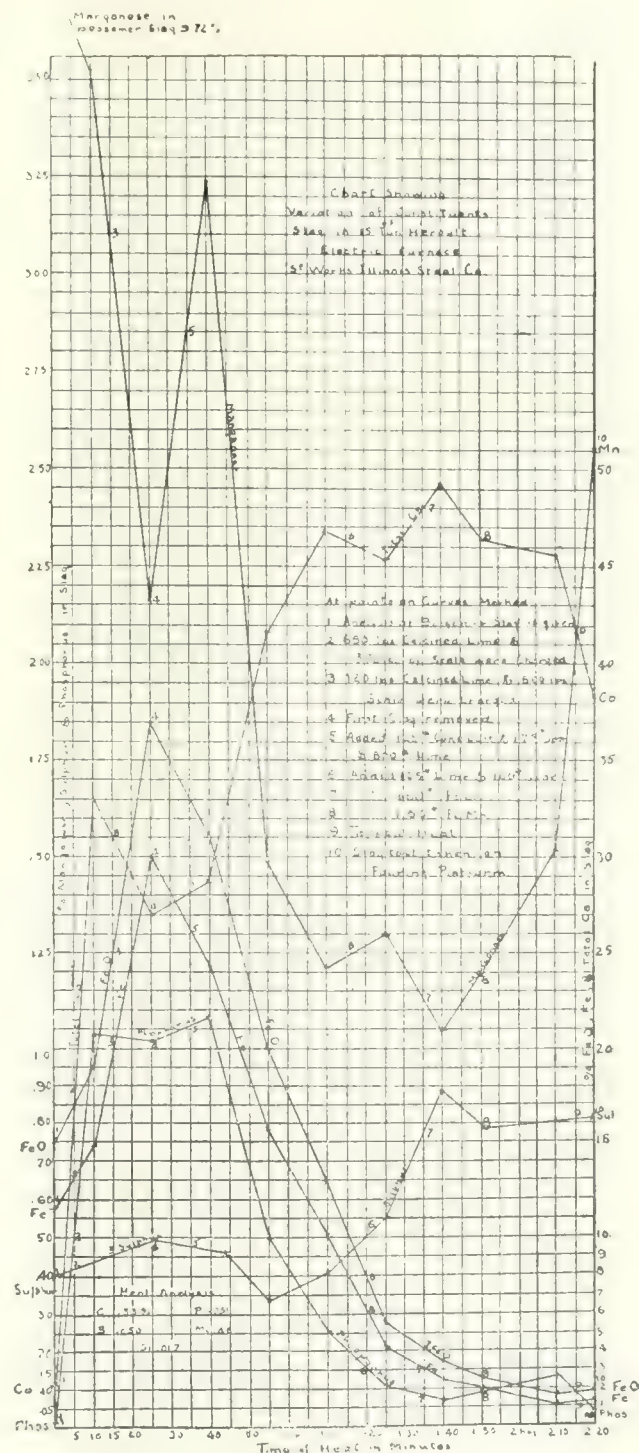


Fig. 6—Diagram Showing Fluctuation in Slag Analysis.

THE VARIETY OF PRODUCT.

The electric furnace at South Chicago has operated on a greater variety of product than any furnace in the world, and with bewildering intermingling of high-grade alloy steels, high-grade carbon steels and ordinary carbon steel. We have made of ordinary carbon steel, rails of a dozen different sections, billets of all

sizes and grades, plates of all sizes and grades, structural shapes, castings, small and large, high carbon and low, and forgings of all sizes. Of alloy steels we

TABLE I.—ELECTRIC FURNACE CHARGE SHEET.

Material.	Pounds.
Bessemer blown metal.....	30,000
Scale	700
Ferromanganese, 80 per cent.....	200
Ferrosilicon, 10 per cent.....	60
Ferrosilicon, 50 per cent.....	80
Recarbonizer	130
Fluorspar	400
Coke dust.....	200
Lime—first slag.....	600
Lime—second slag.....	600
Dolomite	400
Magnesite	25

Tapped previous heat.....	7:00 a. m.
Metal ordered for.....	7:15 a. m.
Metal received.....	7:15 a. m.
Began fettling.....	7:17 a. m.
Current on.....	7:27 a. m.
Slag off—began.....	8:00 a. m.
Slag off—finished.....	8:11 a. m.
Tapped	8:48 a. m.
Time of heat.....	1 hr. 21 min.

have made, nickel, nickel-chrome, chrome, manganese steel, and silicon steels.

TABLE 2.—AVERAGES OF ULTIMATE STRENGTH AND PER CENT ELONGATION (IN 2 IN.) OF ELECTRIC AND OPEN HEARTH PLATE STEELS. ARRANGED ACCORDING TO CARSON ANALYSIS.

Electric Plate Steel.				
Carbon. Per cent.	No. of heats.	No. of tests.	Average ultimate strength.	Average per cent elongation.
0.08	3	5	59,194	27.25
0.09	2	5	59,502	27.25
0.10	3	5	57,662	29.20
0.11	3	5	59,585	27.65
0.12	2	5	64,080	26.05
0.13	3	18	65,880	25.90
0.14	3	12	62,030	26.43
0.15	3	15	61,203	26.70
0.16	5	27	69,220	25.25
0.17	5	22	66,638	25.03
0.18	5	21	65,330	25.44
0.19	3	15	70,336	23.83
0.20	2	7	72,853	22.82
0.21	2	9	68,640	25.16
0.22	1	2	70,820	25.75
0.23	2	7	73,370	23.53
0.24	1	2	69,540	23.12
0.16	2.8	10.7	65,640	25.67
Basic Open Hearth Plate Steel.				
Carbon. Per cent.	No. of heats.	No. of tests.	Average ultimate strength.	Average per cent elongation.
0.08	3	5	51,690	32.00
0.09	2	5	52,570	32.00
0.10	3	5	54,666	29.90
0.11	3	5	54,854	30.65
0.12	2	5	56,510	29.70
0.13	3	18	56,595	28.54
0.14	3	12	55,322	28.64
0.15	3	15	58,683	28.21
0.16	5	27	52,901	28.61
0.17	5	22	55,780	28.81
0.18	5	21	58,469	28.31
0.19	3	15	59,504	27.50
0.20	2	7	58,294	28.82
0.21	2	9	63,050	27.00
0.22	1	2	54,500	29.75
0.23	2	7	61,148	27.39
0.24	1	2	63,560	26.25
0.16	2.8	10.7	56,947	28.94

Average of averages.

The above table shows 15.5 per cent increased ultimate strength for the electric steel and 11.3 per cent decreased elongation as compared with open hearth plate steel of approximately the same chemical analysis.

Perhaps many of you will wonder at the adaptability of this furnace for handling cold materials. We have made a number of heats from cold materials. In several of these we took the worst scrap we could pick up—this appears to us to be stove plate scrap. With this stock and axle heat was made which showed excellent physical tests. If I remember correctly, 77 blows from a 1,640-lb. hammer falling 45 ft. broke the first axle, and 68 similar blows the second.

THE QUALITY OF THE STEEL.

The main chemical or physical differences in electric steel seem to be these:

A comparative freedom from oxidation.

A comparative freedom from segregation.

A higher tensile strength and slightly higher ductility for the same chemical analysis up to about 40 carbon where the difference becomes less apparent. (See Table II.)

A steel of greater density than our other commercial steels with the possible exception of crucible.

In conclusion let me say that the three electric furnaces of the Steel Corporation are the largest in the world that have as yet been put in operation. They have presented many interesting and complex problems and the manufacturer has encountered occasional troubles. Although we feel that we are as yet but on the threshold of this method of manufacture, we feel that electric steel is a success and its future assured.

A recent U. S. Consular Report states that quinine stands first on the list of drugs for the Tropics; Epsom salts next; then comes calomel, castor oil, tincture of iron, or liquor of perchloride or iron, opium, and brandy. A large majority of the drugs most used are those well known to the doctors of two generations ago. One medical missionary has worked out the quantities of 12 drugs that would be necessary for the treatment of 20,000 patients in a year. He estimates that the approximate cost would be \$325. His list includes 30 lbs. of quinine, 300 lbs. of Epsom salts, and 200 gross of a tonic pill.

In 1910 there were received at the Patent Office in Washington 63,293 applications for mechanical patents, 1,155 for design patents, 181 for reissues of patents, 6,843 for registration of trade-marks, 755 for registration of labels, and 226 for registration of prints. There were 35,807 patents issued, including designs; 123 patents reissued; and 4,239 trade-marks, 370 labels, and 120 prints registered. The number of patents that expired was 22,768. The number of allowed applications, awaiting the payment of final fees, was 11,336; and the number forfeited for non-payment of the final fees, 7,442. The total receipts were \$2,025,537; the expenditures, \$2,005,712, and the surplus of receipts over expenditures, \$19,825. The total balance to the credit of the Patent Office in the Treasury of the United States on December 31, 1910. was \$6,998,228.

THE SCHUMACHER AND RONAY PROCESSES OF IRON ORE AND FLUE DUST BRIQUETTING.*

Pig iron producers have given much attention in the past decade to the problem of briquetting dust ore. The difficulties attending the use of fine ores are naturally greater with the increase in dimensions of blast furnaces. These increased dimensions and proportionately increased blast, as is well known, have added greatly to the accumulation of flue dust. This is valuable, not only owing to its high percentage of iron, but also because of the coke and lime contained in it. Many processes for solving the problem of briquetting fine ores and flue dust have been tried.

The German blast furnaces have made a practical success of briquetting fine ores, owing to the fact that

the Schumacher and the Ronay. The former was particularly referred to in the paper of C. de Schwartz, Liege, Belgium, on "The Briquetting of Iron Ores," read at the Buxton, England, meeting of the Iron and Steel Institute. Installations of the Schumacher system have been in operation in Germany for some time at Königshütte in Upper Silesia; Rombacher Hüttenwerke, Rombach; Eisenhütten Aktienverein Düdelingen, Düdelingen, Luxemburg; Dortmunder Union in Dortmund; and Hasper Eisen & Stahlwerke at Haspe, Westphalia. There is also an installation at the Cockrill Works in Belgium. The Ronay system is installed and is working satisfactorily at the following plants: Oberschlesische Eisenbahnbedarfs Aktiengesellschaft, Friedenshütte, Upper Silesia; Lothinger Hüttenverein Aumetz-Friede in Kneuttingen, Lorraine, and Gutehoffnungshütte in Oberhausen. It is being used,

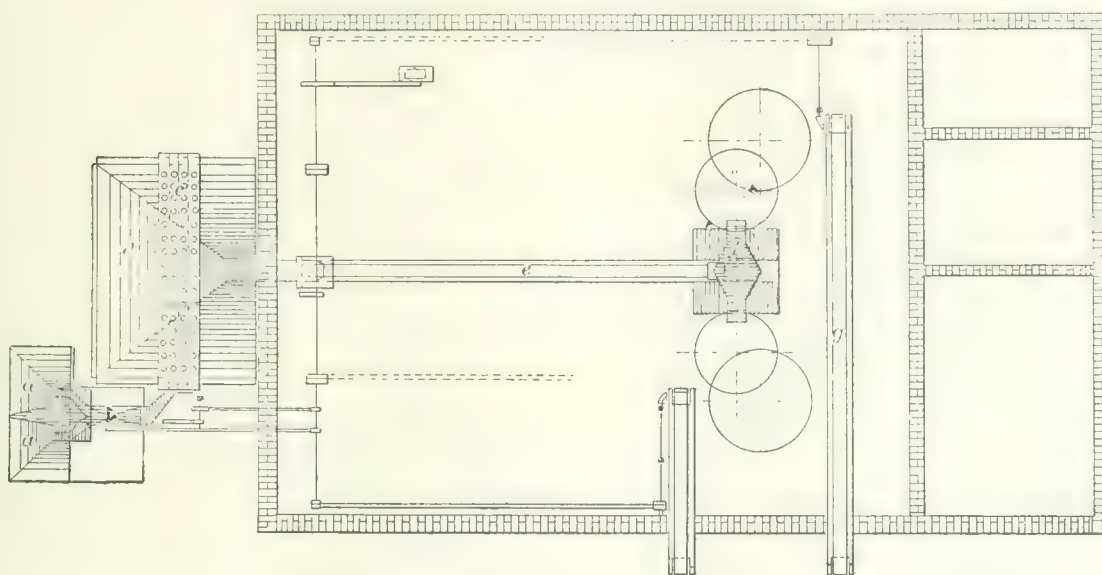


Fig. 1—Plan of Ronay Plant for Briquetting Dust Ores and Fine Dust.

they have been compelled to utilize waste products to a greater extent than American pig iron producers. Of the 1,500,000 tons of fine dust, which, according to Prof. G. Franke, of the Mining Academy of Berlin, constitutes the annual waste product of blast furnaces, 250,000 tons is briquetted and used as raw material. These briquettes contain an average of 35 per cent of iron. Their value differs according to the value of rock ore delivered at the various blast furnaces. It has been clearly shown, however, that the briquetting of the fine dust is of considerable economical value in connection with pig iron production. The use of flue dust is of far greater importance in this country, since it is estimated that over 2,500,000 tons, containing a high percentage of iron, is added each year to the accumulated dust piles at blast furnaces.

Briquetting installations have been made at many of the German blast furnaces. Some of these are of local origin and their success is dependent on local conditions. Two processes which are of established efficiency have already become well known, namely,

too, at a large number of works for the briquetting of iron and other metal turnings and borings. Both systems are operated at low cost. According to German conditions, this would amount to 25 to 30 cents per gross ton.

Figures 1 and 2 show the general arrangement of a Ronay briquetting installation. The dust ore or flue dust arrives at the storage bin *a* and is lifted by elevator *b*, and after passing through a screen *c*, in order to eliminate all the larger pieces of ore, coke or stone, it drops into the bin *d*. The screened material is transported by belt conveyor *e* to the small hopper *f*, from which it passes into the press. The finished briquettes are taken from the press by belt conveyor *g* and loaded into small cars, which run to the blast furnace.

For the handling of the installation there is needed a machinist, a man at each press and two men at each belt conveyor to discharge the briquettes. The capacity of each press is, according to the weight of the dust ore or flue dust, from five to eight tons per hour. The power necessary for the press and all auxiliary apparatus for the Schumacher process is 60 hp. For

*From The Iron Age.

the Ronay process, it is somewhat more. The cost of the waste chemical product required with the Schumacher process—it is used rather for its catalytic action than as a binder—is put at $2\frac{1}{2}$ to 5 cents per ton.

The difference in the two processes may be referred to: The Schumacher process is solely intended for the briquetting of flue dust and requires from 0.5 to 1 per cent of the above mentioned chemical. The briquette is formed by comparatively low pressure and can be immediately loaded on cars ready for use in the blast furnace. In the Ronay system no added material is required, and the dust ore or flue dust is simply subjected to a specially applied pressure, which also gives the briquette such mechanical strength that it can be immediately loaded on blast furnace cars. In both cases, the dust ore passes directly from the receiving bin to the press without any previous modification. Their simplicity makes these two processes espe-

The Westinghouse Air Brake Co. has decided to install at Wilmerding, Pa., a Ronay briquetting plant for the briquetting of metal turnings and borings, the order having been placed through the American representative, G. Leve, 30 Church street, New York. In Germany and elsewhere the two processes referred to have been introduced for the briquetting of ores by the Allgemeine Brikettierungs Gesellschaft. A subsidiary company, in which the important locomotive firm of A. Borsig is interested, handles the Ronay process of briquetting borings and turnings in all countries except the United States. It is known as the Hochdruck Brikettierungs Gesellschaft.

A report of the U. S. Geological Survey on "The Gypsum Industry in 1909" states that the following machinery is necessary to manufacture 100 tons of ordinary wall plaster a day: (1) 1 crusher, esti-

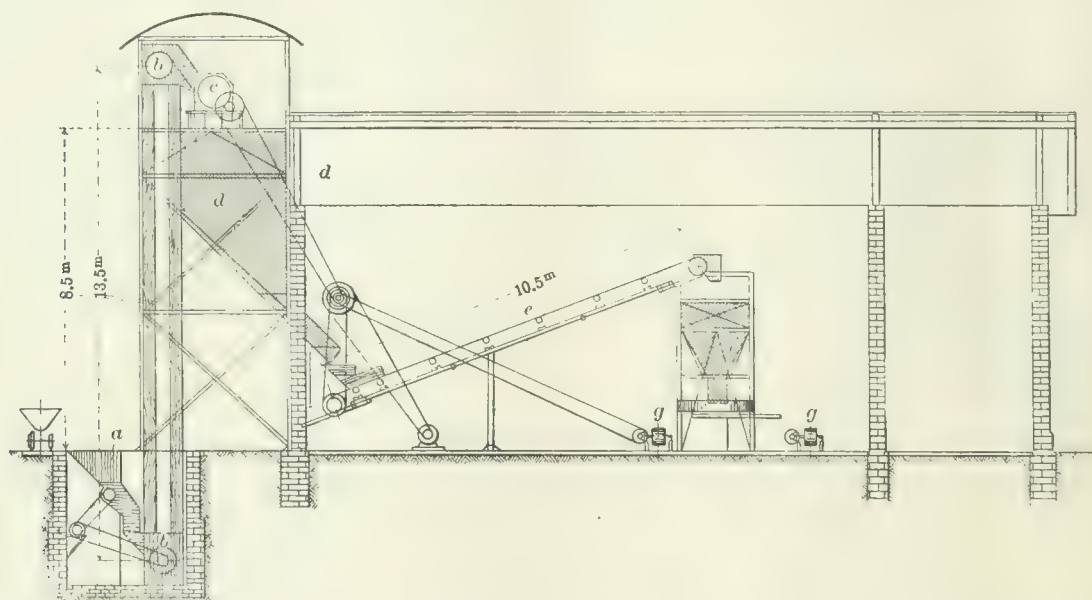


Fig. 2—Sectional Elevation of Ronay Plant for Briquetting Dust Ores and Fine Dust.

cially advantageous, inasmuch as the economical utilization of waste products demands as a first consideration a minimum manipulation. No binding material has to be used in either process.

The mechanical strength of these briquettes, it is claimed, is such as to permit of their use in the largest blast furnaces. It is sufficient to allow a considerable proportion of coke dust to be mixed with the flue dust, thereby adding to the quality of the waste product. The value of these processes is indicated by the fact that even at point where ore is taken from the ground as cheap as 40 cents per ton, as in the Minette district, briquetting plants of the types referred to are now in operation.

The low operating cost of the Schumacher and Ronay systems is responsible for their use in Germany rather than the cintering method. One disadvantage of the latter is that the high temperatures cause some smelting of the material and thus hinder the reducing process in the blast furnace.

estimated cost \$1,000; (2) 1 direct-heat drier, 48 inches in diameter and 27 feet long, and 1 dust room, estimated cost \$2,500; (3) 1 pot or bowl crusher for grinding the material after drying, estimated cost \$300; (4) 4 French burr-stones for fine grinding, cost about \$300 each; (5) 2 calcining kettles, \$200 each. In addition to this machinery, machines for mixing plasters besides the necessary elevators, conveyors, shafting, belting, and bins are required. On account of the nature of the process the elevators and conveyors should be made of steel, the bins of concrete or other noncombustible material, and the entire plant should be as nearly fireproof as possible.

Experiments in growing rice in Manchuria are said to show that rice can be grown there more profitably than in China proper or Japan and more profitably than either corn or soya beans. The season is short, but long enough to produce high-grade rice.

USE OF OIL IN ZINC SMELTERS AT JOPLIN, MO.*

According to reports from the zinc districts of Missouri and Kansas, the recent scare sustained by the zinc industry by the seeming weakening of the surrounding gas fields and the fear that gas could no longer be depended on for factory fuel has been dissipated, and a radical readjustment is under way.

The use of oil as cheap and reliable fuel in smelters and cement plants of Missouri and Kansas has not only been demonstrated as feasible, but extremely practicable, a notable example of the success of the new fuel being the absolutely satisfactory operation of the Monarch cement plant at Humboldt and the Edgar smelters at Cherryvale with oil burners.

Oil-burning apparatus is being installed in the cement plants at Neodesha and other points in Kansas. This is hailed as making a new era in the zinc smelting and cement manufacturing industries of Missouri and adjoining states.

Because of lacking the gas supply demanded in enormous quantities by these big factories a total readjustment of the zinc smelting industry was well under way on account of the failure of the gas supply. It had even gone so far as to be used as a market factor in manipulating the price of spelter.

Now comes the announcement that oil burning in the gas smelters has been successfully solved, and this places the entire smelting capacity of Kansas back into the producing class as soon as the necessary oil pipe lines can be constructed and the burners installed.

It has been known a long time that the Edgar Zinc Company, at Cherryvale, Kan., was experimenting with the use of oil. It was even known that as a fuel for roasting it was perfectly feasible, but it was hardly supposed that oil could be used in firing the retorts. In fact, the experiments of the New Jersey Zinc Company in Texas and Oklahoma in trying to use oil as a fuel for a zinc furnace were taken as conclusive evidence that it was too expensive.

Now comes the announcement that the Edgar Zinc Company has started four blocks of its furnaces of this kind of fuel in addition to using it for roasting. A 6-in. pipe line is being built into Cherryvale from Sycamore to supply the smelter with this kind of fuel. Simultaneous with this announcement comes the second one that the Neodesha plant of the Granby company is installing the same kind of burners and will also burn fuel oil in its furnaces instead of abandoning the plant when the gas gives out entirely.

Both the Edgar and the Granby companies are preserving as much secrecy about their new discoveries as they can. It is understood the experiments in the burning of oil occupied much of last year at the Edgar plant. A German process proved to be the most satisfactory, and is the type installed. The Granby plant will employ a similar process. The Granby plant is just now being equipped with these burners.

WASTE OF ZINC BLENDE.

A recent issue of the St. Louis Globe-Democrat contained the following dispatch from Joplin, Mo., regarding the action of air and deleterious water on the tailings at the concentrating plants in the Joplin district:

Zinc blende worth hundreds of thousands of dollars, discarded in the tailing or waste gravel from concentrating plants for the Joplin district, is lost beyond recovery through the action of air and deleterious waters, according to Frank Kelley, chemist of the Illinois Zinc Company, who has completed an assay of the tailing heap of the old Morning Hour mine west of this city.

Turned out at a time when milling practice was much more inefficient than present day methods, the waste gravel from the Morning Hour mine, like the waste from hundreds of other early day mines, was rich in zinc blende. It is a common practice in this district to construct "tailing" mills near heaps of mine gravel, regrind the material, remill it, and thus recover a large percentage of the ore. Tailing heaps, as a rule, are richer when the waste comes from a mill improperly equipped for thoroughly treating the mine run dirt. Virtually all of the early day mills were inadequate in equipment, and as a result the tailing piles would run high in mineral.

Some tailing piles will contain as high as 5 per cent ore, and as actual mining can be conducted at a profit in the thin sheet ground mines where the ore percentage is sometimes less than 3 per cent of the dirt mined, it is easy to see how gravel, containing 5 per cent ore, already on the surface, ready for the milling, could be retreated at an enormous profit. And yet, in the case of the Morning Hour operations, the Mays Milling Co., which is doing this work, has discovered to its dismay that the one time rich tailings are now almost totally devoid of ore particles, and the operators of the plant declare that it is impossible to make a profit from handling the gravel.

The report of Mr. Kelley enlightened them as to the cause of their failure; the presence of sulphuric acid in the ground water had resulted in the mountain of flint tailings becoming saturated with a devouring element, which worked even more venomously when the air, in circulating through the porous heap, came in contact with the acid.

It is claimed that an English chemist has succeeded in producing a dealcoholized beer free from all intoxicating properties, but in no way different in taste or flavor from ordinary beer. The alleged new process is stated to be the result of many years' experimenting, and is described as follows: "It was found that by slightly warming the beer and then driving through it a brisk current of carbonic acid gas the alcohol could be extracted in the form of minute bubbles. By continuing the process every trace of alcohol could be absolutely eliminated without destroying any of its former qualities as a beverage." It is further claimed that every hogshead of beer treated by this process yields 3 gallons of spirit, driven out by the carbonic acid gas, which among other purposes, if produced in sufficient quantities, could be placed upon the market at a price which would enable it to compete favorably with petrol. The product is reported to be superior to petrol for driving motor cars, and emits no fumes or odors.

*From the Industrial World.

HARDENING GRAY IRON CASTINGS.

The Vulcan Furnace Co., of Warren, O., has perfected a process for hardening gray iron castings, which is described as follows by a member of the company in a letter published in a recent issue of the *Industrial World*:

The depth of the hardened surface in our process for treating gray iron castings is controllable to quite a considerable degree. Pieces can be case-hardened 1/32 or 3/64; or the piece can be hardened to a depth of 1/4 in., metal to depth of 3/16 in. being of practically the same density.

We take castings that are to be machined and finish them complete, subsequently hardening; and after hardening, if extreme accuracy of fit is required they can be ground; but for most commercial purposes grinding is not necessary as there is no appreciable change in form, excepting for a coefficient of expansion which we allow for in the original dimensions of finish.

For the sake of demonstrating the hardness of the iron, we have made lathe tool 1/4-inch. by 5/16 in., hardened and supported in an Armstrong tool holder, as we are in the habit of doing with high speed steel, and used this tool for reducing steel shafting. It seems to stand the same speeds as employed with novo superior high speed tool steel, without burning, and makes an equally true cut.

It is not, of course, contended that cast iron is a suitable material for this work, as the strength is not sufficiently great; but this method has been taken to demonstrate the hardness of material after being treated by our process.

The hardening treatment is not confined to small pieces as larger work can be hardened with equal satisfaction, with, however, a higher cost per piece but a lower cost on pound basis than small pieces.

Our researches do not permit us to say definitely whether or not the brittleness of hardened small pieces can be reduced by draw tempering, although our experiments indicate that this can be done to a considerable degree.

Eliminating this possibility, however, it is apparent that there is a very large field in which glass hard castings will be extremely advantageous. For bearings, bushings and surfaces subject to erosion, which are substantially supported, we can employ this metal even though very brittle and not suited to conditions where not properly supported or subject to repeated shocks tending to crystallize or fracture.

We are of the belief that in brake shoes especially the characteristics of this metal offer peculiar advantage. This, however, is based wholly upon an analysis of the situation as we have not had an opportunity as yet to obtain conclusive data resulting from service tests.

The testing of structural materials "both belonging to and for the use of the United States," formerly conducted by the Geological Survey, has been transferred to the Bureau of Standards, the equipment for the work being also transferred. The Standards Bureau has now, in addition to its main laboratories in Washington, a laboratory at Atlantic City for the study of the effects of sea water and sea air on structural materials; another at Northampton, Pa., for the testing of cement for the Isthmian Canal Commission, and a third at Pittsburg for testing a variety of materials. These facilities of the bureau are available to all departments of the Government.

The amount of creosote oil imported into the United States in 1910 was 42,008,386 gals.

THE SIX-DAY WEEK.

We are indebted to the *Industrial World* for the following abstract of a paper read recently by R. H. Wilkinson, Jr., on "Industrial Efficiency":

"Any system which works ill to one or many to the apparent disadvantage of few or many is working its own destruction. In short, the Golden Rule is an economic rule.

"The man who is debilitated either through sickness or inadequate sustenance, or is exhausted by long hours or the seven-day week, has lost in a large measure his power to resist temptation, easily falls into excesses, is vulnerable morally and physically, cannot be efficient. I assert that the six-day week established in any iron and steel community will not only make possible industrial efficiency, but will do more to uplift the men affected than all our moral and religious agencies under present conditions.

"Intelligent selfishness impels the men who operate our industries to work out their industrial salvation with the well-being of the employes at heart. Indeed the huge corporation will continue to flourish only as it discovers the line of least resistance—the law of righteousness applied to its various activities."

Mr. Wilkinson then took up the question of the work-day, the work-week and compensation. He advocated the three-shift system for the blast furnace as the system which would make a man produce, day after day, year after year, a maximum quantity of product of desired quality. He also advocated a six-day week for blast furnaces, and stated that an effort was being made to work out a six-day week plan at the Struthers furnace. He declared that the six-day week has immeasurably raised the efficiency of the men of his force whom it has affected. Mr. Wilkinson continued:

"We are this month trying out the six-day week among the furnacemen, paying full wages for two of the four rest days. We have for this purpose, where possible, grouped the force in squads of seven, carrying for each seven men an extra man, who becomes a part of the regular force, thus permitting a man off each day. Under this plan we are in a number of positions eliminating the long turn by having the extra men work the change turn. The advantage to the organization in always having an extra man trained for every position, and the larger force upon which to rely on in case of emergency is obvious."

It is interesting to note in this connection that several of the subsidiary corporations of the U. S. Steel Company are now trying to work out a plan for a six-day week which will apply to their furnacemen and the men in the rolling mills. Instruction has been given that all work not absolutely necessary shall be shut down from Saturday to Monday.

The chemical industries are also guilty of a seven-day week and should attempt to solve this problem.

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NOTES AND COMMENTS.

Purchase of Material on Specification.

There is perhaps no single item upon which a large industrial corporation wastes more money than in the purchase of material, either in a raw state for its own manufactures or in a finished state for its own use, than in the purchase of material under a trade name and not by purchasing the same material in an open market under a properly drawn specification. A number of cases have come to the attention of the writer lately where firms were paying from three to ten times the normal price of the article simply because they were purchasing it under some fancy name and not on the merits of the article.

The preparation of a proper specification is a matter which requires the closest attention. A specification need not be drawn so that only the best material on the market of this kind will meet the specification. It should only be drawn with such stringency that the material supplied under the specification will meet the

needs of the purchaser and on this point is needed the judgment of a man who is perfectly familiar not only with the use to which the material is to be put but also with the grades of material which are obtainable on the open market. This will mean that the specification must be drawn by a man who has not only the proper technical training but who also has had the practical experience which will enable him to select the proper material. It is useless to draw a specification, submit this to the manufacturers who expect to supply the articles in question and expect the specification to enforce itself. The specification will amount to nothing if proper samples were not taken from each consignment and examined to see that they conform to the specification. In the purchase of many materials it will be possible, perhaps, to draw the chemical specification alone which will cover the case. In many instances a physical specification alone will meet the needs. There are few, if any, cases where a practical specification including both physical and chemical properties will not enable a consumer to purchase satisfactory articles on the open market at a much lower price than they can be obtained if purchased only under a certain trade name or on the reputation of certain dealers. Any consumer of large quantities of material will be wise to submit his needs to some such man as has been indicated, who will draw a proper specification, and he can then rely on his routine man in the laboratory to enforce it. The trouble many times is that the manufacturer expects his chemist, who probably is a recent graduate, to draw the proper specification and then leaves the enforcement of the specification to the purchasing agent who has no direct interest in the question other than in obtaining the material sought at a low price.

Congress of Technology at Boston, Mass.

What promises to be a very remarkable and striking record of the place of science in modern industry will be presented in the series of papers which will constitute one of the main features of the Congress of Technology to be held in Boston on April 10-11, of this year. The first of these dates is the fiftieth anniversary of the chartering of the Massachusetts Institute of Technology, and the primary purpose of the Congress is fittingly to mark that anniversary.

But the interest of the celebration goes far beyond any personal associations with the Institute of Technology. While "Tech," as its friends know it, has been a leader in the application of science in the industries, the progress of that institution and of its graduates marks the enormous development that has taken place in all industries the world over. A large number of Technology graduates who have been conspicuously successful in varied lines of engineering will present papers at the Congress, dealing with various aspects of the country's manifold industrial problems and treating of those problems not only as they exist

now but as they promise to take different shape in the future. The whole body of papers will therefore constitute a survey of engineering and industrial science as a whole, from a body of men who speak from first-hand experience with industrial problems all over the country.

The papers, separately, will discuss the conditions and prospects in specific industries and will therefore be of exceptional interest to the great number of men practically engaged in them. No similar discussion of the industries has been attempted on such a scale, and the record promises to be of unique suggestive value to the manufacturers of the country. The meetings will be open to the public.

RECENT PATENTS.

The following patents relating to industrial and engineering chemistry are reported by C. L. Parker, solicitor of chemical patents, McGill building, Washington, C. D.:

978,853. (Method of and Apparatus for Making Gas.) William T. Cutter, East Lyme, Conn., assignor by mesne assignments to Hydrocarbon Converter Co., New York, N. Y., a corporation of Delaware. Patented Dec. 20, 1910.

978,866. (Method of Eliminating the Active Toxic Agent from Coffee.) Heinz Evers, Zurich, Switzerland, assignor to Gerster & Cie., Clarens-Montreux, Switzerland. Patented Dec. 20, 1910.

978,878. (Manufacture of Artificial Silk.) Giuseppe Guadagni, Pavia, Italy. Patented Dec. 20, 1910.

978,905. (Method of Making Steel.) Horace W. Lash, Cleveland, Ohio. Patented Dec. 20, 1910.

978,934. (Process for Continuous Electrolysis of Aqueous Solutions.) Albert Pietzsch, Triberg, Germany, and Ewald Steinbuch, Monthey, Switzerland, assignor to Society of Chemical Industry in Basel, Switzerland. Patented Dec. 20, 1910.

978,970. (Process of Producing Gas.) Geo. J. Webber, Kansas City, Mo., assignor to Louisa M. Webber, Kansas City, Mo. Patented Dec. 20, 1910.

979,113. (Heat Treatment of Steel.) Samuel S. Wales, Munhall, Pa., assignor to Carnegie Steel Co., Pittsburgh, Pa., a corporation of New Jersey. Patented Dec. 20, 1910.

979,136. (Paint or Varnish Remover.) Carleton Ellis, Larchmont, N. Y., assignor to Chadeloid Chemical Co., New York, N. Y., a corporation of West Virginia. Patented Dec. 20, 1910.

The remover comprises approximately methyl acetate 45 gallons, monochlor methyl acetate 15 gallons, benzol 30 gallons, benzene 20 gallons, turpentine 5 gallons, carbonate of magnesia 100 pounds, and wax 15 pounds. Patented Dec. 20, 1910.

979,337. (Manufacture of Steel.) Alleyne Reynolds, London, England. Patented Dec. 20, 1910.

979,394. (Process for the Production of Alloys of Tin and Titanium.) Augustie J. Rossi, Niagara Falls, N. Y., assignor to the Titanium Alloy Mfg. Co., New York, N. Y., a corporation of Maine. Patented Dec. 20, 1910.

979,431. (Nitrocellulose Gunpowder.) Conrad Herbert Heinrich Claessen, Berlin, Germany, assignor to E. I du Pont de Nemours Powder Co., Wilmington, Del., a corporation of New Jersey. Patented Dec. 27, 1910.

979,465. (Method of Treating Carbon Electrodes.) Walter Gaston, New York, N. Y. Patented Dec. 27, 1910.

979,476. (Explosive.) Chas. E. Griffin, Salt Lake City, Utah. Patented Dec. 27, 1910.

979,541. (Metallurgical Process.) Joseph Charles Morris, Findlay, Ohio. Patented Dec. 27, 1910.

979,542. (Transfer Ink and Process for Producing the

Same.) Chas. F. Mores, New York, N. Y. Patented Dec. 27, 1910.

979,901. (Process for Treating Wood.) Frank J. Taylor, Phoenix, Ariz. Patented Dec. 27, 1910.

979,902. (Recovery of Rubber.) Harold T. G. Vander Linde, New York, N. Y.

The method consists in increasing the bouyance of the rubber particles, for the purpose described, by subjecting it to the action of a specific gravity lowering intumescent. Patented Dec. 27, 1910.

980,364. (Process of Separating Mercury From Poor Ores by Electrolysis.) Bela Szilard, Paris, France, assignor to said Bela Szilard and Henry Bergerat, Paris, France.

The process comprises treating finely divided ore with a lixivating agent comprising a mixed solution of sodium chlorid and calcium sulfid, and then subjecting the solution to electrolytic action. Patented Jan. 3, 1911.

980,369. (Manufacture of Steel.) Wm. E. Walker, New York. Patented Jan. 3, 1911.

980,641. (Method of Treating Zinc Ores for the Recovery of Metallic Zinc Therefrom.) Harry H. Hughes, Springfield, Mo., assignor of one-half to F. X. Heer and A. J. Eisenmayer, Springfield, Mo. Patented Jan. 3, 1911.

980,802. (Manufacture of Incandescent Gas Mantles.) Ignaz Kreidle, Vienna, Austria Hungary. Patented Jan. 3, 1911.

981,008. (Process of Scouring Wood.) Peter Schmid, Basel, Switzerland. Patented Jan. 10, 1911.

981,042. (Process of Manufacturing Fiber Pulp.) Conrad L. Weiberg, Detroit, Mich., assignor of one-half to Albert Sesselbacher, Detroit, Mich. Patented Jan. 10, 1911.

981,178. (Substance to be Used as Varnishes and for Impregnation and Insulation and Method for Their Production.) Gottlieb Gottfried Diesser, Zurich-Wollishofen, Switzerland. Patented Jan. 10, 1911.

981,225. (Bituminous Paving Cement.) Clifford Richardson, New York, N. Y., assignor to The Barber Asphalt Paving Co., Philadelphia, Pa., a corporation of West Virginia. Patented Jan. 10, 1911.

981,280. (Method of Reducing Iron Ore.) John T. Jones, Iron Mountain, Mich. Patented Jan. 10, 1911.

981,474. (Carbonizing Steel.) Hugh Rodman, Pittsburg, Pa., assignor to Rodman Chemical Co., a corporation of Pennsylvania. Patented Jan. 10, 1911. The method consists in making an electrode and subjecting it to the action of electrolysis in a carbon containing bath.

981,575. (Unmagnetizable Steel.) Frederick Kohlhaas, Dusseldorf, Germany. The steel contains about 9.8-10.3% manganese, about 0.9-1% carbon, about 0.2-14% titanium, maximum 0.8% silicon, maximum 0.03% sulfur and maximum 0.015% phosphorus. Patented Jan. 10, 1911.

981,647. (Process for the Manufacture of Resin Soap.) Frank Stuart Havens, Hartford, Conn., assignor to Harrison Bros. & Co., incorporated, a corporation of Pennsylvania. Patented Jan. 17, 1911.

981,775. (Process of Detinning Tin Scraps.) Carl Von Der Linde, St. Tonis, near Crefeld, Germany. Patented Jan. 17, 1911.

981,850. (Smelting Process and Apparatus Therefor.) Richard Fleming, New Rochelle, N. Y. Patented Jan. 17, 1911.

981,880. (Process for Roasting Sulfid Ores.) Chas. W. Renwic, Isabella, Tenn. Patented Jan. 17, 1911.

981,900. (Process for Productions of Compounds of Persulfuric Acid.) Gustav Teichner, Nuremburg, Germany, assignor to The Firm of Consortium fur Elektrochemische Industrie, Gesellschaft mit baschränkter Haftung, Nuernburg, Germany. Patented Jan. 17, 1911.

981,902. (Paint.) Merriweather J. Waugh, Lincoln, Neb. Patented Jan. 17, 1911.

981,969. (Explosive.) Victor L. Bedier, Seattle, Wash. Patented Jan. 17, 1911.

982,235. (Method of Producing *Ferroboron*.) Chas. A. Hansen, Schenectady, N. Y., assignor to General Electric Co., a Corporation of New York. Patented Jan. 17, 1911.

982,218. (Process of Making *Aluminum and Copper Alloys*.) Joseph Greenville Mellen and Wm. Francis Mellen, Bridgeport, Conn. Patented Jan. 17, 1911.

982,288. (Preparation of *Nitrogen Compounds*.) Einar Honoratus Meyer and Job Morton August Stillesen, Niagara Falls Centre, Ontario, Canada. The process of preparing lime nitrogen consisting in treating calcium carbon with nitrogen, and eliminating carbon from the resulting product. Patented Jan. 24, 1911.

982,466. (Process of Making *Nitric or Other Oxyntrogen Acid and Metal Peroxid*.) Henry Spencer Blackmore, Mt. Vernon, N. Y., assignor to Robert E. Robinson and Daniel C. Spruance, trustees, New York, N. Y. Patented Jan. 24, 1911.

982,524. (*Paint and Varnish Remover*.) James S. Patty, Chicago, Ill., assignor to the Ohio Varnish Co., Cleveland, O., a Corporation of Ohio. Patented Jan. 24, 1911.

982,569. (Use of *Nitrogen-Fixing Organisms* in Agriculture or Horticulture.) Wm. Beegroft Bottomley, London, England. Patented Jan. 24, 1911.

982,620. (*Palmitin Waterproofing Compound and Process of Making the Same*.) Ernest Mas, New York, N. Y. Patented Jan. 24, 1911.

982,945. (Process of Manufacturing *Cement*.) Carl Von Forrell, Hamburg, Germany, assignor by mense assignments to the Atlas Portland Cement Company, New York, N. Y., a Corporation of Pennsylvania. The process consists in adding lime to molten furnace slag in a rotary kiln, in passing heated fluid containing oxygen through said kiln, in mechanically agitating the mixture by the movement of said kiln to separate said material and project portions thereof through said fluid to desulphurize said mixture, and in disintegrating said mixture by fluid jets containing oxygen to produce highly cementitious clinker material. Patented Jan. 31, 1911.

982,992. (*Lead Pigment and Method of Making the Same*.) Alexander S. Ramage, Buffalo, N. Y. This is a method of preparing a lead pigment, which consists in reacting upon a basic lead pigment with an alkali salt of stearic acid, and washing and drying the product. Patented Jan. 31, 1911.

983,124. (*Refrigerator*.) James H. Dagg, Hopkinsville, Ky. Patented Jan. 31, 1911.

983,199. (Catalytic Synthesis of *Methane*.) Fred Bedford and Chas. E. Williams, Sleaford, England, assignors to Bedford Gas Process Company, London, England. Patented Jan. 31, 1911.

983,377. (Method of Protecting *Iron* from Rust.) Albert lang, Karlsruhe, Germany, assignor to Hans Freiherr von Seldeneck, Frankfort-on-the-Main, Germany. The method consists in forming a layer of iron sulfid on the surface of the metal to be treated, and then coloring such surface with a dyeing agent. Patented Feb. 7, 1911.

983,453. (Method of *Treating Ores* by the Blast-Furnace Process.) Fredrik Adolf Kjellin, Stockholm, Sweden. Patented Feb. 7, 1911.

983,521. (Process of Manufacturing *Cement*.) Rudolf Tornay-Schosberger, Budapest, Austria-Hungary. Patented Feb. 7, 1911.

983,727. (Electrochemical Process for Cleaning and Polishing *Silver Plate and the Like*.) Anthony Maurice Kohler, Brixton, London, England. Patented Feb. 7, 1911.

983,783. (*Polishing Compound*.) Charles S. Taylor, Los Angeles, Cal. Patented Feb. 7, 1911.

983,851. (Process of Treating *Bituminous Sandrock*.) Henry F. Williams, San Francisco, Cal. Patented Feb. 7, 1911.

983,854. (Method of Flavoring *Butter*.) George W. Yeoman, St. Paul, Minn. Patented Feb. 7, 1911.

984,525. (Method of Making *Lead Chlorid*.) Edwin O. Barstow, Midland, Mich., assignor to The Dow Chemical Company, Midland, Mich., a corporation of Michigan. The process consists in treating metallic lead with a solution of chlorin in the presence of carbon. Patented Feb. 21, 1911.

984,543. (Compound for *Forming Paving*.) David Crockett, Birmingham, Ala. Patented Feb. 21, 1911.

984,566. (*Detergent*.) Edwi Eli Johnson, San Martin, Cal., assignor, by direct and mesne assignments, of one-half to M. J. Pope, St. Louis, Mo., and one-half to Frank D. Thorne, New York, N. Y. Patented Feb. 21, 1911.

984,578. (Manufacture of *Sugar*.) Henry Armand Joseph Manoury, Paris, France. Patented Feb. 21, 1911.

984,605. (Method of Producing *Nitrogen and Carbon Dioxid* from Gaseous Products of Combustion.) Max Reichel and Heinrich Braun, Berlin, Germany, assignors to Nitrogen-Gesellschaft M. B. H., Berlin, Germany. Patented Feb. 21, 1911.

984,703. (Process of Making *Sulfuric Acid and Electrolytic Iron*.) Alexander S. Ramage, Detroit, Mich., assignor by mesne assignments, to National Tube Company, Pittsburgh, Pa., a corporation of New Jersey. The process consists in dissolving a ferric ore in sulfuric acid reducing the ferric sulfate to ferrous sulfate by sulfur dioxid, electrolyzing the resulting ferrous sulfate solution in an electrolytic cell provided with an insoluble anode and with means for preventing admixture of the positive and negative electrolytes, and depolarizing the anode by sulfur dioxid while maintaining the electrolyte in the region of the cathode substantially free from sulfur dioxid. Patented Feb. 21, 1911.

984,722. (Composition Yielding *Ozone*.) Alexander H. Twombly, Summit, N. J., assignor to Gerard Ozone Process Company, New York, N. Y., a corporation of New Jersey. The composition consists of cocoanut oil and ozone. Patented Feb. 21, 1911.

984,818. (Process of Extracting *Volatile Products from Wood*.) Frederick A. Kummer, New York, N. Y. Patented Feb. 21, 1911.

984,905. (Process and Apparatus for the *Electrolytic Decomposition* of Alkaline Salts.) James Greenwood, London, England. Patented Feb. 21, 1911.

984,925. (Process of *Oxidizing Nitrogen* of Air by Means of *Electric Discharges*.) Karl Kaiser, Wilmersdorf, near Berlin, Germany. The process consists in adding a small amount of gaseous ammonia to a mixture comprising oxygen and nitrogen, and subjecting the mixture thus produced to the action of an electric discharge. Patented Feb. 21, 1911.

983,887. (Process of *Graphitizing*.) Carl Fredrik Jakob Forsell, Lakewood, Ohio, assignor to National Carbon Company, Cleveland, Ohio, a corporation of New Jersey. Patented Feb. 14, 1911.

984,975. (Process for Blackening Brownd *Ferro-Ferric Hydrate*.) James Taylor Carrick, Johannesburg, Transvaal, deceased; Elliot St. Maurice Hutchinson and Robert Gow Ralston, executors. Patented Feb. 14, 1911.

984,090. (Treatment of Gold-Bearing *Antimony Ores*.) John Jones and Horace Sandford Bohm, Shandon Hill, Mount Morgan, Queensland, Australia. Patented Feb. 14, 1911.

984,137. (Process of Hardening *Copper*.) Robert A. Hamilton and Joshua Henry, Connellsville, Pa. The process consists in first making a compound by heating together metallic aluminum and iron pyrites, and subsequently incorporating this compound into a molten mass of metallic copper whose proportion is in excess of that of the said compound. Patented Feb. 14, 1911.

984,143. (Process of Manufacture of *Tungsten Dioxid*.) Anton Lederer, Vienna, Austria-Hungary, assignor to Westinghouse Lamp Company, a corporation of Pennsylvania. Patented Feb. 14, 1911.

984,221. (Production of *Gases* of High Oxidizing Efficiency.) Charles Hornbostel, New York, N. Y. Patented Feb. 14, 1911.

984,312. (*Bleaching Composition*.) Armand Stiegelmann and Erich Dehnel, Ludwigshafen-on-the-Rhine, Germany, assignors to Badische Anilin & Soda Fabrik, Ludwigshafen-on-the-Rhine, Germany, a corporation of Germany. Patented Feb. 14, 1911.

984,477. (*Mineral Paint*.) Minas D. Giffin, Arthur G. Lingley, Terence Cassidy, and James Cassidy, Butte, Mont. Patented Feb. 14, 1911.

984,483. (Process of Manufacturing *Starch Soluble* in Cold Water.) Julius Kantorowicz, Breslau, Germany. The process consists in adding caustic alkalis to starch suspended in solutions of salt of the alkalis by which the starch is precipitated, then adding acids and separating the liquid from the starch. Patented Feb. 14, 1911.

984,503. (*Producing Calcium*.) William C. Arsem, Schenectady, N. Y., assignor to General Electric Company, a corporation of New York. The method consists in heating calcium carbide in a substantial vacuum to a temperature at which it decomposes, conducting away the resultant calcium vapor, and collecting and cooling said vapor. Patented Feb. 14, 1911.

985,164. (Process of Making *Printers' Rollers*.) Charles S. Hadley, Brooklyn, N. Y. Patented Feb. 28, 1911.

985,225. (Process of Making *Steel and Ingot-Iron*.) Benjamin Talbot, Darlington, England. The process consists in subjecting a charge of unrefined molten metal to a surface blast and producing an oxid of iron slag, adding a further charge of metal to be purified, and blowing the combined charges. Patented Feb. 28, 1911.

985,404. (*Paint and Varnish Remover*.) Carleton Ellis, Larchmont, N. Y., assignor to The Chadeloid Chemical Company, New York, N. Y., a corporation of West Virginia. Patented Feb. 28, 1911.

985,528. (*Producing Acetylene Tetrachlorid*.) Erich Hoefer and Martin Mugdan, Nuremberg, Germany. Patented Feb. 28, 1911.

985,559. (Process of Preparing *Bismuth Betanaphtholate*.) Joseph L. Turner, Philadelphia, Pa., and Charles E. Vanderkleed, Collingswood, N. J., assignors to H. K. Mulford Company, Philadelphia, Pa., a corporation of Pennsylvania. Patented Feb. 28, 1911.

985,667. (Process of Obtaining *Sulfur*.) Walther Feld, Honningen-on-the-Rhine, Germany. The process comprises the steps of subjecting to the successive action of hydrogen sulfid and sulfur dioxid, a compound of a metal the sulfid of which is insoluble under the conditions of the reaction, but is decomposed by the sulfur dioxid, whereby free sulfur and a soluble salt are formed. Patented Feb. 28, 1911.

985,709. (Method of Waterproofing *Cement Blocks*.) David F. Shope, St. Paul, Minn. Patented Feb. 28, 1911.

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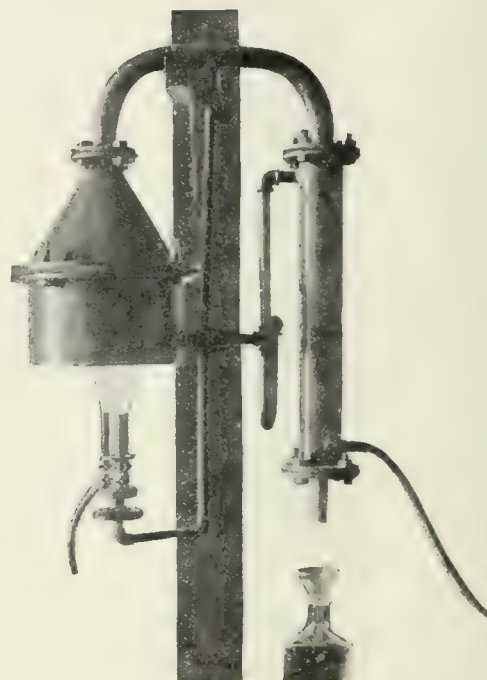
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No. 4.

THE UTILITY OF THE PYROMETER ON CARBURETTED WATER GAS MACHINES.*

By CHESTER S. HEATH, CHICAGO.

Assistant Chemist Peoples' Gas Light & Coke Co.

The pyrometer is an instrument which is used to measure comparatively high temperatures, such as would be found in blast furnaces, muffle furnaces or retorts, reverberatory furnaces, and in gas machines. In connection with the first three furnaces mentioned, the blast, muffle and reverberatory, the pyrometer has been used for some time as an aid to the daily operations, and considerable literature has been written about the pyrometer as used with those furnaces, but the use of the pyrometer in the daily operation of gas machines is hardly past the experimental stage, and practically no literature can be found upon the subject. It is, therefore, the intention of the author to set forth some of the observations made, and results obtained, in the daily use of the pyrometer since it was first installed in a gas machine under his supervision; namely, since September, 1908.

This article is not intended to be a purely scientific treatise on the subject to be discussed, but one which will be of practical value to the man in charge of a gas plant, essaying to give a clearer understanding of the conditions and temperatures found in water gas machines, and to disclose such improvements in the operations of the machines as have been the result of the use of the pyrometers. Consequently, in various parts of the paper commercial terms which are readily understood by men in the gas industry may be used instead of a scientific expression of the same conditions.

For reasons which will be apparent in the discussion of this paper, the class of pyrometers most adaptable to our use is the thermo-electric pyrometer, consisting of a thermo-electric couple or fire-end, a temperature indicator and a temperature recorder connected in parallel to the fire-end by copper wire.

The purpose of the pyrometer in the gas machine is primarily to aid the operator in maintaining uniform temperatures ("heats," according to work's parlance) in the various parts of the machine, at the best temperature for making gas of a desired quality; and, secondarily, to keep the general superintendent in

touch with the operations of each gas maker on both day and night shifts, as shown by the recorder instrument. The object of this paper may be divided into the following classification:

1. A determination of:
 - (a) the most efficient temperature to maintain in manufacturing gas of certain quality.
 - (b) the effects of carrying other temperatures.
 - (c) the range of temperature that is practicable.
 - (d) the limitation of theoretic operation by practical difficulties.
2. Illustration of the method of installation of the pyrometer, so that:
 - (a) the superintendent while at his desk in the office may always be in touch with the operations of the gas makers.

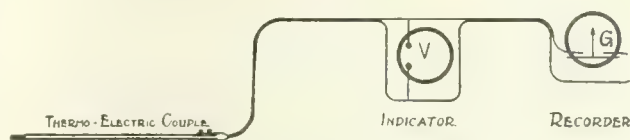


Fig. 1—Diagram of Pyrometer.

- (b) the gas makers may readily watch and control the temperatures in various parts of the machine without leaving the operating valves.
3. A determination of the exact temperature in various parts of the machine while in operation in order that we may have a clearer and more accurate understanding of gas machines.
4. An exemplification of features other than the temperatures of operation, whereby the use of a pyrometer has been of benefit in practice.
5. A discussion of the results obtained and of subsequent improved methods of operating the machine which are primarily due to the aid of pyrometers, in such a manner as will be of interest to the average gas man and will aid him in an understanding of machine operations, even if he has no intention of using a pyrometer in his plant.

Before discussing the question of temperatures most suitable for the proper and practical operation of gas machines it may be well to describe a few of the ordinary conditions and troubles encountered before the pyrometer was used. At that time the gas maker was required to go downstairs to the floor below the charging floor and walk around his machine (in the case of type No. 2) or to climb up two flights of stairs

*From a paper read at the 7th Annual Meeting of the Illinois Gas Association, Chicago, Ill., March 15 and 16, 1911.

and walk around his machine (in case of type No. 1) to sight holes, where he might look into the machine and judge whether the brick were too hot or too cold; or whether the oil spray in the carburetter was working properly or was causing "dark streaks" through the checker brick. It will be readily seen that the operator could not make this trip very often and attend to other necessary work, such as proper adjustment of primary and secondary blast valves, regulations of steam pressure and of oil admitted to the carburetter within the limited time of these operations. Very often it is impossible for the foreman to watch the

Before the instruments were installed, the life of the machine was from 800 to 1,000 hours, due to the formation of lampblack in the superheater. The checker brick would often become so thickly coated with carbon that the resultant back pressure would decrease the amount of gas made to a marked degree; often the machine had to be shut down for two days at a time in order to burn out some of the carbon by admitting air through the checkering doors. When the machine was let down for repairs the bricks would be covered with carbon and ash, burned so hard as to require a pick or sledge and bar at times

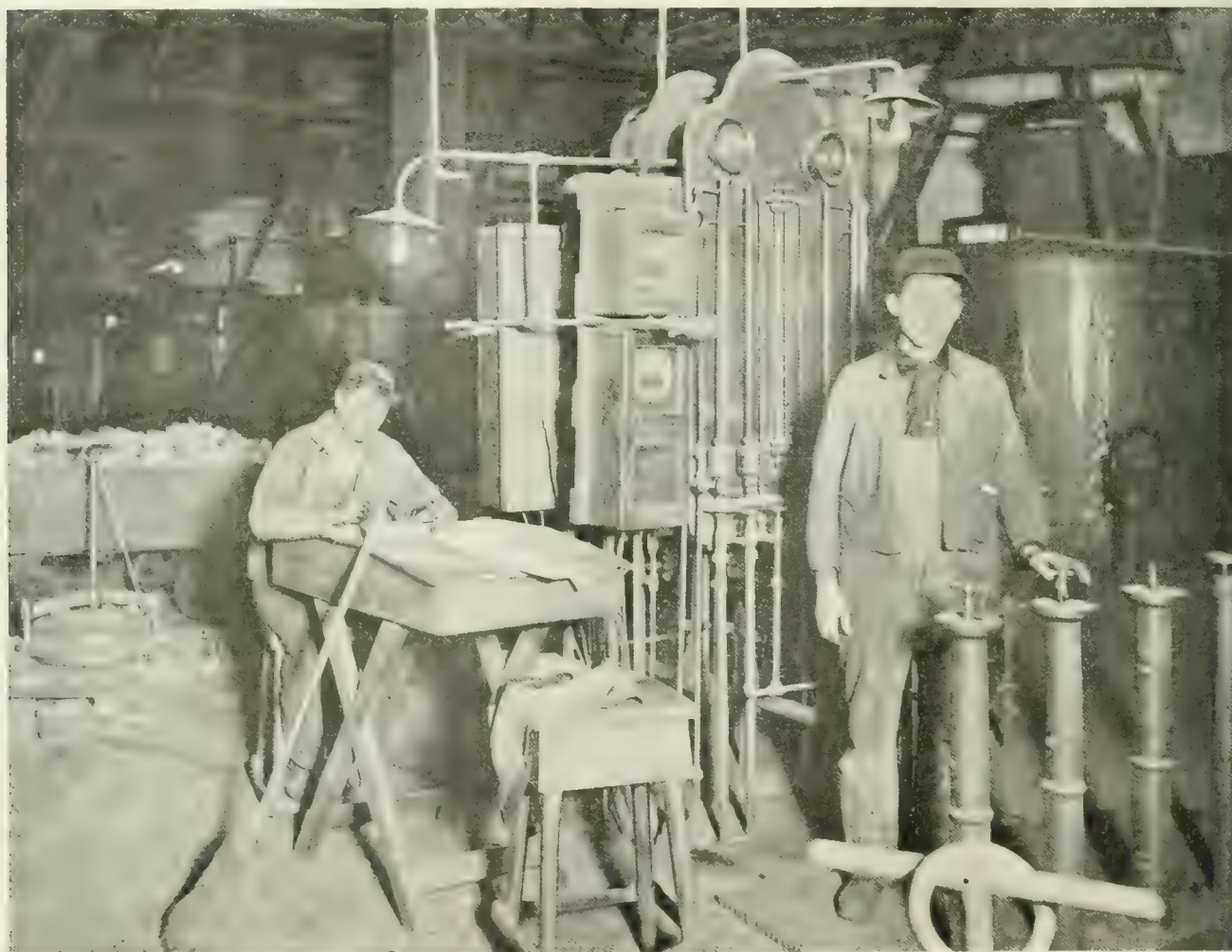


Fig. 2—Pyrometer Indicators in Generator House.

Instruments are located in the center of the picture between the gas maker's desk and the gauge board, the upper one for the superheater and the lower for carburetter.

temperatures in each machine, as he has many other duties which demand his constant attention. It may be noted that in stating the fact that the gas maker would judge the temperature of the brick the word "judge" was selected, for the eye may be deceived in many ways as to the true temperature of brick surrounded by a gas or gaseous vapor, and judgment at its best, as we all know, is subjected to the personal equation. If one authority would say a machine was too cold another would say it was too hot, what should an ordinary gas maker do? There is no personal equation to a pyrometer and, as previously stated, it indicates the true temperature irrespective of the surrounding gas.

to remove them from the upper part of the superheater. Strict attention to the temperatures carried in the operating machine and every other known precaution were employed to overcome these conditions, without any results until the pyrometer told the story. The condition that the pyrometer revealed will be discussed and illustrated in the following paragraphs.

Upon the introduction of pyrometry in the gas industry in Chicago we found that there were three points to be considered in placing the instrument: First, the best position of the fire-ends in the machine; second, the most accessible position of the indicating instrument for the gas maker; and, third, the most desirable position of the recording instrument

for the superintendent. It was necessary to have two sets of fire-ends in each machine to control the temperatures properly, one in the carburetter and one in the superheater. The carburetter temperature was taken from the lower part of the carburetter, while the superheater was taken from the top, which at that time was considered to be, and

repairs, especially if the occurrence was during the night shift.

When the first instrument was installed at Pitney Court Station the temperature at the bottom of the carburetter was not carried as uniformly as we now carry the temperatures. (See Fig. 3 and Fig. 4 for comparison.) It may be noted that with the pyrometer newly installed and before the gas maker knew its purpose the variation in temperatures while the machine was in operation was not excessive, being less than 100° for the 24 hours, excepting for the period just after cleaning, when the temperature had to be carried low in the carburetter (by blasting more on the fire and using less secondary blast) until the superheater was cooled down to a cherry-red color desired. We observe by means of the superheater indicator that the temperature of the upper courses always increases to $1,600^{\circ}$ or $1,800^{\circ}$ during the cleaning or clinkering time, an increase of as high as 400° above operating temperatures. This was also noted in the carburetter (as shown by records, Fig. 5), although not always to such a marked degree as shown in the superheater. When you are informed that it requires from 6 to 10 hours to bring this excessive temperature down* to the desired $1,350^{\circ}$ degrees in the superheater (although the carburetter temperature can be reduced in about an hour) you can readily understand that this is the period during which coke is being

Fig. 3.

was, usually, the hottest part of the machine. The two indicator instruments (one for the carburetter and one for the superheater) were placed directly in front of the gas maker's stool and beside the gauge board, as is illustrated in Fig. 2, and connected to the fire-ends by copper leads 90 ft. in length. The recorder instruments for the various machines were placed along the wall in the superintendent's office, and connected in parallel with the indicator instrument to the fire-ends by copper leads 600 ft. in length, which fact illustrates the adaptability of the thermo-electric pyrometer. By this arrangement the gas maker can watch the temperature rise or fall at all times without leaving his operating valves. He can therefore regulate his primary, secondary and superheater blast valves as conditions demand, instead of operating by a "rule of thumb" method. The superintendent by simply turning in his chair is in constant touch with the generator house. He can tell at a glance which machine is down for cleaning; how long each has taken to clean; how the cleaning time compares with the record of previous days; what temperature is carried by each machine in operation during the present run and for any previous run; which machine may not be in operation and how long they have been shut down; what temperatures were carried during the night shift; and how long a machine has been down for repair work. The recording chart may prove of value in case of dispute as to the exact time an accident had happened on a machine and the length of time required to make



Fig. 4.

wasted and lampblack formed, with the resulting loss of candle power in the gas manufactured.

Improvements in the methods of handling the machine were then devised to prevent this increase of temperature in the brick work during clinkering. The gas machine had been allowed to stand open to the

*By means of careful manipulation of the blast valves, such as increased primary blast (because the temperature of the fire is low after cleaning) and decreased, or often no secondary blast with a large loss of heat and waste of coke from excess gases burning at the stack.

circulation of a natural draft of air through the carburetter and superheater and out the stack. This air would burn any fine coke dust or lampblack which may have lodged on the brick during the previous period of gas making and thereby raise the temperature of the brick far above good operating conditions. To overcome this trouble the circulation of air through the machine was stopped, on some of the machines by closing down the purge cap and on others by closing the up and down run valves in the hydrogen pipe

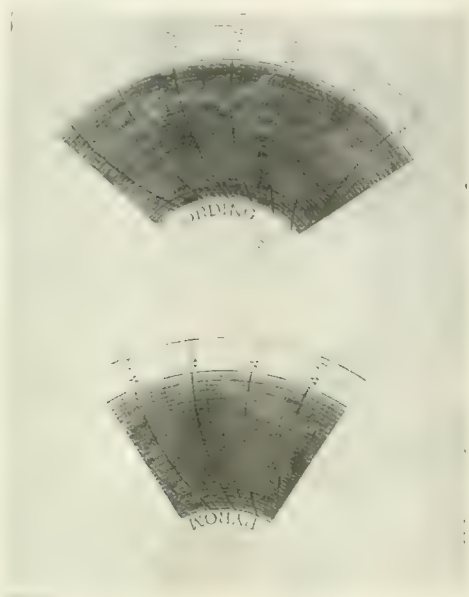


Fig. 5.

between the generator and carburetter, according to local conditions. (Note. Before starting to blast through the machine after clinkering a small amount of steam was turned on to cause a circulation through the machine and prevent small explosions of the gases which may have formed.) By this operation an even temperature was maintained in both carburetter and superheater while the stokers were removing the clinkers, but as soon as the blast was put on the generator the excess air for the first few minutes while the fire was still cold would cause an increase in temperature in the machine for the same reasons. This trouble was not so bad, as it only took about an hour or two to bring the superheater temperatures down to operating requirements, but since the best operating conditions are none too good from the very first minute that gas is being made and sent into the holders, it was decided to blast on the fires until they were hot enough to make gas without allowing the excess air or comparatively cool blast gases to pass through the carburetter and superheater. To accomplish this the charging doors on the top of the generator were opened and the blast gases allowed to pass through until considerable flame showed above the top of the coke. The primary blast valve was then closed, the up-run valve in the hydrogen pipe or the purge cap, as the case may be, was opened, the charging doors were closed, a small amount of steam

turned on for a moment, the primary blast valve finally raised, and the entire machine was then in the best operating conditions before a cubic foot of gas was sent into the holders. With these changes in the operation we find that the temperature in both carburetter and superheater is almost constant during clinkering, excepting for the slight loss due to radiation. (See Fig. 4—cleaning time from 8:05 a. m. to 9:45 a. m.; also Fig. 14—cleaning time from 10:10 a. m. to 12:00 m.)

By these improvements in the methods of handling the machine (which you will note can hardly be called a change in operation during gas manufacture, but rather was a change in the conditions of the machine while idle and when no one would think of watching the temperatures of checker brick) the life of the brick has been increased about 100 per cent and in some cases as high as 175 per cent, and the brick are now quite free from lampblack when the machine is let down for repairs and recheckering. There is no time lost for gas making, as there is no necessity of burning out any lampblack in the machine. Fig. 6 shows the clear-cut outlines of the brick as removed from an 11-ft. water gas machine. In the foreground a portion of the brick from the superheater is shown.

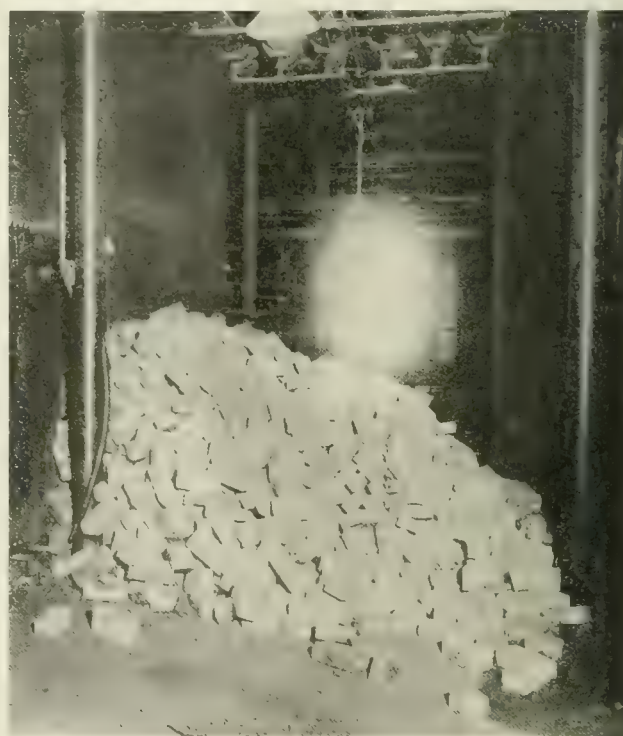


Fig. 6.

The condition of the brick is better illustrated in Fig. 7. The bricks are arranged in the direction of the travel of gas through the machine, starting at the left side of the figure. The first was taken from the top course of brick in the carburetter, the second from the middle course, the third from the bottom course, the fourth from the bottom course in the superheater, the fifth from the middle course and the sixth from the top course. The fifth and sixth bricks have been in

the gas machine twice, as the upper half of the superheater is always checkered with old brick. The first brick shows that some of the lighter fractions of the gas oil have been burned on the brick, which fact is noticeable only on the first and sometimes second course. This trouble is overcome in a large measure by delaying the admission of oil for a fraction of a minute after the steam has been turned on, thereby reducing the temperature of the upper courses so that the cold oil will not be over-cracked, or in work's parlance, taking the "sharp heat" off the top courses. When the machine is shut down for repairs the brick immediately begin to cool and may easily be removed by a long handled hook or by hand when sufficiently cold, whereas it previously required two or three days to burn out the carbon and three or four more to cool off the bricks which had become almost white hot by

chime to fix the oil vapors as gases when the brick are old. It will be noted in comparing the following table with the preceding that although the drop in temperature of the brick is somewhat less in the bottom of the carburetter or 17 courses from the oil spray than it is nine courses from the oil spray when the brick are old, yet the drop is very slight in the bottom course when the brick are new.

No. Machine.	Date when rechecked.	—Drop in temperature—	
		New brick.	Old brick.
No. 7 machine.....	Oct., 1909	100	250
No. 8 machine.....	Apr., 1909	75	275
No. 8 machine.....	Oct., 1909	75	300
No. 9 machine.....	June, 1909	100	200
No. 9 machine.....	Jan., 1910	75	150

This increased drop in temperature upon the addition of the same quantity of oil per run is a fair indication that the machine needs new checker brick:

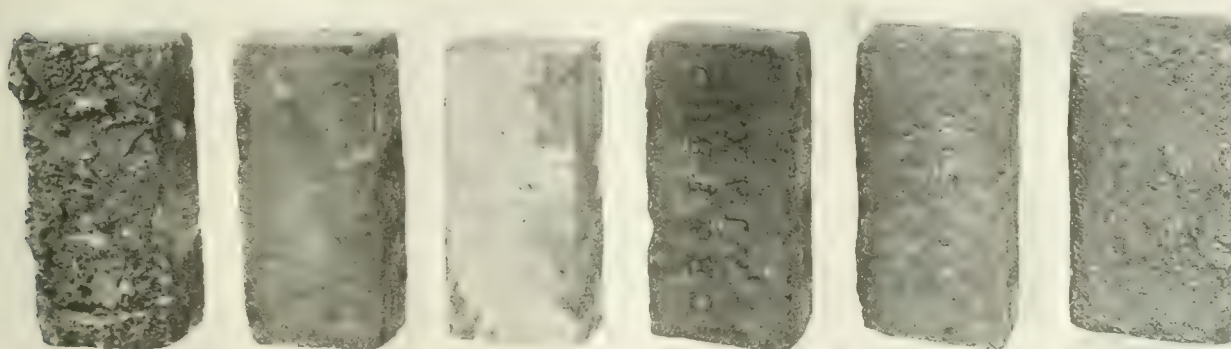


Fig. 7—Checker Brick from Gas Machine Using Pyrometer.

Note the freedom of brick from lamp black. The top course in carburetter shows some carbon on which is difficult to prevent on the top course; other bricks show a thin layer of red soot dust. The last two bricks have been used twice as the upper half of the superheater is always recheckered with old bricks. These bricks have been taken from an 11 ft. gas machine after 1,000 cycles in actual gas making time.

the intense combustion of this fine carbon.

With the carburetter and superheater checkerwork as free and open the day the gas machine was let down for repairs as it was the day it was started, it became necessary to determine in a general way by means of the pyrometer when the life of the brick was exhausted. When the brick are new there is only a slight drop in temperature in the bottom of the carburetter with the addition of a given quantity of oil but as the brick become old the drop in temperature is greater for the same amount of oil used. The following table shows the loss in temperature as indicated by the pyrometer with its fire-ends placed in the middle of the carburetter or nine courses down from the top.

No. Machine	Date when rechecked	—Drop in temperature—	
		New brick	Old brick
No. 7 machine.....	June, 1909	150	300
No. 7 machine.....	Oct., 1909	200	350
No. 8 machine.....	Oct., 1909	250	500
No. 9 machine.....	June, 1909	250	400
No. 9 machine.....	Oct., 1909	125	325
No. 10 machine.....	July, 1909	175	375
No. 10 machine.....	Nov., 1909	200	325

This increased drop in temperature is due in a large measure to the fact that the heat stored in the brick is not as quickly conducted to the surface of the old brick as it is in the new and therefore, more heat is required from the courses farther through the ma-

chine as the gas work's foreman would say, the machine "won't hold her heats."

In order to obtain information regarding the distribution of heat through various types of gas machines and the variation in temperature at different points in the machine under operating conditions, I took simultaneous records of the temperatures at given points for 14 consecutive days, noting the changes in operation. The diagram of the three types of carburetted water gas machines give a clear conception of the points at which these continuous records were taken. (See diagram of types 1, 2, 3, Figs. 8, 9 and 10). The black areas with their corresponding numbers indicate the position of the fire-ends in each type, numbering from No. 1 on, in the direction of travel of gas through the machine.

In type of gas machine No. 1, records were taken of the temperatures of down-run gases at the base of the hydrogen pipe just above the "Williamson" water-sealed hot valve; of the gases at the top of the hydrogen pipe; of the first course of brick in the carburetter; of the 12th course of brick in the carburetter near the center wall between the carburetter and the superheater; of the 20th course of brick at the farthest point from, and at right angles to, the center wall; of the 23d course as shown in the dia-

gram; of the 23d course near the center wall; of the 39th course of brick (39th from the oil spray or 8th from the bottom of the superheater); of the 50th course of brick; of the 61st course of brick near the center wall; of the 61st course away from the center wall; of the 62d course at right angles to the center wall (top of superheater).

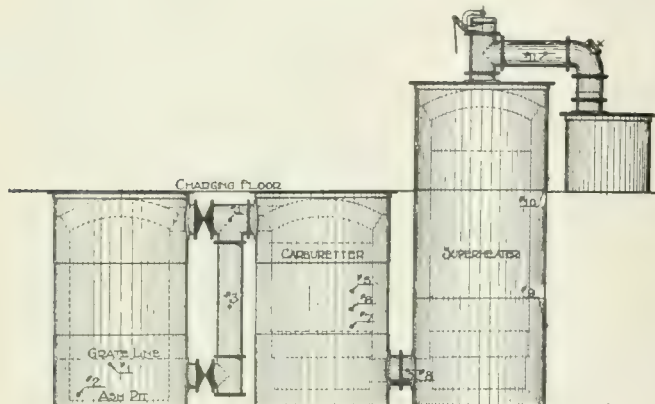


Fig. 8—Type No. 1.

In type of gas machine No. 2 records were taken of the temperatures of down-run gases in the generator 4 ins. below the grate bars; in the ash pit; and in the hydrogen pipe as indicated in the diagram; of the temperature of up run, down run and blast gases at the top of the hydrogen pipe near the "Levy" valve; of the 9th course of brick from the oil spray in the carburetter; of the 13th course; of the 17th course; of the gases passing through the connection pipe be-

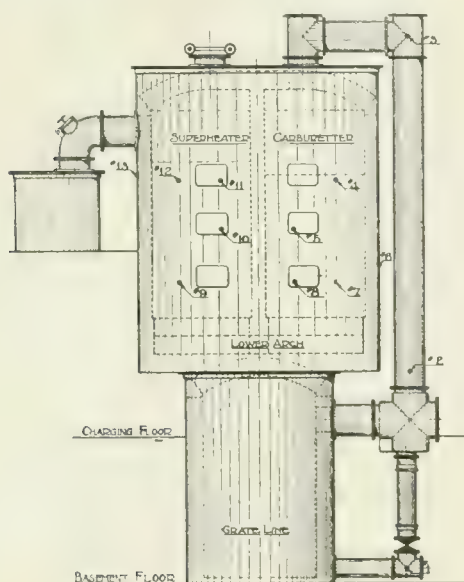


Fig. 9—Type No. 2.

tween the carburetter and superheater; of the 19th course (or first course in superheater); of the 56th course (or top of superheater); and of the gas in the take-off pipe.

In type of gas machine No. 3 readings were taken from the top course and bottom course of brick in one shell and the bottom course and top course of the twin shell.

To discuss the results obtained at each of these points in detail would require more time than may well be taken in this meeting, but I shall be pleased to go more fully into the discussion of any results or conclusions in which you may be especially interested.

The average temperatures obtained by series of tests on type No. 1 water gas machine may be found in Table I. The first column in the table indicates the point at which the temperature was taken (see Fig. 2); the second column indicates the number of courses of checker brick between each position of the fire-end and the oil spray; the third column indicates the maximum temperature at each point, i. e., the temperature attained after blasting; the fourth

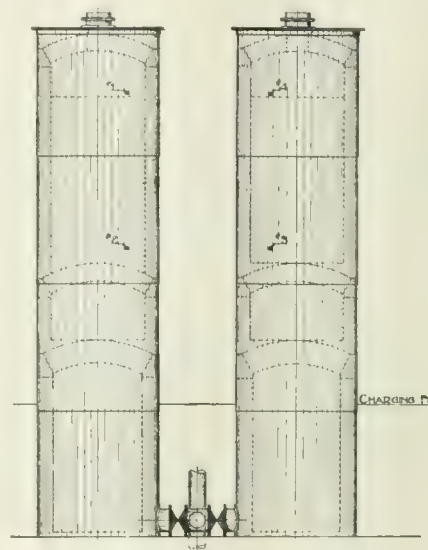


Fig. 10—Type No. 3.

indicates the minimum temperature, i. e., at the end of the run; the fifth indicates the loss in temperature at each point upon making gas; and the sixth indicates the average temperature carried at each point.

TABLE I.

Test Course of	Point	brick.	Max.	Min	Drop	Average	Remarks.
1						625	Center of 12 in. pipe
						1610	End of blasting
2						1360	End of up run
						720	End of down run
						1500	End of blasting
3						1220	End of up run
						920	End of down run
4	1	1650	1000	650	1270		
5	12	1350	1300	50	1325		12 in. from wall
6	20	1270	1170	100	1220		
7	23	1300	1240	60	1270		
8	23	1335	1300	35	1320		12 in. from wall
9	30	1295	1265	30	1280		
10	50	1310	1300	10	1305		12 in. from wall
11	61	1320	1320	0	1320		12 in. from wall
12	61	1310	1290	20	1300		
13	62	1330	1300	30	1315		

In this type of gas machine the carburetter and superheater are built side by side within a single shell, separated by a 14-in. center wall of brick extending from the lower arch up to the top of the machine. There are about 31 courses of brick in the carburetter

and 34 in the superheater. It will be noted from the above table that the temperatures are quite uniform on both sides of this wall, as shown by tests taken at points 5, 8, 10 and 11, and also that the drop in temperature during the run is very small at all these points. Evidently this wall acts as a reservoir of heat tending to maintain more uniform temperatures throughout the fixing chambers.

The first few courses of brick performed the "heavy duty" of vaporizing and cracking the oils as is strikingly indicated by the plotted curve (see Fig. 11). The cooling effect of the oil on the first course is very

generator the temperature of the blast gases will seldom exceed 1000° F., but as each successive blasting increases the temperature of this upper layer of coke, the gases become hotter until they may reach 1750 to 1800° F., as was found after three successive up runs. The down run cools off the top of the fuel bed to such an extent that the temperature of the blast gases averaged 100° F. lower than after the preceding up run, all other conditions being equal. It was found that the average temperature of the blast gases at the end of the blasting period was about 1610° F. at point No. 2 and about 1500° F. at point No. 3, showing a

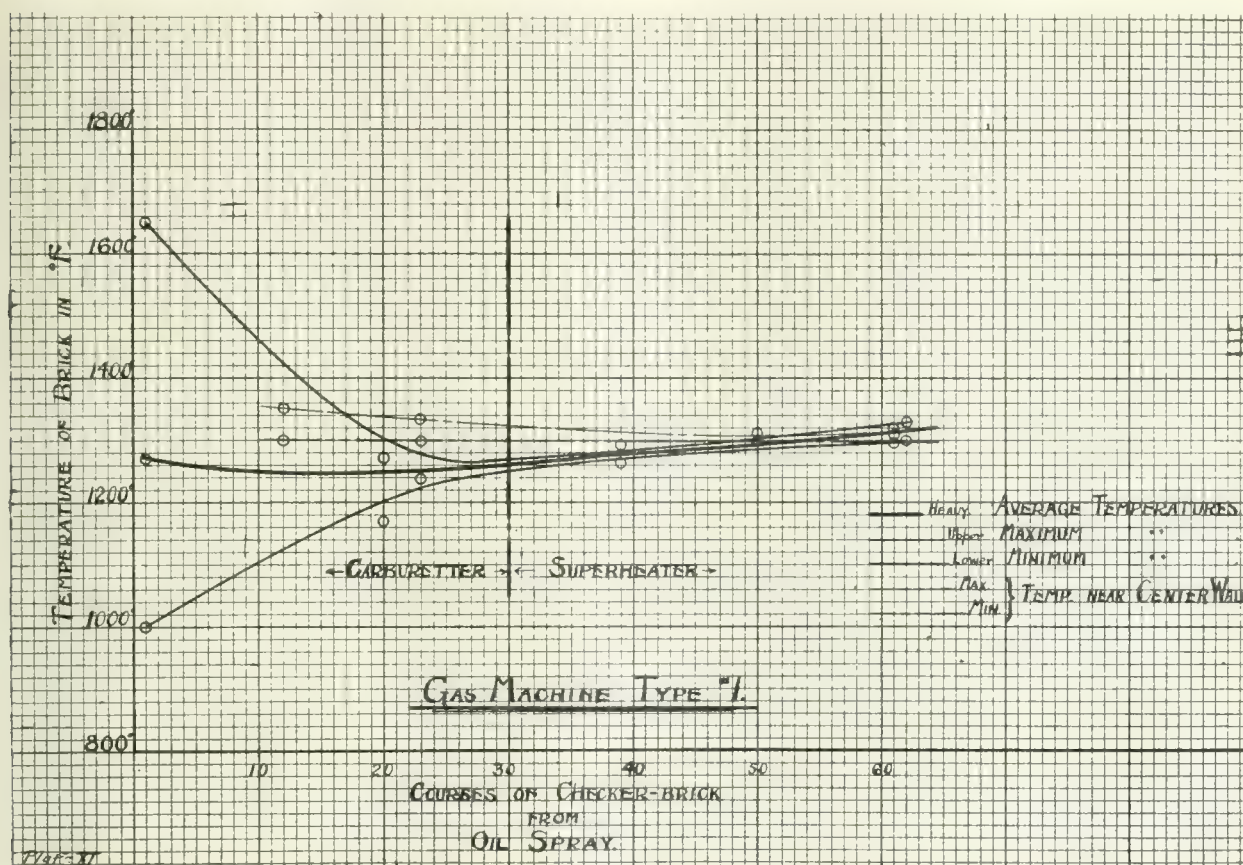


Fig. 11.

marked, being about 650° F., while the drop in temperature at the 23d course (point 7) in the same relative position as that taken at the first course (point 4) is only 60° F.

The temperatures of the down run gases taken at the base of the hydrogen pipe (point 1) averaged 625° F., due to the cooling action of the grate bars, blast boxes, etc., as shall be discussed more fully with type No. 2. These down run gases are heated to about 700° F., at a point 3 ft. above the "Williamson" hot valve (point 2) and to 920° F. at the top of this hydrogen pipe (point 3) the temperature of these gases is increased by the heat stored in the fire-brick lining of the pipe during the blasting period. The temperature of the blast gases depends very largely on the condition of the fire, as we have known in a general way. When a fresh charge of coke is put into the

loss of 110 degrees due to radiation from the hydrogen pipe. The temperature of the up run gases at the end of the run averaged 1360° at point No. 2 and 1220° at point No. 3, a loss of 140° due to radiation.

A test was made to determine the effect of radiation from the shell of the machine upon the temperature of the gases in the checkered chamber, and thereby decide what should be the minimum length of the fire-end. The results are shown in the following table:

TEMPERATURES					
Thickness of shell.	At shell.	12 in. from shell.	15 in. from shell.	28 in. from shell.	54 in. from shell.
18 in.					
18 in.	1230	1280	1290	1300	1300?

In a gas machine with an 18-in. shell the fire-end should be at least 4 ft. long.

The average temperatures obtained by series of

tests on type No. 2 water gas machine may be found in Table II.

In this type of gas machine the carburetter and superheater are in two separate shells connected at the bottom by a 24-in. pipe lined with fire brick. The curve plotted from Table II (Fig. 12) shows very clearly that the top nine courses of brick perform the "heavy duty" in cracking the oils; that the average temperature of the brick is lower the farther the course is from the point at which the oil enters; that the variation in temperature during each run becomes less as the distance from the source of

almost a perfect circle. In the take-off pipe with the fire-end in the cross above the wash-box (point 11) we find that during the run the temperature of the gas averages 1225° when the temperature at point 10 is 1275° F.

It is well at this point of the discussion of the temperatures of the brick in the two types of water gas machines to compare the character of the two curves. (Fig. 11 and Fig. 12.) We find that in type No. 1 machine the oil has been completely "cracked" before it leaves the carburetter and the superheater performs its true function of fixing the gaseous hydrocarbons.

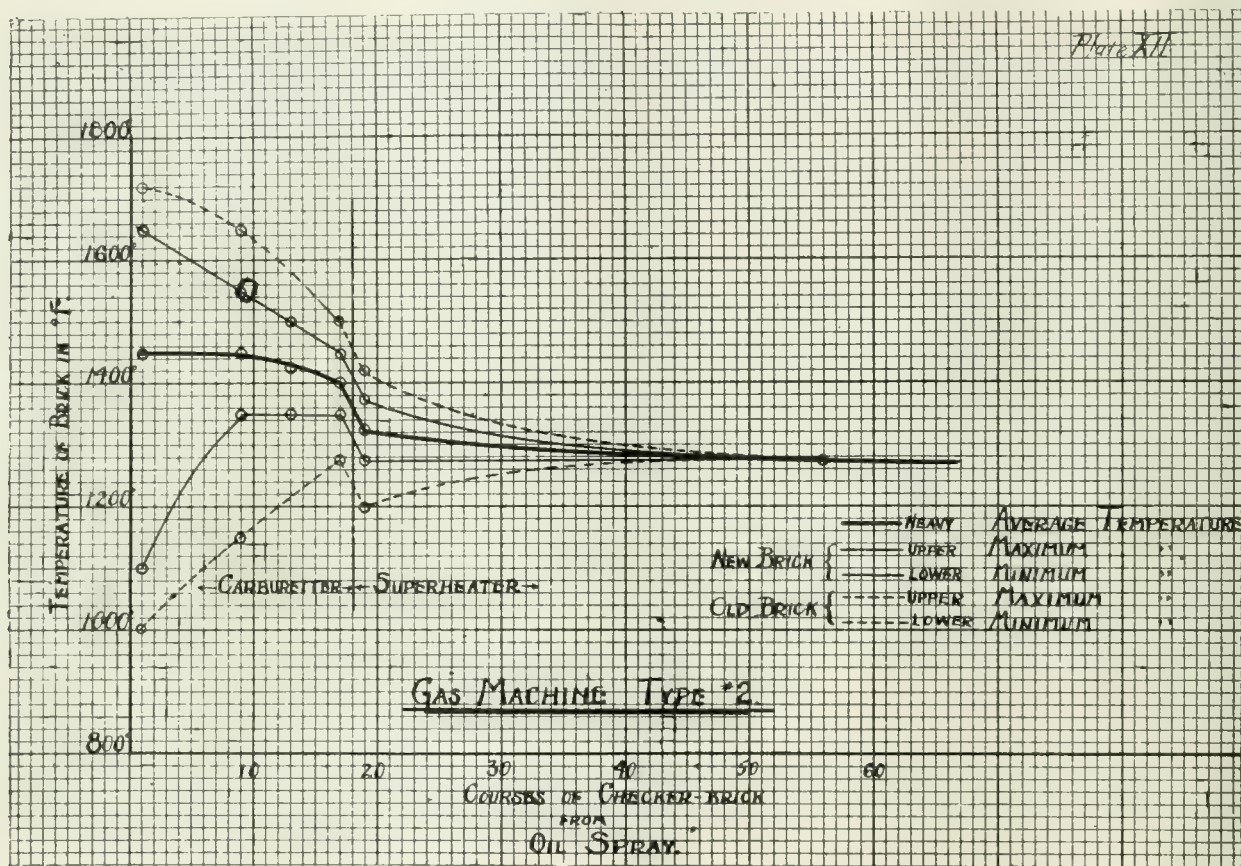


Fig. 12.

oil increases; that the variation at the bottom of the superheater is about the same as at the bottom of the carburetter (excepting at such times as the gas maker uses the superheater blast for one or more runs when the variation is about 250° .*). The loss in temperature in passing through the false bottoms of the carburetter and superheater and the 24-in. connecting pipe is clearly shown on the curve. The temperature at the top of the superheater (point No. 10) recorded

*Note. When the superheater blast valve is opened wide the velocity of the air evidently drives the zone of combustion higher than No. 9 hole as the temperature remains about constant for the first part of the blast, while the temperature at No. 10 hole rises about 50 degrees. Toward the end of the blast when the Carbon Monoxide in the blast gases increases, the zone of combustion is brought lower and the temperature at No. 9 increases about the usual amount (100 deg.). If the superheater blast is opened a small amount at first and increased as much as may be necessary during the latter portion of the blasting, the temperature of the brick at point No. 9 increases about 250° deg., while at No. 10 it increases only 25 deg., indicating that the zone of combustion is lower in the superheater.

In type No. 2 machine the curves indicate very clearly that the oil has not been fully "cracked" in the carburetter; that a large portion of the work must be completed in the superheater, in addition to the "fixing" function of that chamber; and that these partially decomposed hydrocarbons are subjected to a sudden cooling of about 100° F. in passing through the 24-in. pipe connecting the carburetter and superheater (which condition is not found in type No. 1). We must therefore conclude that type No. 1 is a much better proportioned machine than type No. 2.

A short description of the test on the temperatures of the down run gases before entering the carburetter may be of interest. The fire-ends were especially prepared to secure the temperatures of the gases quickly and accurately. A $\frac{1}{2}$ -in. iron pipe 4 in. shorter than the fire-end was used as a jacket to protect and

support the long wires. The hot junction extended 4 in. beyond the open end of this $\frac{1}{2}$ -in. pipe so as to come in direct contact with the down run gases, while the cold end was held in a stuffing box packed with asbestos at the outer end of the $\frac{1}{2}$ -in. pipe. Three fire-ends, prepared in this manner, were placed in the bottom of the generator about 4 ins. below the grate bars (point No. 1); in the ash pit under the blast boxes (point No. 2); and in the hydrogen pipe between the generator and the carburetter (point No. 3); the results of this test (see Table No. 2) indicate much lower temperatures of the down run gases than was anticipated. The clinker and grate bars, cooled by the cold air blast and by the up run steam, decreased the temperature of the down run gases to an average of 1025° ; the cold blast boxes reduced the temperature to 625° (a loss of 400°); and the radiation from the lower part of the hydrogen pipe re-

duced the temperature to 475° (a further loss of 150°). The relative temperatures found throughout the type No. 2 gas machine are illustrated by the Composite Chart (Fig. 13). The chart is composed of records taken from eight parts of the machine during the time between 11:30 a. m. and 2:30 p. m., and arranged in the order of the gas travel. The numbers indicate the position of the fire-end in the gas machine from which the records were taken, as shown in the diagram of type No. 2.

TABLE NO. 11

Test Course of	Point, brick	Max	Min	Drop	Average	Remarks
1					1025	4 in. below grate bars
2					625	In ash pit
3					475	In hydrogen pipe
4		1750	1400		1575	Up run gases
5	9	1650	1150	500	1400	Old brick
		1550	1350	200	1450	New brick
6	13	1500	1350	150	1425	New brick
7	17	1500	1275	225	1390	Old brick
		1450	1350	100	1400	New brick
8 connection	1600	1150	450	1375		Gas temp pipe]
9	19	1550	1300	250	1425	With superh. blast
		1375	1275	100	1325	Without superh. blast
10	56	1350	1300	50	1325	With superh. blast
		1275	1275	0	1275	Without superh. blast
11					1225	In take-off pipe

duced the temperature to 475° (a further loss of 150°).

The relative temperatures found throughout the type No. 2 gas machine are illustrated by the Composite Chart (Fig. 13). The chart is composed of records taken from eight parts of the machine during the time between 11:30 a. m. and 2:30 p. m., and arranged in the order of the gas travel. The numbers indicate the position of the fire-end in the gas machine from which the records were taken, as shown in the diagram of type No. 2.

The third diagram shown (Fig. 10) is upon a type of water gas machine which is seldom seen in use at the present day, but it illustrates very markedly the use to which the pyrometer could have been put as a decided aid in operations in the past. This machine has two generators side by side connected by pipes and valves, above each of which is a fixing chamber filled with checker brick at which point oil is admitted for carburetting the gas. Above this short chamber is another, but taller, fixing chamber likewise filled with checkered brick, much for the same purpose as the superheater in the other types. Each shell has its own take-off pipe, purge cap, wash box, etc. When up runs are made the steam enters the bottom of each generator and the two shells are operated as independent machines. When down runs are made the

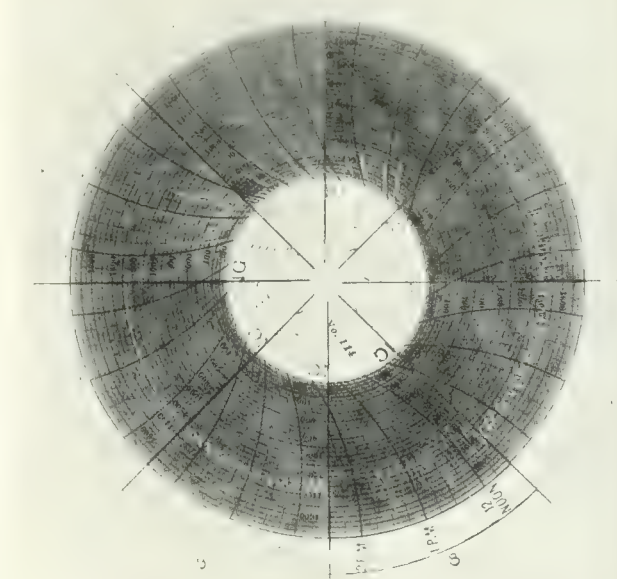


Fig. 13—Composite Pyrometer Records of Comparative Temperatures.

primary and secondary blast on each machine can then be so manipulated with the aid of the four indicators that the colder shell may be brought to the desired temperature without heating the other shell to an excessive temperature during the blasting period. In this type of machine the top of the superheater may be quickly cooled to the desired temperature by reason of the direct effect of the down run steam.

It may be of interest to you at this time to note a few features of more or less importance in the practical operation of a gas machine which I have observed incident to the use of the pyrometer.

The decided advantage a gas maker has in starting a new machine with the constant and accessible aid of the pyrometer by heating the bricks uniformly and gradually throughout the machine without attaining an excessive temperature in the carburetter, is clearly shown by the fourth chart reproduced with this article. (See Fig. 14.) The record of temperatures at the bottom of the carburetter indicates that the blast

was turned through the cold checkerwork at 1 p. m. August 25, 1910, and that the brick were slowly and steadily heated to a temperature of 1350° at this point covering a period of two hours. At 3:40 p. m. the machine was shut down for 10 minutes to readjust the oil meter. From 10:10 a. m. on the following day to 12 m. the machine was cleaned and clinkered. Note the absence of any excessive temperatures during the cleaning time but rather the slight decrease due to radiation.

When gas is made with coke (blasting 4 minutes and running 6 minutes) it is very difficult to detect from the chart, recording the temperatures in the 17th course from the oil spray, just how long the primary blast was used before the secondary was opened; but, when hard coal is used (blasting 6 minutes and running 6 minutes) the chart shows very distinctly the point at which the secondary was opened.

A steam run is readily detected on the chart thus enabling the superintendent to note the carelessness of a gas maker, who may allow the meter to pass an excess of oil for carburetting during a couple of runs and then make the next a steam run to bring the meter statement on the time sheet to the reading of the meter itself. When the gas makers realize that it is possible to check their work they do become more careful. There is a certain moral influence surrounding the pyrometer, a halo of mystery enveloping that chart over in the office, which the average gas maker greatly respects. He will even refrain from smoking in its presence for some months.

Another feature which is readily detected by the carburetter instrument is the action of the oil spray. If it is not properly adjusted, some of the openings become closed with carbon, or the spiral fails to rotate (as in the Johnson spray), the oil will cool the brick in one portion more than another, causing the familiar "dark streaks" in the carburetter. In this connection I might say that the choice between the use of different types of sprays has been simplified in a large measure. On the one hand a stationary spray was recommended which had no movable parts and required a minimum of repairs but gave a slight "dark streak" in the center of the carburetter which condition could not be avoided, while on the other hand a spray was recommended which had to be raised above the carburetter arch after every run to keep it from the heat of the blast gases and which had movable parts, necessitating occasional repairs, but gave no dark streaks when adjusted. The many advantages of the former spray were not of sufficient weight for its adoption when the pyrometer indicated that this "dark streak" represented a temperature about 1000° F., whereas the temperature of the brick 2 ft. from the center was 1350° F.

The instruments in the carburetter and superheater aid the gas maker in determining the condition of his generator. If the stack gases show excessive flame at the purge cap (due to the combustion of an excessive

supply of carbon monoxide formed in the generator) when the temperatures in the carburetter and superheater are at the desired points, the fire is too hot and the primary blast valve should not be opened so wide during the following blow. When the fire is not hot enough to generate sufficient carbon monoxide to heat the checker work to the desired temperature and to show a thread of blue flame at the stack toward the end of the blow, more primary blast should be given to the generator. By operating in such a manner the pyrometer will aid materially in maintaining a good fire, and reduce the amount of coke wasted by excessive blasting with the primary.

I recall one very interesting incident which occurred about a year ago. The recording chart taken from the instrument on the morning of a sweltering July day indicated a decided drop in temperature every

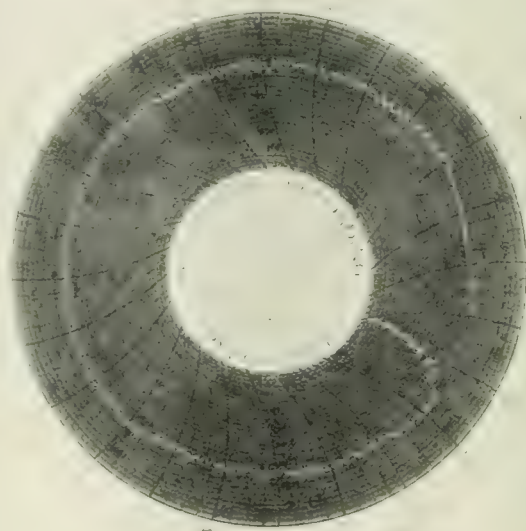


Fig. 14.

run or two during the previous afternoon and night. After the oil spray had been examined and found to be in excellent condition, the chart was again referred to and studied more closely. It was noted that there was a certain regularity to the repetition of this increased variation in temperature (about twice the average variation); that it occurred every second and third run in the same order in which the down runs occurred. Upon examination of the hot valve between the top of the generator and the hydrogen pipe, a small quantity of coke breeze was found in the seat of the valve which prevented a tight seating of the gate during a down run and permitted live steam to escape from the top of the generator to the carburetter, causing the increase in the cooling of the brick during the run. It is needless to say that the trouble was immediately remedied.

Before giving a short summary of the results obtained with the pyrometer in carburetted water gas machines it will be hardly necessary for me to empha-

size a few points which I consider of chief importance. Lamp black, I believe, is formed to a large extent when the machine is started up after rechecking with new brick. The heat in the top of the carburetter is allowed to run too high so as to obtain the desired cherry red heat in the superheater as soon as possible. Lamp black is also formed by excessive temperature on the checkerwork after the machine has been idle during the clinkering time caused by the natural draft through the carburetter and superheater. Methods employed to overcome these difficulties have been previously discussed.

The theoretic operation of a water gas set would be the manufacture of gas with as high a temperature as possible in the checker brick without the formation of lamp black and thereby obtain the greatest possible proportion of fixed, gaseous hydrocarbons. In practice we find other factors entering the problem which limit the temperatures to be carried in the superheater, and it will be noted how narrow these limits are. Temperatures above 1500° F. produce considerable lamp black in the superheater. A machine which carries 1450° F. in the superheater would produce some lamp black and would fill the works with naphthalene in a short period of time. One carrying 1400° F. produces some naphthalene trouble but practically no lamp black. Machines operating from 1300° to 1350° F. produce hardly a trace of naphthalene in the entire plant. Under 1250° F. the machines have dirty seal pots showing tar and uncracked oils. The practical limitations in gas machine control are then complete decomposition of the heavier hydro carbon oils and no serious trouble from the formation of naphthalene, i. e., clean seal pots and a minimum amount of naphthalene. Best practice keeps the temperature of the superheater between the rather narrow limits of 1300° to 1400° F. These temperatures are based upon the use of gas oil having a gravity between 33° Bé and 35° Bé of approximately the following analysis:

Fraction. From	Per cent weight.	Specific gravity.	°B.
0° to 300° F.	1.79	.7666	54°
300 to 400	6.30	.8016	46°
400 to 500	25.49	.8299	40°
500 to 600	36.12	.8541	35°
600 to 700	24.44	.8820	30°

700° and above 5.86 Residue Tar.

Sp. gr. of oil=.8630, corresponding to 33.2° Baume

Flash point=164° F.

Burning point 196° F.

In a general way it may be stated that with a given quantity of oil used to carburet the water gas a temperature of approximately 1250° in the fixing chamber will yield a gas of about 16 per cent methane with correspondingly low heat value, while temperatures approximating 1350° would yield a gas with nearly 20 per cent of methane, and relatively high B. T. U. and temperatures approximating 1400° would yield a gas containing as high as 22 per cent methane.

In conclusion I wish to summarize in a brief way the principal results obtained by the pyrometer, direct-

ly or indirectly, in the practical operation of a carburetted water gas set.

The first and probably most essential point is that a uniform temperature can be maintained in the machine and unless the gas maker has had considerable experience this is a condition difficult to obtain without an instrument.

Second, the carburetting and fixing chambers have been closed during the clinkering time in such a manner as to prevent uneven and excessive temperatures.

Third, methods of operating the blast valves have been devised so as to maintain a healthy condition of the generator fire, with a minimum waste of carbon monoxide gas burning at the stack.

Fourth, the absence of lamp black on the checker work when the machine is shut down for repairs is an indication that the oil for carburetting has been utilized to the best advantage.

Fifth, the freedom of the works from naphthalene has solved many problems, especially the disadvantages of using oxide saturated with the light, flaky crystals in purification of the gas.

Sixth, the gas maker can operate his machine more carefully and intelligently with the constant and accessible indication of the temperatures in various parts of the machine.

Seventh, the continuous record of temperatures carried on each machine by night as well as day shifts in the office where the superintendent may find ready reference, is of untold value.

Eighth, the indicator and recorder will easily show the condition of the oil spray. The charts indicate very clearly the time taken by each machine in charging, clinkering, repairing or waiting.

Ninth, the pyrometer may be utilized to indicate the useful life of the checker brick.

Tenth, a better knowledge of the exact temperatures found in various parts of the machine becomes very useful in practical operations.

Eleventh and final, the extended use of the pyrometer in operating all the machines in the plant under a given temperature for a considerable period of time, and subsequently under other known temperatures for sufficient time, has proven of great value in determining the practicable range of temperature for good operating conditions in the carburetted water gas machines.

The value of the production of gold in the United States in 1910, according to the United States Geological Survey, fell below that of the record output of 1909, which was nearly \$100,000,000. Preliminary statistics compiled by the Director of the Mint indicate that in 1910 the value of the total production of gold in the United States (including Alaska and the insular possessions) was \$96,055,214, a decrease of \$3,618,186 from the value of the output in 1909, which was \$99,673,400.

A CHEMICAL STUDY OF A 50-HORSE-POWER SUCTION PRODUCER.*

By MR. B. M. FERGUSON.†

The computation and results given in this article are based on test data taken with the object of making a chemical study of the operation of the gas producer, and also to determine the thermal efficiency of the machine. No meter was available to measure the gas generated, and various methods were used to determine this quantity, as will be shown below.

The fuel used was a Virginia pea-anthracite. It was not a good quality of coal for this purpose, as it contained a high percentage of ash and clinkered considerably, constant poking at short intervals being necessary to keep the fire clean and to promote combustion. A test of a similar coal has been made at the government testing plant at St. Louis.

The analysis of the coal was made by the standard method. The amount of combustible matter present in the clinker taken from the producer plant from day to day was determined from a sample taken from each day's run, the object being to determine the amount of available combustible matter that came through the producer unburned. The residue was weighed and represents the per cent of ash in the clinker, and 100 minus this quantity equals the per cent of carbon in the clinker.

Calculations of the number of cubic feet of gas generated in the producer, from the composition of coal and gas as determined by analysis, were made as follows, the figures representing an average analysis of the gas produced:

Component.	Per cent.
Carbon dioxide.....	6.66
Oxygen	0.59
Carbon monoxide.....	19.59
Methane	0.95
Hydrogen	13.49
Nitrogen	58.72

Average B. T. U. per cubic foot, calculated from analysis at 0° and 76 cm. = 125.5.

Taking carbon dioxide plus carbon monoxide as the basis of calculation, 1 cubic foot of gas contains 26.25 per cent of these gases. One gram-molecule of carbon (12 grams) will occupy, as gaseous oxides, 22.38 liters of space as a gas, at temperature of 0° and 76 cm. pressure. That is, for every 12 grams of carbon available in the coal, 22.38 liters of a mixture of carbon dioxide and carbon monoxide will be produced. The total coal used in first week's test was 4,076 lbs.

From the analysis of the coal we know it to contain 76 per cent of carbon; therefore, available carbon in the coal equals:

$$4,076 \times 0.76 = 3,100 \text{ lbs}$$

$$3,100 \text{ lbs.} \times 453.6 = 1,406,160 \text{ grams.}$$

Since every 12 grams = 22.38 liters

$$\frac{1,406,160 \times 23.38}{12} =$$

$$\frac{2,622,488}{28.3} = \text{cubic feet} = 91,253.$$

Or,

$$\frac{2,622,488}{28.3} = \text{cubic feet} = 91,253.$$

If the gas were a pure mixture of carbon dioxide and carbon monoxide the coal used would give off 91,253 cu. ft. of gas, but the gas obtained contained only 26.25 per cent of carbon monoxide, plus carbon dioxide; therefore, the cubic feet of gas given off equal

$$\frac{91,253}{.2625} = 347,500.$$

The B. T. U. equivalent of 1 cu. ft. of gas is 125.5; therefore, total B. T. U. in 347,500 cu. ft. = 347,500 × 125.5 = 43,510,000.

B. T. U. equivalent of 4,076 lbs. of coal = 4,076 × 11,285 = 48,190,000.

The efficiency of the producer would, therefore, be $\frac{43,610,000}{48,190,000} = 90.5$ per cent, if all the carbon available were actually converted into gas.

The actual efficiency of producer was calculated from the data obtained during the test, taking into account the loss of carbon going into the clinkers. Since no gas meter was available to measure the amount of gas generated in the producer and passing into the engine the following methods were taken as a basis of calculation:

The first method depends on the weight of coal added and the weight of ash taken out during the test. Then, from the analysis of the coal and ash and also from the analysis of the gas generated, the cubic feet of gas generated can be calculated.

If a complete set of data be taken during the test this ought to prove the most accurate method; but in this test the amount of coal and ash in the producer was not weighed, and reliable calculations cannot be made on this basis. From the analysis of the clinker the per cent of carbon was found to be as follows:

	Per Cent of Carbon.	Per Cent of Ash.
For the first week.....	44.38	55.61
For the second week.....	57.59	42.41
Average for the two weeks.....	51.59	48.41

Per cent of ash in original coal = 16.53. Therefore, per cent of carbon lost in clinkers is as follows:

$$\text{First week } \frac{44.38 \times 16.53}{55.61 \times .76} = 17.4.$$

$$\frac{57.59 \times 16.53}{42.41 \times .76} = 29.6.$$

Since the observed data are incomplete these results cannot be confirmed.

Of the 3,100 lbs. of carbon put into the producer in the first week only 82.6 per cent was converted

*From the American Gas Light Journal.

†Detroit Gas Light Co., Detroit, Mich.

into gas, therefore, the pounds of carbon burned equal $3,100 \times 82.6 = 2,560$.

Cubic feet of gas produced $= 347,500 \times 82.6 = 287,000$.

Total B. T. U. in gas $= 125.5 \times 287,000 = 36,000,000$.

Total B. T. U. in 4,076 pounds coal $= 4,076 \times 11,825 = 48,190,000$.

Per cent efficiency of producer $=$

$$\frac{36,000,000}{48,190,000} = 74.5$$

Second week: Total coal used $= 4,962$ lbs.

Total carbon in 4,962 lbs. coal $= 4,962 \times .76 = 3,775$.

Pounds of available carbon converted into gas $= 3,775 \times .703 = 2,652$.

Cubic feet of gas generated $=$

$$\frac{2,652 \times 453.6 \times 22.38}{12 \times 28.3 \times .2625} = 300,000$$

Efficiency of producer $=$

$$\frac{300,000 \times 125.5}{4,962 \times 11,825} = 64.2 \text{ per cent.}$$

Average results for the 2 weeks $=$

$$\frac{74.5 + 64.2}{2} = 69.4 \text{ per cent.}$$

2

THE CUBIC FEET OF GAS GENERATED, CALCULATED FROM THE DENSITIES OF THE GASES.

Weight of 1 liter in grams of carbon dioxide¹ $=$

¹See Dowson and Sarter, page 269.
1.9650.

Weight of 1 liter in grams of carbon monoxide $= 1.2505$.

From the analysis of the gas we find 6.66 per cent of CO₂ and 19.59 per cent of CO, therefore, the weight of CO₂ and CO present in 1 liter of gas equals:

$$\text{CO}_2 = 1.9650 \times 6.66 = .131 \text{ gram.}$$

$$\text{CO} = 1.2505 \times 19.59 = .245 \text{ gram.}$$

Reducing CO₂ and CO to grams of carbon, we get

$$\frac{.131}{13} \times 12 = .03575 \text{ grams carbon}$$

$$\frac{.245}{28} \times 12 = .105 \text{ grams carbon}$$

Total grains of carbon in liter of gas $= .1407$.

Pounds of carbon converted into gas in first week's test $= 2,560$; therefore, cubic feet of gas produced $=$

$$\frac{2,560 \times 453.6}{1.4074 \times 28.3} = 292,000$$

Per cent efficiency of producer $=$

$$\frac{292,000 \times 125.5}{4,076 \times 11,825} = 75$$

Pounds of carbon available in second week's test $= 4,962 \times .76 = 3,775$.

Per cent carbon lost in producer $= 29.65$.

Therefore, pounds carbon converted into gas $= 3,767 \times .7035 = 2,648$.

Cubic feet of gas produced $=$

$$\frac{2,648 \times 453.6}{1.4074 \times 28.3} = 300,100$$

Efficiency of producer $=$

$$\frac{300,100 \times 125.5}{4,962 \times 11,825} = 64 \text{ per cent.}$$

Pounds carbon available in two weeks' run $= 9,038 \times .76 = 6,869$.

Per cent carbon lost in producer $= 23.5$.

Pounds of carbon converted into gas $= 6,869 \times .765 = 5,250$.

Cubic feet of gas produced $=$

$$\frac{5,250 \times 453.6}{1.4074 \times 28.3} = 598,000$$

Efficiency of producer $=$

$$\frac{578,500 \times 125.5}{9,038 \times 11,825} = 67.7 \text{ per cent.}$$

These results are only approximately correct, and are given mainly for the purpose of showing methods of calculation. Calculation of cubic feet of gas used by engine during test from heat absorbed in scrubber water and heat given up by the gas in passing through the scrubber was made as follows: The cubic feet of water passing through the scrubber rise in temperature of the ingoing water and the fall in temperature of the gas passing through the scrubber were taken for each day. From this data we can calculate with a fair degree of accuracy the cubic feet of gas passing the scrubber.

B. T. U. given up by the gas for second week:

$$\begin{array}{ll} 2/22 = 353,270 & 2/25 = 451,640 \\ 2/23 = 346,000 & 2/26 = 562,620 \\ 2/24 = 474,120 & 2/27 = 426,100 \end{array}$$

Fall in temperature of gas multiplied by the specific heat of the gas:

$$\begin{array}{ll} 2/22 (514 - 55.25) \times .019 = 8.71 & \\ 2/23 (615 - 60.5) \times .019 = 10.50 & \\ 2/24 (630 - 58) \times .019 = 10.82 & \\ 2/25 (651 - 58.2) \times .019 = 11.25 & \\ 2/26 (679 - 58.3) \times .019 = 11.80 & \\ 2/27 (692 - 54.8) \times .019 = 12.1 & \end{array}$$

Cubic feet of gas delivered each day:

$$\begin{array}{ll} 353,270 & \\ 2/22 = \frac{353,270}{8.71} = 40,600 & \\ 346,000 & \\ 2/23 = \frac{346,000}{10.50} = 32,970 & \\ 474,120 & \\ 2/24 = \frac{474,120}{10.82} = 43,800 & \end{array}$$

$$\begin{array}{r}
 451,640 \\
 2 \times 25 = \frac{11.25}{562,620} = 40,200. \\
 2/26 = \frac{11.80}{426,100} = 47,600. \\
 2 \times 27 = \frac{12.1}{35,200.}
 \end{array}$$

$$\text{Total} = 240,370.$$

The equation expressing the above is:

$$\text{Cubic feet of gas} \times .019 (\text{specific heat}) \times (T_1 - T_2) = \text{B. T. U. given up.}$$

$$\text{Cubic feet} = \frac{\text{B. T. U.}}{(T_1 - T_2) \times .019}$$

Where $(T_1 - T_2)$ represents fall in temperature of producer gas passing through the scrubber.

$$\text{Efficiency of producer} = \frac{240,370 \times 133.4}{4,382 \times 11,825} = 62.$$

4,962 lbs. is the amount of coal used for the week, but at the end of the test approximately 580 lbs. of unburnt coal remained in the producer. This, subtracted from 4,962 gives 4,382, the figure used above.

Calculations of cubic feet of gas used in the engine from revolutions:

	Counter at Start.	Counter at Finish.	Difference in Cycles.
2/22	833,593	907,970	74,377
2/23	911,030	982,556	71,426
2/24	986,625	1,059,274	72,649
2/25	60,844	125,710	64,866
2/26	130,780	202,183	71,403
2/27	206,002	258,243	52,241
Total			406,962

To get total intake strokes multiply by $2 \times 406,962 = 813,924$.

Area of piston = 95.033 sq. ins. ; $95.033 \times 12 = 1,140.4 \text{ cu. ins. displacement per stroke} = .6635 \text{ cu. ft.}$

Pressure in cylinders at beginning of compression stroke = 12.7 lbs., absolute. Temperature of gases in cylinder estimated at 77° F.

To correct volume—

$$P_1 V_1 = P_2 V_2$$

$$\frac{T_1}{T_2}$$

$$P_1 V_1 T_2$$

$$V_2 = \frac{P_1 V_1 T_2}{T_1 P_2}$$

$$\frac{T_1}{T_2} \times \frac{P_2}{P_1}$$

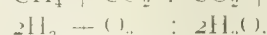
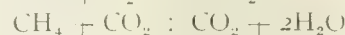
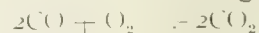
$$12.7 \times 1 \times 522$$

$$= 84.$$

$$537 \times 14.7$$

Total displacement = $813,924 \times .6635 = 540,000$; 540,000 corrected to volume $V_2 = 540,000 \times .84 = 453,000$; 453,000 cu. ft. represents gas and air displaced in the cylinders of the engine. If now we find the ratio of gas to air, and multiply 453,000 by this quantity, we get cubic feet of producer gas used by the engine.

The amount of air required for combustion of the producer is based on the following:



For every volume of CO present in the gas $\frac{1}{2}$ volume of oxygen is required for combustion. Since air

contains $\frac{21}{100}$ of oxygen it will require

$$\frac{1}{2} \times 100/21 \text{ of air for every volume of CO}$$

$$2 \times 100/21 \text{ of air for every volume of CH}_4$$

$$\frac{1}{2} \times 100/21 \text{ of air for every volume of H}_2.$$

The average analysis of the gas for the second week shows: 23.90 per cent of carbon monoxide; .73 per cent of methane; 12.58 per cent of hydrogen.

Volume of air required for combustion of each 100 CO —

$$23.90 \times \frac{1}{2} \times \frac{100}{21} = 56.9.$$

$$.73 \times 2 \times \frac{100}{21} = 6.95.$$

$$12.58 \times \frac{1}{2} \times \frac{100}{21} = 29.9.$$

$$\text{Total} = 93.75.$$

Amount of gas in 453,000 cu. ft. of a mixture of gas and air equals

$$453,000 \times 100 = 234,100 \text{ cu. ft.}$$

$$193.75$$

Efficiency of producer =

$$\frac{234,100 \times 133.4}{43,82 \times 11,825} = 60.5 \text{ per cent.}$$

$$43,82 \times 11,825$$

The last two methods of obtaining the cubic feet of gas used by engine agree fairly well, and the mean of the other two results can be taken by the producer to the engine. After all, even if we weighed the amount of unburned coal and ash remaining in the producer at the time of test, the last two methods would have to be relied upon, because we have no way of telling how much carbon is used up by the blower in starting the test each morning. This is no inconsiderable amount, as shown by the log sheets.

The total time of run was 45 hours, and $13\frac{1}{2}$ hours were spent in getting the fire up with the blower. Ex-

$$\text{pressed in per cent of time, it equals } \frac{13\frac{1}{2}}{58.5} = 23.1.$$

Consul Albert Halstead states that the electroplating industry has been noticeably growing in Birmingham, England, many establishments using electric power from the city. In the important Birmingham jewelry trade almost all the power needed is obtained in this way.



IMPROVEMENTS IN THE KNORR FAT EXTRACTION APPARATUS.*

Modified flasks and tubes in connection with the Knorr fat extraction apparatus have been used for some time in the laboratory of the Bureau of Chemistry of the U. S. Department of Agriculture, and are stated to have proved more satisfactory than the regular forms. The following description of these improvements is given by H. L. Walter and C. E. Goodrich, Assistant Chemists, Miscellaneous Division, Bureau of Chemistry, in a recently issued circular of the Bureau.

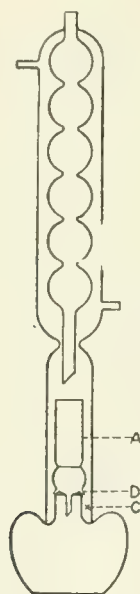


Fig. 1—Old Form of Knorr Extraction Apparatus.

The common form of Knorr extraction apparatus presents a few difficulties in manipulation, some of which the authors have effectively overcome. These difficulties are:

(1) In many cases the ether which condenses in the lower part of the condenser cannot return freely to the flask because the extraction tube fits too snugly in the mouth of the flask. In this case all the ether soon collects in the space *C*, Fig. 1, and the flask becomes dry.

(2) There is a tendency for many of the samples, together with layers of the asbestos felt, to push upward in the tube by the expansion of air and ether vapor which they inclose.

(3) When a sample is very fluffy, as alfalfas, for example, the ether runs through so rapidly that it fails

to reach all parts of the sample, and consequently gives a poor extraction.

(4) The form of extraction tube ordinarily employed has several disadvantages which will be discussed later.

The first of these difficulties was overcome as follows: A glance at Fig. 1 will show that the escape of ether vapor from the flask of the Knorr extraction apparatus is accomplished by supporting the tube *A* upon the flask by means of three glass projections *D* on the lower part of the tube, which raise it somewhat above the rim of the flask. This is easily accomplished, even though only a small space is left between the flask and the tube, but those who have used the tube to any extent know that considerable ether condenses in the lower part of the condenser and soon fills the space *C*

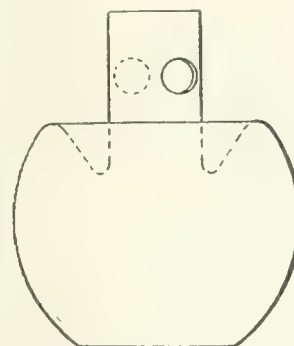


Fig. 2—Modified Form of Flask.

above the mercury seal. After enough has condensed to reach the top of the flask, the ether begins to flow back into it. To take care of this return flow is very difficult, and in many cases impossible, without using some other means to support the extraction tube.

Two holes in the neck of the flask effectually overcame the trouble, besides making possible other valuable improvements in the apparatus. Fig. 2 shows the flask with the holes. It will be noticed that the amount of ether collected in the space *C* when the flask with this modification is used is much less, making a saving of about 10 cc. in each extraction.

The holes are about $\frac{1}{4}$ in. in diameter and are placed opposite each other at a distance approximately $\frac{1}{4}$ in. above the upper mercury line or above the line of the trough.

The second and third difficulties were overcome by means of a simple spring shown in Figs. 3 and 4, a description of the spring and its uses being given in the following paragraphs.

*To insure the free use of this invention to the public, letters patent have been applied for.

It is well known to all who have used the Knorr ether extraction apparatus that the sample and sometimes layers of asbestos are pushed up toward the top of the tube by the expansion of the air and ether vapor which they inclose. Not only is there danger of losing in this way the sample for the crude fiber determination, but there is a chance of foreign sub-

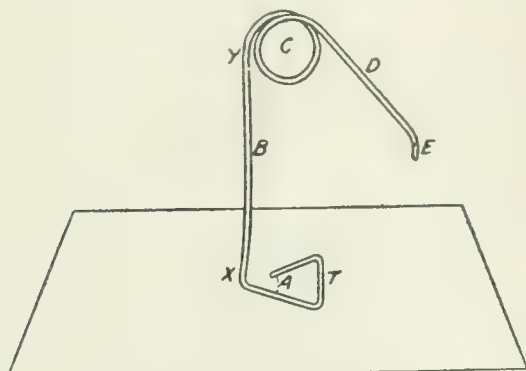


Fig. 3—Perspective Drawing of Spring.

stances reaching the interior of the fat flask, and the cushion formed of air, ether vapor, or both, materially checks the ether and extract on their way to the flask. This, as one can readily see, prevents thorough extraction, even though the sample be not pushed over the top of the extraction tube.

The remedy hitherto employed consists in striking gently the portion of the condenser surrounding the tube, at the same time holding the flask to prevent

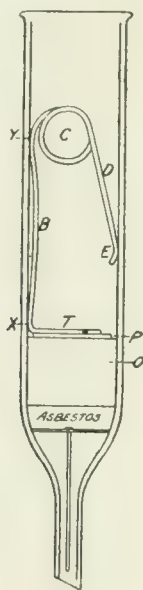


Fig. 4—Spring in Position in Tube.

breaking. As a rule this serves to break up the cushion of air, but it is often necessary to remove the tube from the condenser and push down the sample with a stirring rod. Every time a tube is taken from the condenser there is danger of mercury getting into the fat flask, despite the best precautions. Even after the sample has been restored to its proper place in the tube by either of the foregoing methods, the cushion of

vapor is likely to form again, requiring a repetition of the procedure just described.

A simple spring was, therefore, designed to avoid this unnecessary care and loss of time. It is made of No. 19 spring brass wire, and is of the form shown in the perspective drawing, Fig. 3. The parts consist of the upright *B*, at the lower end of which is formed the open equilateral triangle *T*. At the top of *B* the wire is bent into a single coil *C* and extended in the portion *D*, the end of which is bent at *E* to facilitate inserting it in the tube. A slight double curve should be given to *B*, as illustrated in the drawings, so that the points *X* and *Y* shall be in contact with the sides of the tube thereby holding the apparatus in a vertical position.

The plane of the triangle *T* should be perpendicular

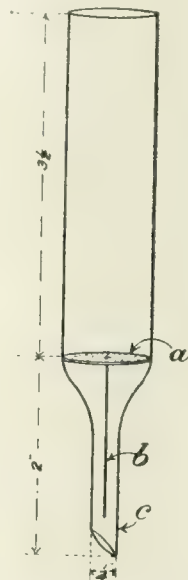


Fig. 5—New Form of Extraction Tube.

a, perforated disk; *b*, wire riveted to disk; *c*, size $\frac{1}{4}$ in.

to *B*. The length of its sides will vary according to the inside diameter of the extraction tube and should be such as will allow a slight amount of play. The plane in which *B* and *D* lie must bisect the angle *A* to insure an upright position of the apparatus in the tube.

To use this apparatus proceed as follows:

Drop a perforated platinum (or nickel) disk on top of the sample so that it will lie in a horizontal position. Press *B* and *D* between the thumb and forefinger and push the spring down into the tube until *T* presses firmly against the disk; in fact it does no harm to compress the sample slightly. The complete combination is shown in Fig. 4, *O* being the sample and *P* the disk. *D* presses firmly against the side of the tube, thereby holding the apparatus securely in place at any height.

B should have a length of about 50 mm., and *D* 40 mm., the loop *C* being 15 mm. in diameter. It is not necessary to use exactly these dimensions, but care should be taken to have *C* below the top of the extraction tube, otherwise the ether in falling from

the condenser may strike the wire and be scattered over the side of the tube.

With flasks modified as described it is possible to use the form of tube shown in Fig. 5. The advantages of this tube over the old form are many. (1) The initial cost is much less, since for most solvents used the disks and wire may be of nickel instead of platinum. A few platinum disks may be kept on hand and used when the nature of the solvent requires it. (2) The sample after being extracted with ether can be much more readily transferred to the crude fiber flask by merely using a glass tube small enough to enter the constricted portion of the extraction tube easily, and pushing out the disk, asbestos mat, and sample. The wire attached to the disk should enter the small glass tube. In most cases this operation takes the sample out clean, but it is very easy to wash out any particles that may adhere to the sides. (3) Many tubes of the old form were defective and not able to stand the vacuum pressure, the disk being drawn into the bulb. Another source of loss was the breakage caused by heating too dry the asbestos felt, a large percentage of tubes developing cracks from the strains due to the projections *D* (Fig. 1). The tubes here described, when broken, can be replaced for a few cents and the disks will last for years.

The price of the old tubes is about \$1.30 each, while the new form, with the disk, will cost probably about 10 cts. These modified flasks and tubes have been used in this laboratory for some time and have proved much more satisfactory than the regular form.

A recent report of the Indian Foreign Department furnishes interesting data in regard to India's ability to meet the world's demand for wood pulp. Among the many materials referred to in the report the most important is bamboo, which, it is claimed, if properly taken in hand, may be expected to become the leading material, as the pulp made therefrom resembles closely that made from American and European spruce and fir. In those districts where the bamboo grows with most luxuriance (Bengal, Assam, Burma, etc.) the supply may be described as almost inexhaustible, for it reproduces itself naturally; and by a judicious system of cutting a mill located in a suitable district could depend upon a perpetual supply from the surrounding area. Paper mills in India have so far not been largely remunerative, owing to the fact that they have to obtain the raw material from long distances and the cost of the freight eats up the profits; but by the establishment of wood-pulp factories in this country the freight item could be eliminated and India would be in position not only to supply its own requirements in cheap paper now purchased abroad (the imports of paper and pasteboard in the fiscal year 1909 amounted to \$3,007,500 and in the year 1910 to \$3,284,900), but could also export wood pulp.

THE ACCURACY OBTAINABLE IN FUEL CALORIMETRY.*

By G. NEVILL HUNTLY.

The economic importance of the determination of the calorific value of solid and liquid fuels is receiving increasing recognition among fuel users, and this is especially the case in the United States and on the Continent. The widely discrepant results obtained in the past in different laboratories for the same sample of coal led many engineers to regard laboratory tests as useless. These discrepancies have been traced to two causes, the imperfections of the calorimeters used and the errors of sampling. Comparisons of the various types of calorimeter¹ have shown that the bomb calorimeter gives more consistent results than any other form, and the Berthelot calorimeter is now by general agreement looked upon as the standard instrument. The operation of sampling is one over which the analyst has usually no control, but if carried out with due care the limits of error can be shown to be of the same order as those of the calorimetric measurements. Consecutive determinations with a bomb calorimeter give such concordant results that there is a natural tendency to exaggerate the accuracy obtainable. Numerous bulletins of great value have been issued by the U. S. Geological Survey and by the University of Illinois embodying the results of a systematic study of various coal fields in the United States, and in all of these the calorific value is given to five significant figures (e. g., 14,949 B. T. U. per lb.) These arithmetical exercises appear to be taken quite seriously by engineers, as for example in Bulletin No. 31, University of Illinois, where the corresponding boiler efficiencies are worked out to the second place of decimals, as 58.18 per cent. Similar examples could be easily quoted from our own engineering journals. It will be shown in this paper, that apart from errors of sampling, an experimental error of 30 B. T. U. per lb. represents the limit of the most careful work, and for estimations carried out under commercial conditions errors of 150 B. T. U. per lb. may be regarded as probable.

The possible errors may be classified in three groups concerned with (1) the thermometer; (2) the correction for cooling; (3) the determination of the water equivalent.

THE THERMOMETER.

There seems to be no work in English in which the difficulties of accurate thermometry are adequately dealt with. The only exhaustive book on the subject is that of Guillaume, "Traite pratique de la thermometrie de precision," Paris, 1889, which, although written 21 years ago, still remains the standard work on the subject.

The electrical resistance thermometer under favor-

*From the Journal Society Chemical Industry

¹See Gray & Robertson, Journal Society Chemical Industry, 1904, p. 704, and Brame & Cowan, loc. cit., 1902, p. 1230.

able conditions undoubtedly gives a higher accuracy in temperature measurements than the mercury thermometer, but in practice, in spite of its defects, it is the latter which is generally used. Apart from the cost of equipment, the question of vibration militates seriously against delicate electrical measurements especially in chemical laboratories in large towns or in a power station. With the mercury thermometer it seems to be generally assumed that the longer the degree and the larger the number of divisions into which it is divided, the more accurate will be the readings, and hence an instrument of the Beckmann type, with a range of about 6° C. divided into hundredths of a degree, and possessing a degree length of about 30 mm., is a favorite one. Apart from the disadvantages of the German type of thermometer² it was pointed out by Guillaume that no real advantage is gained by increasing the length of the degree above 10 mm., as the capillarity errors then become more important than the calibration errors, and this is especially the case in calorimetric thermometers, in which the capillarity effect is doubled if the thermometer is rising in the preliminary period and falling in the after period. The type of thermometer adopted by the International Bureau at Sevres and constructed by Tonnelot, is made of hard French glass with a degree length of about 10 mm., and is divided on the stem into tenths of a degree. The stem is transparent so that the readings are not obscured by the thickness of the lines, an important advantage. From a thermal point of view (temporary depression of the zero) the French hard glass is only slightly inferior to Jena glass and it possesses the great advantage that fine sharp lines can be etched on it, down to 0.02 mm. wide. The author has not been able to get thermometers of Jena glass with divisions narrower than 0.06 mm., and these are somewhat ragged in outline when magnified in the telescope. When suitably illuminated and read through a telescope the division is 0.1 and be read by estimation to one-hundredths, or 0.01 (see Guillaume on this point).

An eye-piece micrometer may be used for these readings, but it will be found that after some practice the micrometer may be dispensed with. A simple way of testing the accuracy of estimation is to prolong the observations of the cooling period after a combustion and plot the results on an open scale; the deviations of the readings from the straight line will show the errors of estimation.

Since, in spite of its defects, the short range Beckmann thermometer is very generally used, it may be worth while to point out its most obvious faults and the means of eliminating them. Tubes with a flat elliptical bore should be rejected, as the capillarity errors are very irregular and the thread descends in jerks even though the stem be constantly tapped. The lines should be as fine as possible, as with, say three lines to the millimeter, a line 0.1 mm. wide represents

an error of 0.003° when the thread is over the line.

Instruments by different makers vary very much in the thickness of the divisions. In some thermometers of English make divided into fiftieths, the lines are so coarse that readings closer than 0.01° cannot be made. A more serious error, and one which sometimes appears to have been quite overlooked, is due to the fact that the value of the degree is variable, and depends on the temperature at which the instrument is used. A rough correction table is given in the Reichenstalt certificates, but this only applied when the thermometer is immersed to the zero line, a condition not generally possible.

Moreover, it is obvious that the value of the degree must be determined by comparison with a standard, divided in tenths, and if, as seems to be commonly assumed, the latter can be read only to hundredths, the possible error of comparison over at 6° range is 1 in 300, and as a matter of fact it may be larger than this, owing to the impossibility of checking any zero change on the Beckmann thermometer. (For another method of dealing with this, see determination of the value of the degree.)

In the determination of a temperature the thermometer reading requires to be corrected for: (1) Calibration. (2) External pressure. (3) Internal pressure. (4) Zero. (5) Correction for the fundamental interval (value of the degree). (6) Exposure correction. (7) Reduction to the normal hydrogen scale.

Calibration.—The effect of this correction is to compensate for the inequalities of the bore of the tube, and to reduce the readings to those of a truly cylindrical tube of the same mean cross-section. If the maker has endeavored to allow for the inequalities of the bore in his graduations, an accurate calibration is difficult, if not impossible; the divisions should be equidistant and if the calibration is to be carried out to an accuracy of 1/100 of a division, the equidistance of the divisions must be of the same order of accuracy and the lines as narrow as possible. If, as on a 6° Beckmann thermometer, an accuracy of 1/10 of a division is sufficient, the mean of half-a-dozen Gay Lussac curves with threads of different lengths will be good enough; if an accuracy of 0.01 division is aimed at the operation is more delicate, the Neumann-Thiesen³ method requiring the least expenditure of time. The labor spent in the calibration is well repaid, since a comparison of the observed and calculated thread lengths clearly defines the accuracy of estimation of the observer. In this country there is some difficulty in getting a well-calibrated thermometer; the National Physical Laboratory does not undertake the calibration of thermometers. The International Bureau at Paris carries out such calibrations with the highest precision, but it would appear that the staff has been reduced and there is some difficulty in getting such work done now. The ordinary Reichenstalt

²See *Methodes de Calorimetrie*, W. Lougimire & A. Schukarew, p. 3.

³See Guillaume and the Report of the B. A. Committee (1882) for details.

certificate, such as is obtained if a thermometer is bought with a certificate, is accurate to 0.1 division only, but thermometers satisfying certain conditions can be calibrated with a higher accuracy. The B. A. committee considered 0.002° (0.02 division) as the limit of possible accuracy; Guillaume regards an accuracy of 0.001 (0.01 division) as attainable provided certain conditions are observed by the maker. In this connection it should be noted that the thermometers studied by the B. A. committee were not etched with equidistant lines.

External Pressure.—Since the bulb is fixed during the experiment and the time of an experiment is short, the effect of external pressure on the temperature difference is negligible.

Internal Pressure.—If a thermometer shows a difference of n true degrees in a horizontal position, it reads $n(1 - B)$ degrees in a vertical position, where B is the coefficient of internal pressure and 1 the length of the degree. For calorimetric work this may be conveniently included in the value of the degree.

Zero.—For a rise of 3° the temporary depression of the zero is of the order of 0.003° for French glass, rather less for Jena glass. The action, however, requires time, and would be scarcely appreciable in the five minutes of an experiment; even if appreciable, the error cannot be exactly determined.

Correction for the Fundamental Interval.—Except by accident, no thermometer has a degree exactly $1/100$ of the fundamental interval. If the instrument contains the two fundamental points this error can be determined directly. A convenient type is graduated in tenths from -2° to 52° C., then a bulb and further graduations from 98° to 102° . This forms a self-contained standard, but since the transition point of sodium sulphate is now known accurately (32.379°) a range -2° to $+34^\circ$ is equally suitable for calorimetric work. This is first calibrated in principal points 6° apart in the interval (-2° , 34°) converted into the fundamental interval (0 , 32.38°) and, to save labor, only such sections further subdivided into 1° intervals as may be required by the temperature range of the laboratory. For short range thermometer with variable zero, the only really practical plan seems to be an experimental determination with the bomb itself. For this purpose a considerable quantity of a pure substance is prepared, such as naphthalene or benzoic acid, and every time the quantity of mercury in the bulb is altered the water equivalent is determined in the usual way, keeping the weight of water fixed. The temperature rise is corrected for calibration, radiation, fuse and nitric acid. The mean temperature of the water in the calorimeter is measured with an ordinary thermometer.

In this way, a series of results will be obtained from which the apparent water equivalent can be plotted as a function of the temperature. The following corrections are thus eliminated: Internal pressure, value of the degree, exposure of the stem, variations in the specific heat of water and of the calorimeter, bomb,

etc., and reduction to the hydrogen scale if the calorific value taken for the naphthalene or benzoic acid has itself been expressed in that scale. It is worthy of note that the errors of calibration of the tube are not corrected in this way.

Exposure Correction.—This correction is an uncertain one at the best. If the water equivalent is determined by the combustion of substances the heat of combustion of which is taken as known, as above, it is eliminated. It is not more than one or two-thousandths of a degree, and is usually neglected in calorimetric work.

Reduction to the Normal Hydrogen Scale.—This correction just comes within the limits of error. A rise of 3° ($15^\circ - 18^\circ$) measured on the scale of Jena glass 16 III is 3.010° on the normal scale.

These considerations lead to the conclusion that for a temperature rise of 2° to 3° with a mercury thermometer of the first order completely studied before use, and after all possible corrections are applied, an accuracy of 0.1 per cent to 0.2 per cent cannot be exceeded. With a thermometer coarsely graduated and uncalibrated the error might amount to over 1 per cent.

The concordance of two or three consecutive determinations is quite misleading as a criterion of the real accuracy. As illustrating this point five determinations of the water equivalent of the same calorimeter (by combustion of naphthalene) made at intervals of about two months with two thermometers divided into tenths, at temperatures ranging from 8° to 20° , after fully correcting, gave for the water equivalent at 15° C.: (1) 2,504; (2) 2,497; (3) 2,495; (4) 2,505; (5) 2,500. Nos. 1 to 3 were made with one thermometer, Nos. 4 and 5 with another. Two consecutive observations with a Beckmann 6° thermometer gave 2,516, 2,517, for a temperature of 15° C. The two last determinations, in spite of their close agreement, are 0.7 per cent too high, owing to the changed value of the degree.

THE CORRECTION FOR RADIATION.

This correction is second in importance only to the thermometer corrections. With a properly designed bomb calorimeter it is of the order of 1 per cent of the temperature rise. It is generally recognized that the so-called Regnault-Pfaundler correction gives the nearest approximation to the actual loss, but this seems to be looked upon as too complicated for technical work, and either the correction is dropped altogether (Hempel, Jakob), or an approximate empirical correction is applied (Mahler, Gramberg, Langbein, Gray, and Robertson, etc.). If the observations are entered on a suitably printed or graphed log, the calculation of Regnault-Pfaundler correction, assuming a slide rule is used, is a matter of five or six minutes at most, and it is a little difficult to understand why 30 to 40 minutes should be regarded as profitably spent in determining the amounts of nitric acid and sulphuric acid formed in the combustion, the combined

effect of which rarely exceeds 0.5 per cent of the heating value, whilst a quarter of the time is regarded as too much for the more important cooling correction. The meaning of the Regnault-Pfaundler formula does not seem to be well understood, since several authors remark that instead of using this complicated expression the simple application of Newton's law of cooling is sufficient. But all expressions for the loss by radiation, unless purely arbitrary, are based on Newton's law of cooling, and the Regnault-Pfaundler formula is no exception. The latter has the advantage of including the cooling effect due to evaporation, the heating effect of the stirrer, and any other constant source of heating or cooling, such as a cooling effect due to a slight leak of the bomb.

All cooling corrections are based on the assumption that the temperature of the surroundings remains constant during the experiment. This fundamental condition is satisfied by surrounding the calorimetric vessel with a double copper jacket containing at least 20 liters of water. It is not satisfied by using a wooden jacket, felt, or similar badly conducting substances. The object to aim at is not so much to prevent heat losses, but to ensure that these losses shall take place in a definite manner.

It seems worth while to show upon what assumption the Regnault-Pfaundler expression is based, so that if a rougher approximation is deemed sufficient it may be derived on a rational instead of on a purely empirical basis.

Let the loss of temperature in time dt be $+d\Theta$.

Loss due to evaporation per minute $= w$.

Loss due to heat developed by stirrer per minute $= s$.

According to Newton's law, the velocity of cooling due to radiation is proportional to the difference of temperature between the cooling body (Θ) and its surrounding (T) or

$$\frac{d\Theta}{dt} = -k(\Theta - T).$$

k is the "cooling constant" of the calorimeter and is evidently the rate of cooling for a 1° difference between Θ and T .

(I.) Hence $d\Theta = (w - s) dt + k(\Theta - T) dt$.

The experimentally determined figures are:

v = velocity of cooling at (or near) Θ_0 , the initial temperature; v' = velocity of cooling at (or near) Θ_n , the final temperature.

It should be noted that Θ_n is not the highest observed temperature, but may be any temperature after the steady rate of cooling has set in. If v' is determined starting from the maximum temperature, it will be too low, as heat is coming out of the bomb after the maximum temperature has been attained.

From (I.) $v = (w - s) + k(\Theta_0 - T)$

$v' = (w - s) + k(\Theta_n - T)$

$$v - v' = k(\Theta_0 - \Theta_n) \quad \text{or} \quad k = \frac{v' - v}{\Theta_n - \Theta_0}$$

Also since $v = (w - s) + k(\Theta_0 - T)$

$$w - s = v - k(\Theta_0 - T).$$

$$(II.) \quad \text{Hence } d\Theta = v dt - k(\Theta - T) dt + k(\Theta - T) dt \\ = v dt + k(\Theta - \Theta_0) dt.$$

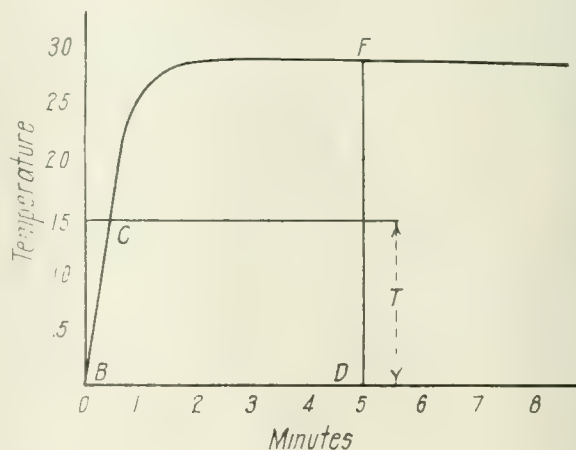
Hence the total correction over the principal period of t intervals of time is:

$$\Sigma \frac{\Theta_n - \Theta_0}{\Theta_0} d\Theta = vt + k(\text{area BCFD}).$$

If it be noted that v and v' are not determined strictly at Θ_0 and Θ_n , but at the closely adjacent temperatures Θ (mean temperature of the fore period), and Θ' (mean temperature of the after period), the final expression is:

$$(III.) \quad \Sigma \frac{\Theta_n - \Theta}{\Theta_n} d\Theta = vt + \frac{(v' - v)}{(\Theta' - \Theta)} (\text{area BCFD}),$$

and this is equivalent to the Regnault-Pfaundler formula, where t is the number of time intervals (minutes or half minutes) in the principal period, v and v' being the temperature losses in the same time interval. Any cooling correction must be of this form, and the accuracy attained will depend on the degree of approximation with which the area BCFD is meas-



ured. For this purpose it is not necessary actually to plot the curve, but it is worth while doing this for a few experiments. Insufficient stirring is then brought out by the points not lying on a smooth curve; the length of the time interval (60, 30, or 20 seconds), can also be best chosen from a study of these curves, and the effect of a given error of area may be determined by the planimeter or by counting the squares or any of the usual formula for approximate integration may be adopted. The latter is more practical since the actual temperature readings are the ordinates required. The "trapezoidal" rule, half the sum of the first and last ordinates + the sum of the remainder, is a first approximation. Taking the temperatures read as ordinates, this gives the area down

to zero temperature, hence $t\theta$ has to be subtracted, giving the usual Regnault-Pfaundler expression,

Correction =

$$vt \left[\frac{v' - v}{\theta' - \theta} - \frac{\theta_n + \theta_1}{2} + \frac{4}{1} \theta - t\theta \right]$$

the terms within the bracket representing the approximate integration of the area BCFD.

A more accurate result is given by the application of Simpson's or Durand's rules. From the following example it is clear that the trapezoidal rule, with one minute intervals, is good to less than 0.001°.

(1) Plainmeter 0.0258

(2) Durand 0.0257

Correction corresponding to area BCFD.

($t = 5$ minutes.)

(1 min. intervals.)

(3) Trapezoidal 0.0253

(1 min. intervals.)

(4) Trapezoidal 0.0256

($\frac{1}{2}$ min. intervals.)

And hence there is no advantage in employing a more exact method of integration. If the stirrer is worked at fixed rate, and the temperature curves are similar, various expressions for the mean ordinate can be found for any given instrument which will be near enough for technical work. Thus if θ_1 is the rise of temperature in the first minute, for a particular calorimeter, the cooling constant of which averages 0.00283, the rough correction is given by $(v + ck\theta_1)t_1$, where $ck = 0.004$.

From seven examples, taken at random, the maximum difference between $t(v + 0.004\theta_1)$ for $t = 5$ minutes and the full Regnault-Pfaundler correction was ± 0.004 , and the average difference 0.002. It should be noted that the term vt includes the cooling due to evaporation.

By cooling the calorimeter water below the temperature of the surroundings the total correction is reduced and as a special case may vanish. Here the term vt is negative, and the final correction is the sum of the negative and positive terms. It is clear that whether the correction $+ 0.030^\circ$ or 0.001° a given error in the determination of either of these terms will cause the same error in the result.

THE DETERMINATION OF THE WATER EQUIVALENT.

This is usually determined in one of two ways, by calculation from the specific heats of the constituents of the bomb, water vessel, stirrer and thermometer, or by burning a known weight of a definite substance whose heat of combustion is taken as known. The method of calculation gives only a rough approximation: the material of the bomb may be bronze or nickel steel of unknown composition, the weight of the enamel is unknown, and the specific heats taken as a basis of the calculation are usually the mean specific heats between 100° and about 15° , and hence are too high. If the water equivalent of the metal parts be taken as 350 to 400, and an uncertainty of 20 be possible, with a Hempel calorimeter (1,000

grams of water), the possible error is 3 per cent, with a larger calorimeter of water equivalent 2,500, 1.6 per cent, and with W. E. 10,000, 0.4 per cent.

In the latter case the rise of temperature is so small that an electrical thermometer⁴ must be used. As regards the second method of determining the water equivalent, a comparison of the figures obtained for the heats of combustion of pure substances shows differences of 0.3 to 0.7 per cent.

HEATS OF COMBUSTION (OF 1 GRM.) ACCORDING TO DIFFERENT OBSERVERS

	Benzene acid	Naphthalene	Cane sugar.
Berthelot and Recoura...	6345	9688	..
Berthelot and Longumme...	6322	9696	..
Berthelot and Vieille	..	9718	3962
Stohmann	6322	9628	3955
Fischer and Wrede (1904)	6355	9668	3988
Fischer and Wrede (1908)	6328	..	3954

When it is remembered that each of these figures represents the mean of a number of observations by highly skilled observers working under the most advantageous conditions, it is clear that the determination of the absolute heating value of a coal nearer than 0.5 to 1.0 per cent is not to be expected.

The third method of determining the water equivalent is by the addition of a known amount of energy measured electrically. A very full account of the calibration of a bomb calorimeter in electrical units, using a platinum thermometer is given by W. Jaeger and H. von Steinwehr⁵, and this calorimeter has been used by Fischer and Wrede for determining the heats of combustion of numerous substances.

These determinations must be regarded as the best hitherto made, as the probable errors are less than 1 in 2,000. The values for sugar (3,954) and benzoic acid (6,325) appear to be the best established.

Other errors affecting the heat of combustion are discussed very fully by W. Jaeger and H. von Steinwehr in the *Zeits. für Physik. Chemie.* 53 (1905), p. 153, and their conclusions may be briefly cited here. They regard an accuracy of 1 or 2 parts per 10,000 as the limit with mercury thermometers, and show that the lag of the bomb need be taken into account only when the water equivalent is determined from the specific heats. The lag of the thermometer is shown to introduce errors less than 1 part in 1,000, and this the author can confirm as out of twelve thermometers the maximum error due to lag has been found to be ± 0.4 parts per 1,000. They also discuss the error due to the variation of the water equivalent with temperature, and conclude that there is a change in the heat capacity of the whole apparatus of 1 part in 1,000 for each 4° change of temperature from 15° . Since the temperature in experiments made in unheated rooms may vary between 6° and 25° , this correction can by no means be neglected. The specific heat of water itself is uncertain to 1 part in 1,000, as will be seen from the following table.

⁴See W. Jaeger and H. von Steinwehr, *Verh. Deut. Physik. Ges.*, 1902, 353.

⁵*Ann. d. Physik.*, 21 (1906), p. 23.

THERMAL UNIT OF WATER

	Reynolds and Griffiths	Bartoli and Straccanti	Callendar and Barnes
1			
0°	1.0036	1.0069	1.0061
5°	1.0019	1.0016	1.0010
10°	1.0000	1.0000	1.0000
15°	0.9988	0.9993	0.9994
20°	0.9971	0.9998	0.9996
25°	0.9942	1.0015	1.0015
30°	0.9924		0.9971

Hence in the definition of the thermal unit not only must the degree be on the hydrogen scale, but the temperature of the water must also be defined.

The details of construction of the bomb do not much affect the accuracy of the results obtained by its use, provided it is of sufficient capacity. Bombs by some makers have the bottom of the bomb too near the base of the calorimeter vessel; this results in a pocket of water which remains out of reach of the stirrer, and consequently gets warmer than the rest, thus making the results low. The same defect is sometimes caused by an unsuitable framework to support the bomb in the calorimetric vessel. The stirring arrangements are defective in others. It is tacitly assumed in calculating the correction for cooling that the external surface of the water vessel has the temperature indicated by the thermometer, and this is not the case unless the stirrer is efficient. The stirrer should be driven by a small motor, as this secures both efficiency and regularity. As illustrating the effect of efficient stirring and a high water equivalent, the author has found that with a Hempel calorimeter used as supplied the average error of a determination, using naphthalene, was 0.5 per cent; with the same bomb and thermometer, but using a larger water vessel, with mechanical stirring, a large copper water jacket, and also raising the bomb an inch above the base of the water vessel, the average error of a determination was reduced to about 0.2 per cent. The Kroeker form, with two valves and fitted with a wash bottle arrangement of platinum tubes, is very convenient, as it permits of an accurate estimation of the carbon in the fuel at the conclusion of the experiment; the estimation of hydrogen in the same instrument is not so good. The question of the lining, platinum or enamel, is of importance only as regards the life of the bomb; it is without effect on the accuracy of the results. In either case it is better to avoid the use of iron wire for the ignition, as the particles of fused iron oxide rapidly perforate the lining. Fine platinum wire (.002 in.) with or without cotton thread or collodion wool, is effective. If too large a wire is used the amount of heat developed by its fusion ceases to be negligible.

So far, the combustion of pure substance only has been dealt with. With coal or oil there are some additional points to be considered. For a representative sample the coal should be as finely divided as possible, and with very finely divided coal the combustion would appear to be incomplete.

This difficulty is got rid of by adopting the suggestion of Hempel and compressing the coal into small

briquettes. The author has found that the combustion is also occasionally incomplete for another reason. The ash from certain coals is fusible, and if the amount of ash is high, the fused ash is liable to contain unburnt carbon. The maximum error due to this cause which has been observed corresponded to 3 per cent of the calorific value. The carbon was estimated by powdering the ash and burning in a closed tube with lead chromate. As the removal of the fused ash from the platinum capsule was a matter of some difficulty, a little powdered quartz was placed in the capsule. This was found, quite unexpectedly, to reduce the amount of unburnt carbon, possibly by reducing the average size of the fused globules. In nonfusible ashes no unburnt carbon has been detected.

Sampling.—The problem of taking small samples accurately representing large bulks of solid material, has been fairly solved by metallurgists, but the necessity of applying similar methods to coal has not always been appreciated by the engineer. Bulletin No. 323 of the U. S. Geological Survey gives a description of their method of sampling a carload of coal. From 3 per cent to 4 per cent of the carload was taken as a sample, the whole crushed to $\frac{1}{4}$ in. size, and worked down mechanically. Two independent samples gave moistures agreeing within 0.4 per cent and ash within 0.3 per cent. Hand sampling usually has to be resorted to, and the results are not quite so good. For air-dried coals low in ash, the error of sampling should be under 0.5 per cent. Coals with high ash (7 to 12 per cent) may be 1 per cent out, and wet small coal, if not air-dried on a fairly large scale before taking the final sample, may differ by 2 per cent or more on consecutive samples. A sampling error of 0.5 per cent to 1.0 per cent may be looked upon as a fair average. This accuracy can only be obtained when the sampling is done systematically under skilled supervision. Even oil fuel has its own sampling difficulties, owing to the frequent presence of water, an emulsion being formed which is very difficult to separate.

The conclusions arrived at may be summarized as follows:

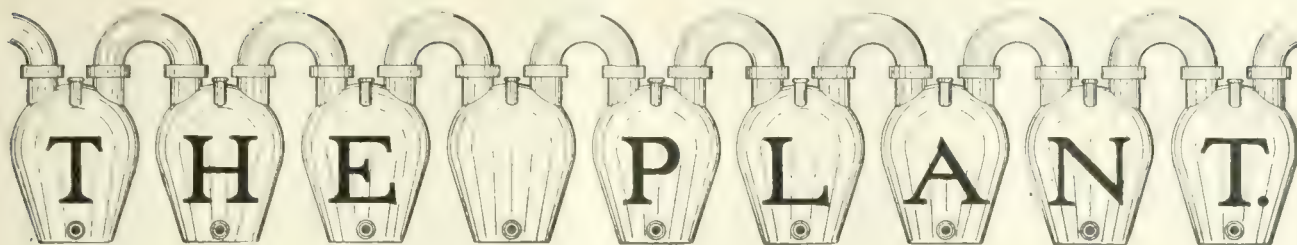
(1) The limit of accuracy in the comparison of the heating values of fuels is from 0.1 per cent to 0.3 per cent, and this can only be reached after a most careful study of the thermometer and calorimeter and the application of numerous corrections.

(2) If the small corrections are neglected the errors may amount to from 1 per cent to 2 per cent.

(3) The heats of combustion of the standard substances used for the determination of the water equivalent are not certain within 0.5 per cent.

(4) The errors due to hand sampling cannot be reduced with certainty below 0.5 per cent, and may easily amount to 1.5 per cent to 2.0 per cent.

Hence, so far as coal is concerned, it is at present more important to devise means for reducing the errors of sampling than to increase the accuracy of the calorimetric determination.



THE RELIABILITY OF ELECTRIC FURNACES FOR CHEMICAL WORK.*

By FRED T. SNYDER.

A few days ago a man who had been East attending the meetings of an association of engineers representing an industry that uses a large amount of high-grade steel, in various forms, was asked about the electric furnace paper on the program. He characterized it as "the usual optimistic hopes by the agent of an electric furnace inventor." While without the furnace inventor, and without the optimistic hopes that held them for ten years to the expenditure of time and energy and money that was all outgo and results deferred, there would be no commercial electric furnace work today to consider, it now seems as though the time had come to point out to those who are less closely in touch with developments than ourselves that the electric furnace has grown beyond a question of this inventor or that inventor, has grown beyond a question of hopes to one of commercial use, and is now a matter of engineering and finance; today the electric furnace has achieved reliability.

But before the time goes by when particular arrangements of electric furnaces are called after their inventors, as such time has gone by for the open-hearth furnace and for the crucible furnace, it would seem proper that we should call by the name of its inventor that simplest of all electric steel furnaces, in which a single electrode above leads the current through an arc to the steel, and a metallic contact with the steel bath below leads the current back to the dynamo. Back in 1880, so far back that the paper describing it was read before the "Society of Telegraph Engineers," as then the only representatives of electrical engineering, William Siemens built and operated his furnace, in which the heat of the electric arc was used to melt steel. This little furnace was a practical furnace. It made good steel. A duplicate of it, copied from the old drawings, will today work without difficulty, making its little batch of steel. It is especially notable that it is to this same William Siemens that the steel industry also owes its open-hearth furnace, second only in importance to the blast furnace itself. As the Siemens electric steel furnace was the first practical electric steel furnace, so now it bids fair to be also the furnace to be used in the greatest numbers. It is characteristic of the electric furnace that it is reliable in small units, and it is in this direction

that the future is tending, towards the making of some steel castings by many of the grey-iron foundries, and the making of their own steel castings by many of the larger users. In these small installations, the simplicity and reliability of the Siemens electric steel furnace will be important. With proper engineering, this single-electrode type of steel furnace can be built reliably up to 500 kilowatts, a production of 250 tons of steel per month with continuous operation. In point of numbers, many more than a majority of commercial installations will be below this tonnage. A worthy use of this opportunity will be to emphasize the good to electrochemistry that has lived beyond the life of this man, by calling this type of electric furnace the Siemens furnace.

Broadly, reliability is commercial, the ability to make a profitable return on an investment. That the electric steel furnace is reliable in this way is emphasized by the growth of those plants in Europe that are making electric steel exclusively. And beyond steel making, the electric furnace has been commercially reliable so long in the aluminum industry that the general patents have run out. The commercial reliability of the electric furnace in the manufacture of carbide has for years been in evidence with each user of acetylene light. Less widely known, but commercially profitable, has been the electric furnace manufacture of carbon-bisulphide, of phosphorus, of graphite, of abrasives, of magnesium for flashlights, of ferro-alloys for steel refining, and in the last few years we have seen the reliability of electric furnace operations pledged in the investments of the millions of dollars that are going into the construction of huge electric furnace plants for the production of fertilizers. And it is the commercial reliability of these older electric furnace applications that is back of the reliability of the newer application of the electric furnace to steel. In them was developed the demand for electrodes which justified the building of electrode factories that now are available for steel furnaces. In them was developed the special engineering ability to handle the large electric currents of furnace work. They had their part in the demand for electric current at lower and lower cost. It is these things that have given the Siemens electric steel furnace of thirty years ago—a technical curiosity then—its standing today as a reliable industrial tool.

Technically, reliability is a matter of apparatus and men and supplies. These react on one another. Experienced and skilled men can force poor supplies and indifferent apparatus to yield a commercial output.

*Paper read at the meeting of the Chicago Section of the American Electrochemical Society, Jan. 20, 1911.

Apparatus that is inherently self-regulating will permit the use of such men as can be had at a permissible cost. Supplies that have a factor of safety in their constitution will carry the equipment through an overload. Involved in the question of commercial reliability is the element of time. Results must be prompt. To make electric steel in the United States means furnaces that are reliable, with American men and American supplies. In the rush of industry, reliability cannot afford the time hazard of importation. Europe developed the electric steel furnace industry first, just as it developed the dynamo first, the steam turbine first, the large gas engine first, steel making first. There the pressure of competition for opportunity, the abundance of trained technical men, and the low cost of capital, force enterprise. Safely we may look in America, with the electric steel furnace, for the same swift overtaking of achievement that has characterized those other technical developments. For this, reliable American electric furnace supplies are now available and tested. American electrodes can be had in sizes as large as furnace requirements call for. Refractories made in this country have been developed with an ability to stand up well under the higher temperature of the electric steel furnace. Campaigns with roofs of American brick have been run into months, as against weeks for European practice. Skilled electric furnace men are scarce. The rapid installation of furnaces more than absorb the few who have had an opportunity to become experienced with electric furnaces working in commercial operation. Trials with electric furnace men imported have not been wholly satisfactory. The drive of the American steel plant does not seem to fit with the methodical nature of the foreign technician. This shortage of skilled furnace men at permissible costs has thrown more of the burden on the apparatus, has directed American electric steel furnace design toward inherent self-regulation, as we say electrically; toward reliability, as the investor looks at it; toward making it "fool proof," as the man on the job phrases it. Furnace detail that works very well in Europe, with the skilled help available there, has to be designed here to work with the help that can be had. The inherent reliability can now be carried to an extent that permits taking a steel melter from a crucible or open-hearth furnace and putting him on an electric steel furnace with the assurance that within a week he can be making salable steel regularly. This does not mean making an electrician of him. Actually he knows no more of electricity than he knows of the kinetic theory of gases in the open-hearth work, but he does know that when the watt-meter needle is in a certain position he has a supply of heat in his furnace, and high temperature heat at that, and heat that, hot as it may be, will not burn his steel.

But a reliable furnace and reliable supplies and reliable melters are not all that are concerned in the commercial reliability of electric steel. The steel produced is different from crucible steel. It is different

from open-hearth steel. If it were not different, the electric furnace would not be used. It is true that under favorable conditions steel can be made cheaper with an electric furnace than in other ways, but the fundamental reason that is speeding electric furnace construction is that the steel is better. Being better means differing chemically, differing physically from the steels of the other furnaces, and this difference chemically and physically must be reckoned with in the subsequent processes the steel goes through. In the foundry the moulding must be done for this new material. It is freer from gases. It usually is hotter. The tendency is to lower the carbon for the same tensile strength, and this makes it set at a higher temperature. The improved elimination of phosphorus and sulphur tend in the same direction. In the forge and rolls, electric steel shows its additional strength and toughness. It must be handled from the point of view of its characteristics. It is a mistake to simply consider the electric steel furnace as a substitute for some other kind of a melting furnace. Its introduction can be counted on to mean changes of method as far as the steel is followed in manufacturing processes, and unless this is clearly recognized there will be disappointment, and the reputation of the electric furnace for reliability will suffer.

From a still narrower technical point of view, reliability means freedom from actual breakdown, from the enforced necessity of change in construction and the consequent loss of time and productive capacity. As to this, electric furnace design is now on an engineering basis. Dependence on experiment is now a matter of volition on the part of the investor. It is no longer a necessity. As a piece of engineering apparatus, an electric furnace can now be more closely designed to meet narrow specified operating conditions than can an open-hearth furnace. In a recent instance, an electric furnace and a small open-hearth were installed side by side in a new steel casting shop. The open-hearth gave trouble, and has been replaced by a better design. The electric furnace, although not of most modern design, has operated without need of alteration. Those details in which an electric furnace differs from established open-hearth practice, the electrode holders, the electrical regulation, the contact with the bath of steel, are now rugged and simple. Electrode holders are in evidence that operate with less than one hour's lost time in 100 heats. Recently I have had occasion to examine carefully in detail the condition of a plant using electric furnaces exclusively, that started operation on a commercial scale in the summer of 1907, which is quite aged as electric furnace design goes. It is proposed to double the size of the unit and increase the size of this plant fivefold, and this examination was made to determine from the condition of the plant and the operating history what changes in the original design could be made to advantage in the extensions. Incidentally, the examination disclosed a condition of continuing reliability that is worth record-

ing. Substantially all of the original equipment was still in use. The plant, which is in the West, has been operated since its installation by Chinese labor under a white superintendent, and neither the superintendent nor the Chinamen knew more of electricity than is involved in the opening and closing of circuit breakers and the reading of ammeters and voltmeters. One of the power transformers had been replaced, but it was learned after its removal that the trouble was an abraded spot on the terminal that could have been readily repaired on a Sunday at the plant. This particular plant, while not a steel plant, is of interest in showing on many of its operating records a heat development efficiency of over 90 per cent. That is, of the heat equivalent of the energy delivered to the plant by the high potential electric wires, over 90 per cent is transferred to the material under treatment.

It is to be expected that the success of this new and flexible tool of industry will lead to over enthusiasm and its application in wrong ways and in wrong places, and that its failure as a panacea for troubles it has properly nothing to do with will bring setbacks. But back of its present success is the cause of that success, the steadily lowering cost of electric current. Thirty years ago the furnace of Siemens worked as an apparatus. It was not until the cost of current dropped below the figure at which the electric furnace was commercially reliable in this country that its industrial application began and its engineering developed. This dropping of the cost of electricity is still going on, and as it goes on it brings a greater and greater motive force to bear on the installation of electric furnaces. At the same time, unfortunately for industry in general, but fortunately for the electric furnace, the cost of fuel for direct and gaseous heat is slowly but steadily rising.

Summed up more briefly, the facts that an investigator of electric furnace results may expect to find are:

1. That the electric furnace itself has passed from the field of experiment to that of engineering, but that the fields of manufacture using the electric furnace products are still experimental.

2. That the electric furnace is technically reliable and will operate continuously with the men and supplies that are available in this country. That the details are simple and rugged, and that the inherent regulating powers can be made such as to bring it well within the ability of usual plant labor.

3. That the electric furnace is commercially reliable. That when installed with the same business care and adaptation to conditions that should be used with other furnaces, it will earn a profit on the investment, and a profit that is larger than the normal manufacturing profit in proportion as the field is newer and more open.

The production of tin and terne plates in the United States in the calendar year 1909, amounted to 1,371,000,000 lbs.

A PROBLEM IN SAVING HEAT IN THE MANUFACTURE OF CARBON DIOXIDE.*

By **STERLING H. BUNNELL.**

A very large business is done in the manufacture of carbonic gas for use by bottlers and others. This gas, pure carbon dioxide, is obtained from the combustion of coke, washed to take out sulphur and other impurities, separated from the other flue gases, dried, compressed to the liquefying pressure and bottled up in steel drums of cylinders for sale. The problems which must be solved in connection with making an economical plant for the manufacture of liquid carbonic gas entail almost every form of steam plant engineering.

The first operation in manufacturing is to burn the foundry coke under a steam boiler. A deep fire is carried and a forced draft used, so that the flue gases may have the highest possible proportion of carbon dioxide. With a properly designed furnace and good firing the proportion of carbon dioxide will run above 14 per cent and may at times reach above 16 per cent. The flue gases leaving the boiler setting are at a high temperature; this heat must all be disposed of before the gas is in condition for absorption. Here is the first point at which waste heat is to be saved. Accordingly, the flue gases are passed through an economizer, the duty of which will be explained later. The gases leave the economizer at a temperature below the boiling point of water.

The gases next pass through two scrubbers in succession, in which they are washed by showers of water to get rid of the cinders and whatever sulphur may be present. In this operation they are cooled to the temperature of the water supplied and this portion of the heat is necessarily wasted. The gases next pass through an absorber, in which they are brought in contact with lye solution. This solution has the property of absorbing carbon dioxide, so that the flue gases emerge from the absorber with a content of carbon dioxide of 10 per cent.

The lye solution from the absorbing tower is allowed to flow into a tank. This part of the solution is called the "strong" lye, because of its containing a considerable quantity of carbon dioxide. The next step is to pump the strong lye into a lye boiler, where it is heated above the boiling point, a portion of the water is evaporated, and a large part of the absorbed carbon dioxide driven off by the heat. Here again is a point at which heat must be applied continuously to a solution, and as the temperature required is little if any higher than 212 degrees, exhaust steam from the power portion of the plant is used for the purpose of supplying heat. The lye solution returns from the lye boiler to a second tank and as its contents of carbon dioxide have been much reduced this solution is now known as the "weak" lye. In order to permit the weak lye to be again used for absorbing a fresh supply

*Metallurgical and Chemical Engineer

of carbon dioxide it must be cooled from its boiling temperature to perhaps 120° .

Observing that the weak lye from the absorber is cooled and must be heated, and the strong lye from the lye boiler is hot and must be cooled, the natural suggestion is to exchange the heat between the two solutions through a countercurrent cooler. This is of the double-pipe type, the hot weak lye from the lye boiler flowing through the inner pipes and the cold strong lye from the strong lye tank flowing through the outer pipes in the opposite direction. In this way an exchange of heat takes place which returns the weak lye at a temperature nearly as low as the temperature of the strong lye on its way from the tank, and delivers the strong lye at a temperature nearly as high as that of the weak lye coming from the boiler.

The strong lye leaves the countercurrent cooler at a temperature which is still considerably below the

been driven off the hot strong lye in the lye boiler and successively passed through coolers and dryers. Another portion of the steam is required for operating the pumps necessary for circulating the lye and for pumping cooling water about the plant. The exhaust steam from compressor and pumping engine still carries the latent heat which it received in the steam boiler. This large supply of heat is devoted to heating the lye in the lye boiler, evaporating a portion of the water and driving off the gas.

The skill required in designing and operating the entire plant is obviously devoted to arranging the closest possible balance between the heat transfers in all parts of the plant. The capacity of the plant—that is, the amount of carbonic gas that can be put into cylinders for sale—depend first upon the amount which can be driven off the lye in the lye boiler; a step further back, upon the proportion of the gas which can be

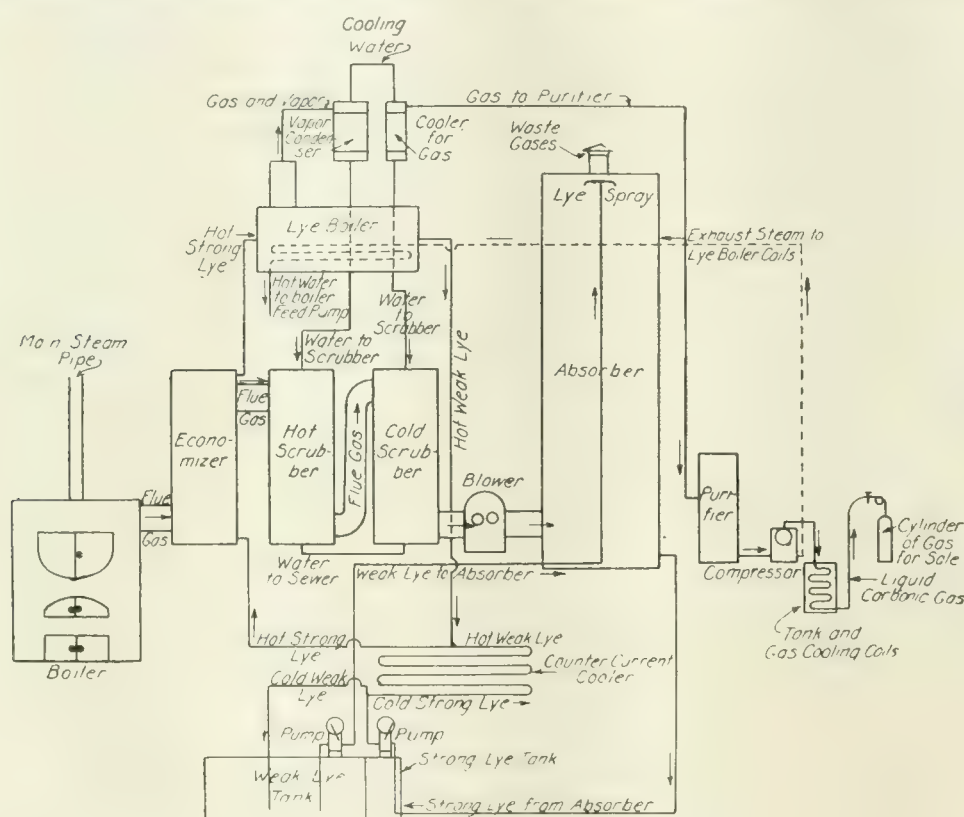


Diagram of Carbon Dioxide Manufacturing Plant.

boiling point, and here comes in the opportunity to use the waste heat of the flue gases from the steam boiler by passing the strong lye through the economizer before mentioned. The strong lye leaves the economizer at a temperature near to boiling, and in passing the economizer abstracts a large part of the waste heat in the flue gases.

All the waste heat in flue gases has now been accounted for. The bulk of the heat caused by the combustion of the coke has gone into the steam boiler and been devoted to the work of evaporation. This affords a good supply of steam available for doing the work of the plant. Most of the work required is for the compressors, which take the gas after it has

recovered from the flue gases in the absorber; a step back of this, upon the proportion of the coke which can be burned into carbon dioxide rather than carbon monoxide, and this again depends upon the efficiency of the firing, the thickness of the bed of coke and the regulation of the draft, so that no free air shall pass through the fire. If the absorption of the gas is efficient the amount of coke burned, and therefore the grate surface, can be decreased. This will decrease the amount of steam made by the boiler, the available power of the plant and the amount of heat which can be recovered from the exhaust steam. Evidently, with a very efficient absorption it might be possible to produce enough to operate the plant, and in that case

every possible saving in steam consumption by engines and all waste by radiation by steam pipes should be cut down as far as possible.

The operation of a complete modern plant for manufacturing carbonic gas would be a profitable study for any steam engineer interested in getting the most out of his steam plant. It is safe to say that very few steam plants of sizes in the neighborhood of 100 hp. are constructed with any such degree of effort to reduce the usual losses to the minimum as is observed in the design and operation of the modern carbonic acid gas plant here described.

THE HILGER REVOLVING GRATE GAS PRODUCER.

The following description of the new Hilger gas producer is taken from a recent issue of *The Iron Age*. The new Hilger producer differs from the existing types of revolving grate producers in several particulars. The lining is of fire brick throughout, no water

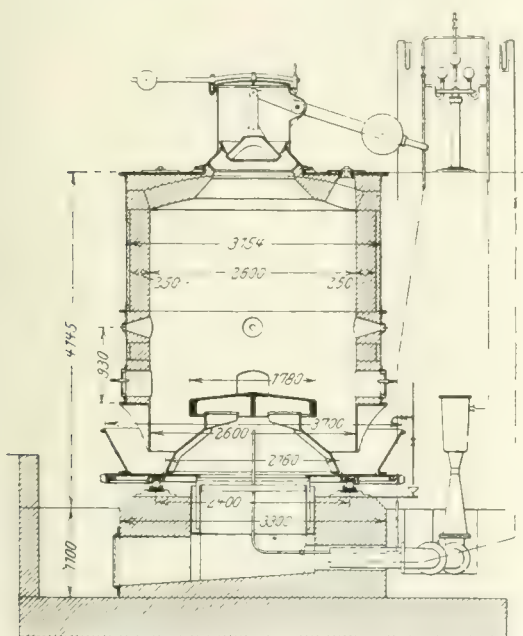


Fig. 1—The Hilger-Gas Producer—Dimensions in Millimeters.

cooled casing being used. This has been found to be entirely satisfactory except for certain coals, which carry a high percentage of alkalis. In these cases, a neutral or basic brick lining must be used in the fire zone.

The producer and the grate are shown in Figs. 1 and 2. The grate is seen to consist of an under part and a cap. Both parts form a star, from which the blast of steam and air can be evenly distributed over the whole cross section of the producer. The star shape is most marked in the cap, which fits into the ashes and by its movement affects the whole column of fuel. The grate is of very simple construction and is relatively low, so that the producer can be built from 2½ to 3 ft. shorter than revolving grate producers of other types. The drive of the grate and ash

pit is another particular feature. The power requirements for one producer vary from ¾ to 1½ hp., according to the size.

The drive, Fig. 3, consists of a forward and backward motion, the effect of both being to give a gradual revolution to the right. It is described in some detail in the paper, and as may be seen in the illustration consists of a worm rack and a worm, which is

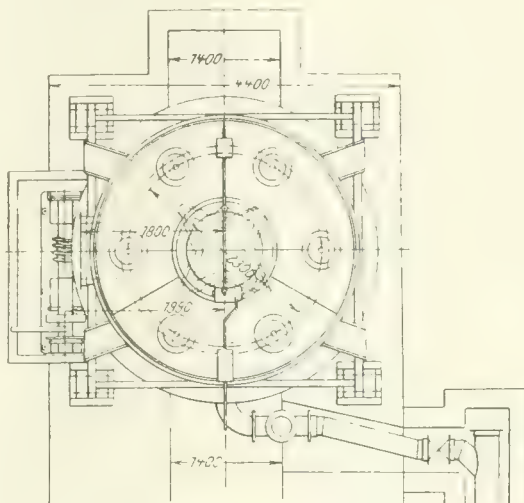


Fig. 2—The Revolving Grate.

driven by a ratchet wheel and a double pawl. Movement is given to the pawl by the eccentric rod shown on the right of the illustration. The action of the pawl is varied, according to the ash of the fuel. If too much ash is expelled the drive is so adjusted that the backward and forward motions are equal, and the resultant effect is nothing.

The ashes are removed by another patented device. A freely swinging shovel is hung in the ash pit, so that on the backward movement it swings freely over the ashes. On the forward movement, however, it locks. In this way no resistance is offered to the backward and forward movement.

The gas furnished by the producer is of uniformly

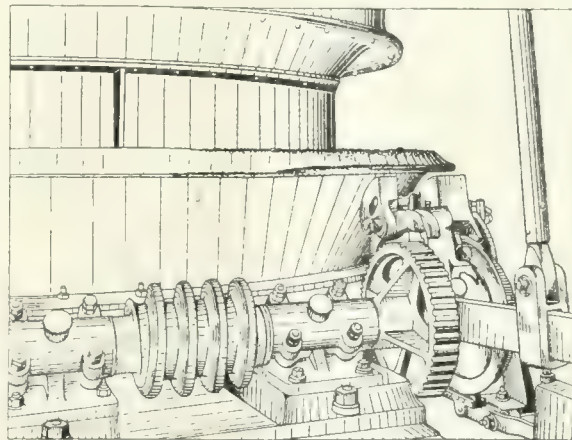


Fig. 3—The Drive of the Grate.

high value, as is shown by the following results, which are averages, and not special analyses:

	GAS ANALYSIS			
	1	2	3	4
CO ₂	3.20 to 3.60	3.40 to 4.00	3.00 to 4.00	3.20 to 3.50
O	0.15 to 0.41	0.20 to 0.40	0.00 to 0.50	0.10 to 0.20

CO	29.50 to 31.84	25.20 to 27.10	28.40 to 31.30	27.30 to 28.30
H	7.60 to 9.00	7.90 to 9.80	10.70 to 12.70	9.50 to 10.40
CH ₄	2.31 to 3.32	3.20 to 3.80	2.10 to 3.30	52.50 to 52.80
N	53.57 to 56.77	57.10 to 58.60	50.10 to 52.00	54.10 to 56.70

No. 1 in the above table shows the results of four analyses of gas from an Upper Silesian coal with up to 50 per cent of dust. No. 2 shows the results of six analyses of gas from average French coal, the charge being 50 per cent lump and 50 per cent dust. No. 3 gives the results of eight analyses of gas from brown coal briquettes. The gas also showed from 0.10 to 0.40 per cent of hydrocarbons, in addition to the methane. No. 4 shows the results of eight analyses of gas from a 3 to 1 mixture of a good Westphalian coal and brown coal briquettes. This producer may be obtained in three sizes, and is built by the firm of Froetter, Duesseldorf.

It is estimated that there are now a million Para rubber trees in various stages of growth in Cochin-China, three-fourths of which were planted before 1910. Many companies are now embarking in rubber growing, but it will be eight years or so before the Cochin-China output will make a good show.

The world's production of artificial silk in 1904 was 1,400,000 kilos; in 1905 it was 2,000,000 kilos; in 1906, 2,400,000 kilos; in 1907, 3,000,000 kilos; in 1908, 4,000,000 kilos; and in 1909, 5,500,000 kilos. Of the total 1909 production it is estimated that the gun-cotton and cupro-ammonium processes were each responsible for more than 2,000,000 kilos, while over 1,000,000 kilos were made by the viscose method. It would appear, therefore, that the guncotton system is finding the competition of the two newer methods formidable, for in 1908 the total output of the cupro-ammonium process was under 1,500,000 kilos, and that of the viscose process only 500,000 kilos, the guncotton process being credited with an output of 2,000,000 kilos.

Newfoundland has extensive peat areas spread all over the country, and a resolution has been introduced in the house of assembly in regard to its manufacture in the colony. The International Carbonizing Co. (Ltd.), of London, England, seeks to obtain crown lands for the erection of factories to manufacture this product into briquets or blocks which will do the same work as coal at a much cheaper rate. Not only is the peat to be manufactured into fuel, but into other articles, such as oakum, mats and matting, yarn and ropes, and a fine substitute for leather can be made from it, also material for paper making. Each factory is to cost at least \$125,000 and have an output capacity of 20,000 tons or more annually. The company agrees to pay the government a royalty of 10 cts. per dry ton of peat coal briquets manufactured, and a further royalty of 10 cts. per ton for other products which they may manufacture. The company will have the use of water powers and will be exempt from taxes and duty on their machinery.

AN ELECTROLYTIC FURNACE METHOD FOR PRODUCING METALS.*

By JOHN WOODS BECKMAN.

An electrolyte is generally understood to be a liquid which will conduct electricity, and the substances dissolved in this liquid are dissociated into two so-called ions. This was Arrhenius' theory, and has been applied ever since to water solutions. I believe though that O. Lehman and Lorenz and his co-workers are to be credited with having shown that the laws governing the water solution electrolyte are equally applicable to the molten electrolyte, and to substances dissolved in the same.

The above mentioned investigators have found and proved beyond a doubt that the substances dissolved in the molten electrolyte are ionised and that the Arrhenius dissociation theory applies thereto; also that the laws governing the dissociation, the movement of the ions, and the speed of their movements, can be determined the same in molten as in water electrolytes. Lorenz in his classic "Die Electrolyse Geschmolzener Salze" has added through his painstaking work enormously to the knowledge of the fused electrolyte, the electrolyte of the future.

Up to the present time the industry which is making use of the molten electrolyte to the largest extent is the aluminum industry. The ionic theory has long been applied to the reduction of aluminum from its oxide, in the present well-known manner. Lorenz in the above-mentioned work publishes a very careful study made of the electrolytic deposition of aluminum from a molten electrolyte, and convincingly proves that it is an ionic reaction. His idea is confirmed by all who have studied the reaction and phenomena connected with the manufacture of aluminum, these phenomena are identical with those met with in plating copper from a copper-sulphate solution.

I once had occasion to do a considerable amount of work in connection with the manufacture of aluminum, and then appreciated the simpleness of the procedure. It should be possible to produce all metals in a similar manner from their oxides; but what electrolyte is to be used?

In the aluminum industry cryolite is used as a bath material; a mineral mined at considerable expense, and together with that of the other ingredients used in the electrolyte, its cost is considerable. The fusing point of this electrolyte is low, and it is on this account suitable for producing metals of low melting points.

As a general law it can be considered that the electrolyte fused or aqueous, from which a metal is to be produced should be a solution containing in some form the desired metal. Cryolite, containing mainly the metals sodium and aluminum, is on this account

*Paper presented at the 19th General Meeting of the American Electrochemical Society, in New York City, April 6-8, 1911. The author states that this paper is what might be termed "preliminary," and he hopes at a later meeting to be able to give some details which, in the present experimental stage of the progress, would not now be advisable to offer.

not suitable as an electrolyte for the production of any other metal than aluminum.

When most salts are heated to their melting point they dissociate, and this is especially the case with the salts of metals not belonging to the alkaline or the alkaline-earth metals. There are, however, exceptions among the heavy metal salts such as fuse but do not dissociate, and among these are the fluorides. Gin in his excellent monograph published in late issues of the Transactions of this Society, has outlined methods using the double fluorides of various rare metals, and using these as fused electrolytes out of which to produce the desired metals. But these fluorides have to be produced artificially, and although the process seems perfectly feasible, the cost seems to be prohibitive to using these electrolytes commercially.

A comparatively little known group of salts, are those occasionally occurring in nature, composed of equivalent amounts of calcium oxide and the oxide of any metal less positive than calcium. These salts occur occasionally in nature comparatively pure, as is the case with Scheelite, and can very readily be produced by heating the oxides together in any suitable manner.

It is evident that there is a great number of these salts possible, and the characteristic feature of them is, that they all fuse at about and over 1000°C ., the temperature varying according to the metal oxide which calcium-oxide is combined with. The boiling point of these salts is very high; I have heated some to 2200°C ., and the molten bath has been perfectly quiet, without any appreciable signs of the ingredients boiling off. The valuable feature of these salts, that I have availed myself of, is that when in a molten condition they carry the current very readily, and are very suitable as molten electrolytes for the production of a great number of metals. On a small scale I have produced a number of metals using the corresponding salt as an electrolyte. Managanese, iron, chromium and a 50 per cent alloy of vanadium are the metals that I have produced so far. In the case of the alloy, it is interesting to note that it was produced as well out of the sulphide as out of the oxide. Crude ore was used, giving not a pure metal, but one containing some impurities such as silicon, unobjectional as far as the steel trade goes.

The following gives in brief the procedure in manufacturing a metal by means of a fused electrolyte composed in the manner mentioned above. Take for example iron. Equivalent amounts of Fe_2O_3 and CaO are mixed together and heated in any suitable manner. The salt $\text{CaO.Fe}_2\text{O}_3$ is formed and fused in a receptacle fitted for electrolytic purposes. Oxide of iron is introduced into this fused bath, and this molten solution is electrolysed, iron being deposited on the cathode and a lively oxygen development taking place on the anode.

If the electrolyte itself is submitted to the direct current, no noticeable reaction takes place, but the moment iron oxide is added the electrolysis starts, and

by adding from time to time more oxide it can be carried on continuously.

The actual intermolecular phenomena that do take place when electrolysis goes on are hard to define, there may be a continuous rearrangement of the salt structure, the iron oxide in the salt becomes detached, and another molecule of iron oxide takes its place, while the former is ionised, and gives off its electrical charges depositing iron and oxygen.

This method of producing metals is protected by patents in U. S. A. (U. S. Patent No. 973,336) and in other countries.

The practical features of this method are: First, cost of electrolyte practically negligible, as it is composed of calcium oxide and the metal oxide of the desired metal; second, a great quantity of metal can be produced out of the same electrolyte; third, in case the electrolyte becomes fouled, the loss in discarding the electrolyte is practically negligible; fourth, the flexibility of the process, making it possible to produce a great number of metals and their alloys such as iron, manganese, chromium, zinc, tin, titanium, vanadium, tungsten, tantalum, molybdenum, nickel, cobalt, lead and a number of others more or less rare.

In the following I am giving a list of amount of metal deposited per ampere-hour, considering only ideal conditions, and it is worth noticing, that aluminum has the minimum amount of metal deposited per ampere-hour, and is the only metal as yet that is produced commercially out of a fused electrolyte. Its use as a reducing agent for the production of many of the rare metals is well known.

Metal.	Oxide.	Gram per amp. hour.
Aluminum	Al_2O_3	0.3377
Lead	PbO	3.868
Chromium	Cr_2O_3	0.655
Iron	Fe_2O_3	0.696
Manganese	MnO	1.027
Nickel	NiO	0.732
Zinc	ZnO	1.220
Titanium	TiO_2	0.145
Vanadium	V_2O_5	0.64

This process opens up a field which has heretofore been very little studied, and offers I believe a solution to many baffling electro-metallurgical problems.

A U. S. consular agent at Almeria, Spain, reports that a new ore, known as a double sulphate of aluminum and potash (sulfato doble alumico-potasico) and called after its discoverer, Senor Calafat, was brought into notice six or eight months ago. The district in which it was discovered is only about two hours' drive from Almeria. No work has as yet been carried out to show the real nature of the deposits, but the owner is said to have estimated the quantity at 60,000,000 tons. The lode has a run of some three miles. Senor Calafat is said to be erecting works near Madrid to put the mineral to a practical test and several carloads have been shipped for treatment in the furnaces. The analysis is given as follows: Anhydrous sulphuric acid, 34.77 per cent; oxide of aluminum, 37.98 per cent; potash, 9.64 per cent; water, 17.61 per cent; specific weight, 2.75; hardness, 2.50 to 3.

METALLURGY.

THE CLANCY ELECTROCHEMICAL CYANIDE PROCESS.*

By JOHN COLLINS CLANCY.†

This process is more immediately concerned with the treatment of refractory ores, these ores generally having their values associated with compounds in chemical combination, which are not susceptible to the action of solvents until roasting or some other method of oxidation has disassociated the said compounds and liberated the precious metals. Roasting is by all means the cheapest oxidation process so far known; any attempt to oxidize the base constituents by other chemical means is obviously of a very expensive nature, therefore, commercially impracticable. It is known that, in some cases, profitable extraction of the precious metals from base ores may be obtained, depending on how rapid the action of the cyanide is upon the finely divided gold in preference to its action upon the base constituents—that is to say, before the base metals are attacked the gold has been dissolved. Sometimes by keeping down the alkalinity of the solution to an almost neutral point in "protective" alkalinity a good extraction may be obtained, as the absence of an excess of alkali will protect the metallic sulphides from decomposition and allow the cyanide to work directly upon the precious metals. Of course, these results are only capable of accomplishment when the precious metals are not in chemical combination, unless a solvent for the chemical compounds is present with the treatment solution. The Clancy process in a measure includes the embodiment of these principles.

As a preamble, a little history relating to my earlier experiments on refractory ores may not be out of place. The ores of Cripple Creek, Colo., may be taken as an example of so-called refractory ore. I need not dwell upon the past or present metallurgical treatment of these ores, but will only state, in passing, that straight cyanidation of the raw undressed ore yields but a trifle over 60 per cent extraction. It will be apparent, therefore, that a process to extract from about 90 to 95 per cent of the values without roasting or concentration, and at a reasonable cost, would be a boon to the district of Cripple Creek, as well as to other similar districts in different parts of the world. To this task I set myself some years ago, and my improvements may be seen from reading the different specifications, which show my line of thought and gradation from earlier processes.

The object of my first experiments on Cripple Creek ores was to find a cheap alkaline solvent for the telluride of gold which occurs in these ores, the tellurides existing principally in the form of $(\text{AuAg})\text{Te}_2$; the mineral calaverite and sylvanite may be approximately represented by this formula. The minerals calaverite and sylvanite are often found alone, but more frequently associated with pyrite; this close association with the pyrite increases the difficulty of treatment by chemical solutions as well as separation by mechanical means.

I discovered that the alkaline hypochlorites hypobromides, hypoiodites (NaClO) (NaBrO) (NaIO) dissolved tellurium readily, and also attacked the tellurium in the gold telluride compounds, while their calcium salts—the analogues of bleaching powder—were not solvents. Sodium hypochlorite being the cheapest, there was therefore a possibility that by crushing the ore fine the tellurium would be dissolved off, and allow solution of gold by cyanidation. The equation representing this reaction may be as follows:

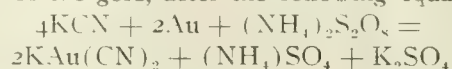


the tellurous acid so formed would dissolve in the excess of alkali—



A preliminary treatment with NaClO , followed by a cyanide treatment on ore crushed to 100 mesh, invariably gave an extraction of from about 80 to 85 per cent of the gold contents, and frequently a higher extraction when the gold telluride in the ore was not closely crystallized with the pyrite. The preliminary treatment, however, had its drawback, inasmuch that the remaining NaClO left in the pulp after treatment destroyed cyanide unless washed out entirely; this washing caused extra filtering and meant two sets of solution treatments, and for that reason, together with the alleged high cost of producing NaClO , the test was not continued.

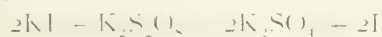
My next process was the employment of a soluble persulphate, such, for example, as ammonium persulphate, as a preliminary treatment to dissolve the tellurium contained in the ore. This solvent, $(\text{NH}_4)_2\text{S}_2\text{O}_8$, was superior to NaClO , owing to the fact that the iron sulphides did not become oxidized to any appreciable extent while the tellurium was undergoing solution. Another advantage—the remaining persulphate did not act to destroy the cyanide on subsequent treatment; on the contrary, it hastened solution of the gold, after the following equation:



The high cost of the persulphate, together with the

*Paper read before the New York Section of the American Electrochemical Society, on December 16, 1910.

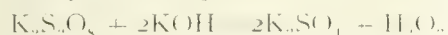
necessity of using two sets of solutions, was the reason for not continuing this method. I next tried if it were possible to dissolve the tellurium and gold simultaneously by using a mixture of cyanide and persulphate. This scheme was found to work satisfactorily on gold telluride crystals, but when applied to the ore the method was prohibitive on account of the large amount of persulphate required, due principally to the presence and formation in the solution of ferrocyanide, which caused the action of the persulphate to be completely nullified. The thought then occurred, could some substance be used to prevent the action of the ferrocyanide upon the persulphate? Reflecting on the behavior of the alkaline hypo-halogen compounds towards KCN, I recalled the weak action exhibited by sodium hypoiodite upon KCN in comparison to that exhibited by sodium hypochlorite and sodium hypobromite; the latter compounds almost immediately converted the KCN into KCNO in alkaline solution—in acid the solution formed cyanogen bromide and cyanogen chloride, respectively. Guided by this process of thought, the idea of using a soluble iodide, together with the persulphate in the cyanide solution at once suggested itself, and I forthwith tried the effect of potassium iodide in conjunction with persulphate and cyanide solution, on gold tellurides, with the result that the tellurides underwent complete solution. The nature of the reaction at once explained itself—there being no alkali present in any of these tests, other than the alkali occasioned by the hydrolysis of the cyanide solution, it was simply the persulphate liberating the iodine in presence of the cyanide solution to form cyanogen iodide. The following equations may represent the action shown in two stages:



then

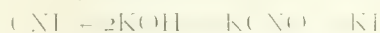


From the study of these reactions I conceived the idea of liberating iodine while the ore was in contact with the cyanide solutions by the use of a substance yielding nascent oxygen, using any oxidant which had not a too high velocity reaction, and one which did not unduly increase the alkalinity of the solution so as to prevent the formation and existence of cyanogen iodide in the said solution. Persulphate apparently filled these conditions, as it introduced no free alkali while giving off nascent oxygen, but, on the contrary, neutralized alkali, owing to its anxiety to part with one of its sulphons (SO_4)



It will be seen from the above description that the solutions would contain free cyanide, together with cyanogen iodide and the oxidizing agent—persulphate—the proportion of cyanogen iodide to cyanide depending upon the speed of the reaction and ratio of persulphate to potassium iodide. To apply this solution to the treatment of ore and maintain the chemical equilibrium necessary for producing cyanogen iodide

required very skillful manipulation. For example, the ore had to be made alkaline by applying the necessary protective alkali to the cyanide solution; if excess of alkali was added, the persulphate would still liberate iodine, but the iodine so liberated immediately formed alkaline hypoiodite; the hypoiodite formed was rapidly destroyed by the sulphides contained in the ore, thus rapidly using up the oxidizing energy of the oxidizing agent. The amount of alkali to be added depended, first, on the acidity of the ore; second, on the proportion of persulphate added, the persulphate requiring alkali in proportion to the removal of its (SO_4); third, the influence of time on the rate of progress of the reaction; fourth, the influence of atmospheric oxygen during agitation, accelerating the oxidation of sulphides, which developed acidity and required alkali. To control this alkalinity, sufficient alkali had to be added, but not enough to destroy the cyanogen iodide; again, the cyanogen iodide formed would, in the presence of alkali and sulphides, form sulphocyanides, representing a direct loss of cyanide. Nor is this all; when the agitation treatment was completed, the solution, after leaving the ore, preparatory to filtration, would drop—*i. e.*, show a greater cyanide consumption than when in contact with the ore. This consumption of cyanide was probably due to the cyanogen iodide formed, together with the remaining persulphate, acting upon the cyanide solution in absence of the retarding influence of the reducing agents in the ore. When it was necessary to add alkali to the solution in order to secure the proper precipitation in the extractors, a further consumption of cyanide took place after the equation:



I discovered that this consumption of cyanide could be completely stopped, and the cyanide which had combined to form cyanogen iodide, regenerated, by the simple addition of sodium sulphite (Na_2SO_3) and lime after the equation:



The lime added neutralized the acid produced. The addition of sodium sulphite acted perfectly at this point, but it did not take care of the decomposition of cyanogen iodide to alkaline cyanate produced by the necessary presence of alkali in the ore treatment process (a measure, of course, not allowable in the treatment tank). It is obvious that the above process would require a skilled manipulator to obtain results at a reasonable cost; in a word, the process is not fool-proof. However, if the market price of persulphate would drop to a reasonable figure, and cheap salt used which would undergo hydrolysis—*i. e.*, give hydroxylions in solution to the extent of taking up the acidity produced by the oxidation means employed—then the process would require no more attention than the ordinary straight cyanide process.

OZONE AS AN OXIDIZING AGENT

In view of the high cost of persulphate, my attention naturally turned to the use of a cheaper oxidiz-

ing agent and one that would not introduce, as before stated, alkali in the solution. Ozone, therefore, claimed my attention, as previous experiments with ozone gave interesting results, and as there were several ozonizing machines on the market at this time, I began to look around for the most economical machine which produced the greatest amount of ozone at the lowest cost. One machine in particular was guaranteed and backed up by a reputable, well-known firm to produce from 100 to 150 grams of ozone per kilowatt hour, and figuring the cost of electrical energy at one cent per kilowatt hour, it seemed that I had now struck the right nail on the head as regards a cheap oxidizer. An ozonizer of the above description was immediately purchased and installed in my laboratory, whereupon I at once set to work to test the action of the ozone on the treatment of ores with cyanide solution containing soluble iodides, using the ozonized air produced by the ozonizer to liberate the iodine from the necessary chemicals while in contact with the ore. It would take up too much space to give in detail the results of the numerous tests made on different samples of ore; in short, the ozonized aid acted as a first-class oxidizing agent and gave excellent results. The only thing against this scheme was that it contained too much cyanide, unless very carefully watched, it being difficult to establish the proper equilibrium necessary to prevent rapid oxidation to cyanide. The equation representing the action of the ozone upon the iodide was as follows:

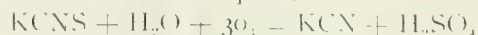


and, of course, the iodine combined with the cyanide to form cyanogen iodide, as already shown in one of the foregoing equations, the excess ozone forming with the cyanide KCNO after the equation:



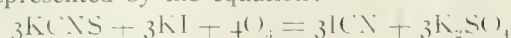
The question now was how to prevent this formation of KCNO.

Learning from previous tests on ores containing large amounts of reducing agents that the consumption of cyanide was not so great as with ores which did not contain reducing agents, the thought suggested itself to add a reducing agent to the cyanide solution. I knew that potassium sulphocyanide, although in a sense a reducing agent, had no action on straight cyanide treatment—*i. e.*, the atmospheric oxygen was not absorbed by it. Experiments showed that when sulphocyanide was subjected to ozonized air, cyanide was generated after the equation:

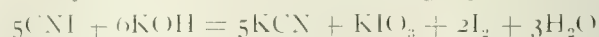


The alkali present of course neutralized the acid as produced, and the cyanide so formed was not decomposed as long as the sulphocyanide was present. This action opened up a field for thought, and in following it I tried the action of ozonized air on sulphocyanide solution containing an iodide. Upon introducing the ozonized air into the vessel containing the sulphocyanide and the iodide solution, free iodine was at once liberated. When this experiment was re-

peated, using part of the same original solution, but with a small addition of alkali, on again passing the ozonized air through the mixture the solution immediately assumed the characteristic iodine coloration. This solution was exceedingly active on gold leaf and on telluride of gold, presumption being that the solution contained cyanogen iodide and free cyanide. The formation of cyanogen iodide by this means may be represented by the equation:



Following up this discovery, as shown by the foregoing equations, I proceeded with my tests, using the sulphocyanide to prevent the direct formation of cyanate, and was rewarded with most satisfactory results, the consumption of cyanide being very much reduced. However, the amount of cyanide consumption which did take place over and above that consumed by the ore, as compared with straight cyanide, was probably occasioned by the cyanogen iodide formed being oxidized slowly to cyanate in presence of alkali. The speed of this reaction was not great, and probably occurred after the following equation:



Potassium hypiodite (KIO) being formed as an intermediate product, the excess of iodine acting upon more alkali forming hypoiodite would destroy the KCN formed and eventually become oxidized to KCNO.

Here was a field for the expansion of the process on the large scale, but there is always a "but." While the production of ozone was apparently cheap enough as regards the cost of electrical energy, the cost of the machine (installation) to produce enough ozone to treat, say, 1,000 tons of ore per day was staggering. The machine required the supervision of trained experts, and for a 1,000-ton-a-day plant a battery of about 500 unit ozonizers would have been required, besides various other mechanical objections too numerous to mention. "Another hope blasted!"

The production of ozone by electrical means is yet in its infancy, and it is only within the last few years that ozonizers have been put on the market to give ozonized air of high concentrations. In the very near future it is probable that ozonizers will be built to give large volumes of ozonized air at high concentration with a small installation of ozonizer units. Meanwhile I had to look around to find a substitute. How was it to be found? I had already exhausted almost the whole list of economical oxidizers known to chemistry.

ELECTROLYSIS.

The employment of the electric current claimed my attention as a means of producing the oxidation necessary while the solution was in contact with the ore. Electrical means for the treatment of ore labors under a stigma, inasmuch as there have been many unfavorable criticisms on the use of electrolytic processes, no doubt caused by the numerous failures. Among these may be mentioned the electrical precipitation of

gold, requiring the installation of an enormous plate surface, electrode difficulties, and the consequent destruction of the cyanide by the use of the current. One of the most severe criticisms on electrolytic processes is where the direct precipitation of the values from the ore pulp is concerned, the objections probably be due to the scouring of the plates by the circulating ore and the consequent loss of finely divided gold amalgam in the ore pulp. Much more could be said on this subject, but the Clancy process is not concerned with any electrolytic process, nor deposition of the gold on plates, nor upon the dynamical theory of the gold being brought against the cathode electrode and there meeting with nascent hydrogen, nor the ore being brought against the anode electrode and meeting with the nascent oxygen. The process thus differentiates itself into a method of using the electrical energy to disassociate the chemicals employed, and by their agency doing the necessary work.

ELECTRODES.

It will be obvious from the foregoing that the use of electrolytic methods requires proper anodes. Numerous substances to act as anodes have been tried, such, for example, as lead peroxide, carbon, iron, graphite, etc., all these substances having been more or less inefficient. Anodes of lead peroxide under the influence of high current densities break down—*i. e.*, disintegrate. Iron when used as an anode is oxidized by the nascent oxygen liberated, and forms hydrated oxides of iron, and should the cyanide solutions contain halogen salts, the corresponding iron halogen compounds are formed. It is needless for me to describe the many difficulties which I have been up against to find a suitable anode. All the common metals, together with different alloys and compounds, were experimented upon to find a substance which would withstand the action of the halogen compound—*i. e.*, some substance that would not combine with either chlorine, bromine or iodine under the action of the current—so as to allow the liberated halogen or the nascent oxygen to do the useful oxidation work in the solution.

Graphite (extruded) stands the electric current fairly well—*i. e.*, makes a good anode while it lasts—but under the influence of high current density it disintegrates and cannot be used for any length of time in circulating solutions containing ore in suspension, as it is worn away by the attrition or friction of the ore particles against the said electrodes. Nor is this all. When it is understood that graphite will not stand a much higher density, say, 10 amperes per square foot of surface, it requires no calculation to figure out the amount of electrode surface required on a working scale. The only metal which would stand up under the electrolysis of cyanide-bearing solutions containing halogen compounds was platinum. On a commercial scale the employment of platinum could not for one moment be entertained.

After trying out every known electrode which I

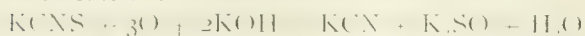
thought might prove suitable, I found that oxide of iron which had been melted in the electrode furnace was the only thing that would stand up under high current densities and not combine with the halogen compound contained in the cyanide solution. The area of anode surface of this material to give the necessary current density is reduced to a fraction of the amount of surface required in the case of graphite. In general, these melted iron oxide electrodes used, as described, are particularly suitable for the electrolysis of cyanogen-bearing solutions, and present the following advantages over all other anodes, including platinum: First, they are cheaper, and they have unlimited life in comparison with the other anodes already mentioned; second, they furnish nascent oxygen free from carbonic acid, which is of great advantage in the electrolysis of cyanide solutions; third, they do not disintegrate under high current densities, and are so hard that they do not wear away by the attrition of the ore pulp in the agitating tank.

Armed with the necessary weapons in the shape of these highly inert refractory electrodes, and with confidence in my process based on previous experiments with platinum electrodes, I felt sure that I could accomplish anything in the shape of electrolysis of cyanogen-bearing solutions containing halogen compounds. I could now use high current densities and liberate chlorine, bromine or iodine in the said solutions without worrying about my anode breaking down. It will be remembered in my earlier experiments that I was confined to the use of persulphate, together with iodine or soluble iodide, as no other oxidizer, not even ozone, would liberate bromine or chlorine in the alkaline cyanide solution. Taking advantage of these new-found electrodes, I experimented with cyanide, sulphocyanide and a soluble bromide, and obtained excellent results on the most refractory ores by keeping the solutions at the neutral point, so as to allow of the formation of bromo-cyanogen while in contact with the ore. The use of a bromide as above described, although giving results equivalent to the use of soluble iodide, was more difficult to control, owing to the necessity of maintaining the solutions at the neutral point, and required more electrical energy to do the same work as iodide. Again, by keeping the solutions at the neutral point, the formation of cyanate was much more rapid, consequently the free cyanide did not coexist very long in the presence of the cyanogen bromide, owing to its high speed of reaction. The bromates produced by the oxidation action would accumulate in the solutions and were apparently not reduced to bromides even after passing through the zinc boxes. In the case of using a soluble iodide, any iodates formed were at once reduced by the action of the sulphocyanides. This reaction between sulphocyanide and an iodate is very striking, so much so that if we take a solution of an iodate and add to it a solution of sulphocyanide, on acidifying this mixture immediate liberation of iodine

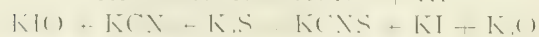
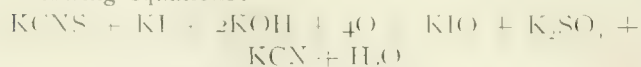
takes place; or, again, if we take a solution of iodate containing sulphocyanide and electrolyze the same, iodine is immediately liberated at the anode. With bromates this is not the case. Bromates are reduced more or less by the nascent hydrogen evolved at the cathode. But this nascent hydrogen has to do other work; in fact, it has no time to look after the complete reduction of bromates, as during electrolysis a thin pellicule of calcium hydrate is produced on cathode. The formation of this pellicule is occasioned by the presence of calcium salts in the solution. This pellicule scales off, but is immediately reformed, thus more or less preventing the hydrogen gas in its atomic state from coming into contact with the electrolyte and causing a union of the atoms to molecular hydrogen. When the hydrogen has once assumed the form of molecules, then its reducing power is at an end. Bromates, therefore, have a chance of partial reduction in the treatment tank. The advantages possessed by the use of iodine over bromine are the following: Iodine is easily liberated from its compounds at a very low voltage; it does not react violently upon the cyanide solution; it is continuously reduced from its higher oxidation state by the sulphocyanides present in the solution; it can be liberated from its compound in alkaline solutions—this fact prevents the loss of valuable iodine due to volatilization. The fact that iodine forms cyanogen iodide in a slightly alkaline solution is of extreme importance in the treatment of ores, as by this fact sufficient free cyanide coexists during the formation of cyanogen iodide. The decomposition of cyanogen iodide to cyanate in the presence of alkali takes place slowly, thus allowing ample time for the precious metal's dissolution.

This method of using cyanide sulphocyanide and a halogen compound in conjunction with the electrolysis of the solution gave excellent results on most of the so-called refractory ores. The cyanide could have been regenerated from the KCNS (*i. e.*, the KCNS which had formed due to the action of the cyanogen iodide upon the sulphides during treatment) but for the presence of the halogen compound.

The reaction:



takes place only to a limited extent. The presence of the halogen salt handicapped this reaction—*i. e.*, it finally became more or less a reversible one after the following equations:



In a word, the formation of KIO oxidizes the KCN to cyanate, and when regenerating the solution in the presence of ore, the KIO acts upon the sulphides in the presence of alkali, producing KCNS. It will be seen that it is difficult to establish the correct ratio of alkali to obtain an equilibrium to allow of the formation of KCNS and prevent the formation of cyanate. How could the formation of cyanate be prevented?

How could the cyanogen-bearing solution be regenerated? Numerous experiments made with all kinds of additions, conducted upon scientific principles, with a view to reducing the cyanate to cyanide, gave most discouraging results. As far as I am aware, there is no record in technical literature in reference to the regeneration of cyanide from solutions of cyanate; therefore I had to launch out into a sea of speculation. My first experiment was to try the action of the electric current upon a chemically pure solution of potassium cyanate. The action of the current upon straight cyanide (as is well known) results practically in the formation of cyanate. I therefore tried the effect of continued electrolysis upon a pure solution of KCNO. It was not my intention to chase any scientific butterflies, but only the question as to what were the products of electrolysis. I forthwith proceeded with my experiment. On examination of the cathode product, NH_3 revealed its presence by its action on the olfactories. At the anode I collected CO_2 , the solution meanwhile increasing in alkali. These products for a start were by no means encouraging. However, as a preliminary to an intended systematic research and to see if there was any possibility of some reduction to cyanide, I withdrew a portion of the liquid from the containing vessel and floated some gold leaf upon the surface of the solution and set it aside for observation, meanwhile continuing electrolysis of the remaining solution. After about ten minutes this solution became very hot. I was about to throw it out as a spoilt experiment, but, thinking there might be some intermolecular change due to the increased temperature (having in mind, amongst other things, Wohler's discovery, *viz.*, conversion of ammonium cyanate into urea), I withdrew a second portion of the solution and floated gold leaf upon its surface, placing it alongside the first experiment. I watched both solutions carefully. No. 1 experiment, which had already stood fifteen minutes, showed no signs of dissolving; No. 2 experiment, after ten minutes, surprised me, for the gold was surely being dissolved, and in thirty minutes it was completely dissolved. No. 1 experiment after standing all night showed no signs of dissolution. Thinking that the reason why No. 1 did not dissolve gold leaf was owing to its not having the same amount of current as No. 2, I set to work and made two experiments under precisely the same conditions, with the exception that one was kept cold by immersion of the containing vessel in a freezing mixture, while the other one was allowed to become warm as before. After giving both the same ampere hours of electricity, I floated gold leaf on each solution. On thirty minutes' standing, the one that was allowed to become hot dissolved gold leaf completely; the one which was kept cold showed no signs of dissolution, even after three days' standing. Platinum electrodes were used in both experiments. What was the cause of the difference between the electrolyzed hot and cold solutions? Was it the formation of urea? I at once electrolyzed a

solution of potassium cyanate containing a small portion of urea, adding a small amount of caustic soda to increase the conductivity. By this means the solution kept at about room temperature. On completion of electrolysis, gold leaf was floated upon the surface of this solution and allowed to stand to await results. In about fifteen minutes the gold leaf underwent complete solution. From this it would appear that the urea theory was probably correct, or was it the cyanate in conjunction with the urea, together with the action of the electric current-producing cyanide, isopurpurate, or some other gold-dissolving compound? Testing with silver nitrate showed apparently considerable quantity of KCN, but nothing commensurate with its dissolving value on gold leaf.

DISCOVERY OF ACTION OF CYANAMIDE.

Having made this discovery, that urea in conjunction with the cyanate and electrolysis of the solution dissolved gold leaf, the next question was: Would any similarly constituted organic compound do the same thing? Urea being an amide compound (carbamide), I at once thought of cyanamide. I thereupon added a solution of cyanamide (using the calcium cyanamide of commerce, which is soluble to the extent of about 58 to 60 per cent in water) to a cyanate solution, electrolyzed same, and treated the solution as before with gold leaf; dissolution of the gold took place in a few minutes. With a view to studying this peculiar reaction, I tried the action of electrolysis on a solution of calcium cyanamide. Without any other admixture other than the addition of a little alkali to increase the conductivity, after electrolyzing for ten minutes at a fairly high current density, the solution dissolved gold leaf in about fifteen minutes. This solution also gave a substantial titration with silver nitrate, showing probably that cyanide had formed. Again treating the original solution with the addition of potassium thiosulphate, and boiling for the purpose of converting any cyanide present into sulphocyanide, on testing this solution with ferric chloride it gave the characteristic blood-red coloration of the iron sulphocyanide—surely ample verification; but to make it more confirmatory, I added a solution of ferrous sulphate to convert any cyanide in the solution into ferrocyanide. On addition of ferric chloride it gave, on boiling, the characteristic Prussian blue. I was therefore satisfied that some cyanide was formed by the electrolysis of calcium cyanide, but as to the amount produced from a given quantity of cyanamide I was in the dark, owing to the peculiar behavior of the titrating solution in presence of organic compounds. It seemed, from different experiments made, that the production of cyanide by electrolysis of cyanamide solution frequently never rose above 0.3 to 0.4 pound KCN per ton of solution. During all these tests I used solutions of analogous strength to working scale practice. On referring to Watt's Dictionary of Chemistry, Vol. II, page 314, line 65, it is stated, in substance, that cyanamide combines with potassium cyanate to form mono-potassium

amidodicyanate, formula $(\text{CN})_2\text{NH}_2\text{OK}$. Reflecting on this reaction, I thought probably that, as the cyanide was formed by the electrolysis of the cyanamide, it was oxidized by the current to cyanate, the cyanate then acting upon the unaltered cyanamide to form potassium amidodicyanate, and potassium amidodicyanate, when electrolyzed, forms a powerful gold solvent, although it does not give any appreciable titration with silver nitrate. It will be, I trust, appreciated that to go into discussion of these reactions would not only require an enormous amount of space, but prove too severe a tax upon my time; suffice it for the present to say that I have a number of notebooks filled with interesting data in regard to these relations, showing how they branch off and become diversified. I therefore propose later on to publish the results of the different experiments and theory relating to the data of my notebooks.

To go into practical work is more to the point of this article. However, before leaving the theory, which is always the necessary scaffolding for building up an edifice, I may state that when a solution of calcium cyanamide is mixed with a solution of alkaline ferro cyanide and allowed to stand for a few hours, and even after weeks of standing, it becomes an active solvent for gold, and without the aid of electrolysis, and when applied to the treatment of ores amenable to straight cyanidation gives results equivalent to the simple cyanide treatment, even in very dilute solutions.

Further, this process can be used on ores which have already been treated by the cyanide process—that is to say, on the residues commonly known as dumps. There exists in these dumps a large proportion of Prussian blue, which, when treated with alkali, becomes in substance potassium ferro and ferri cyanide, or soluble prussiates.

This opens up a cheap means of treating ores amenable to the ordinary cyanide process as well as ores (residues) which have already been treated by the cyanide solution—that is to say, ores from which the precious metals have not been wholly extracted, and which would not pay for re-treatment by the ordinary cyanide process on account of the cost of cyanide.

The following is an example of using this process upon ore amenable to cyanide treatment:

Take 2,000 lbs. of water containing 1 lb. of calcium cyanide, 1 lb. of alkaline ferro-cyanide and 1 lb. of lime. The ore is subjected to this solution in the ordinary way, say, in proportion of two parts of solution to one part of ore, for a period of, say, eight to ten hours, or until extraction is complete.

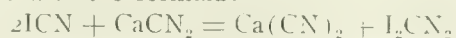
The following is an example of using the process upon ore which has already been treated by the cyanide process and exposed to atmospheric oxygen, which occasions the formation of Prussian blue and other ferro cyanogen compounds:

This ore, when treated with a solution containing 1 lb. of cyanamide, 1 to 5 lbs. of lime (the amount of lime, of course, depending upon the acidity of the

ore), in 2,000 lbs. of water, gives results equivalent to the use of straight cyanide solution when used in the proportion of two or three parts of solution to one part of ore.

Again, if the above mixture is electrolyzed, the solution of the gold is extremely rapid. Calcium cyanamide mixed with alkaline sulphocyanide, electrolyzed, immediately becomes cyanide, and dissolves gold rapidly. To show the action of another amide compound, for example, guanidine carbonate mixed with potassium cyanate, in conjunction with electrolysis of the solution, dissolves gold leaf in less than ten minutes. I might go on here and cite a whole list of amidogen compounds in conjunction with cyanate solution, if space would permit, and I have no doubt it would be interesting to those following the subject.

In the midst of this work on cyanogen-bearing materials, the thought occurred to try the combination of calcium cyanamide and iodine. I forthwith made a solution of calcium cyanamide, adding a solution of iodine to see if this combination would dissolve tellurium, and was rewarded by finding it to be an excellent solvent. I next tried gold leaf on the same combination, and had the satisfaction of seeing the gold go into solution in a short time. A scum of calcium carbonate forms on the surface, preventing more rapid solution. From this I deduced that the addition of iodine formed cyanamidogen iodide. I am unable to find any reference to this compound in chemical literature which I have read, and I therefore take the liberty of calling it by the above name, and further presume the formula to be CN_2I_2 . I next tried the action of cyanogen iodide upon a solution of calcium cyanamide, and, knowing that cyanogen iodide would be destroyed by an excess of alkali, anticipated that this combination would dissolve telluride of gold. Upon applying the solution to tellurium and telluride of gold, these substances at once dissolved, showing that the solution was probably due to cyanamidogen iodide, and I again presume the reaction to be in accordance with the formula:



WORKING OF THE CLANCY PROCESS.

I will now describe the general method of using the Clancy process on the working scale. The ore is crushed by stamps, rolls, ball mills or any other efficient form of preliminary grinding mills. The degree of fineness necessary for the process being about 100-mesh, this degree of fineness is probably most economically accomplished by the use of tube mills. The ore is crushed in the cyanide solution containing calcium cyanamide, sulphocyanide and the halogen salts; it is therefore under the influence of straight cyanide treatment practically after leaving the rock-breakers until it reaches the agitation tank. In the agitation tank it meets with electrolysis, which acts upon the cyanide containing the additional chemicals in solution, forming powerful solvents for the precious metals, as already described. Before describing the treatment solution the manner of dissolving

the cyanamide should be known. Calcium cyanamide of commerce comes as a black powder, and is about from 58 to 65 per cent soluble in water. It is necessary, therefore, to dissolve the cyanamide in a separate tank and filter from its insoluble residue. A small air-agitating tank is admirably suited for this purpose; at the same time it will act as a cyanamide storage tank. The cyanamide solution may be made as highly saturated as desired, and the calculated quantity drawn off when required and added to the cyanogen-bearing solution. The treatment solution is made up to 2,000 lbs. of water containing 1 lb. of cyanide, 2 lbs. of alkaline sulphocyanide, 2 lbs. of calcium cyanamide and $\frac{1}{2}$ lb. alkaline iodine. If using crushing rolls after the rock breakers, the product leaving the rolls at about 12-mesh is fed into the tube mill and converted into a pulp by feeding the mill with the treatment solution and ore in the proportion of one part of ore to one part of solution. The discharged pulp, after separation of the oversize, is transferred to the agitation tank to undergo electrical treatment. If it is thought necessary to remove the sulphides before, after or during the treatment, the following method presents an ideal scheme: When solution containing finely divided ore in suspension is contained in the well-known cone-shaped tank and agitated in the proportion of two of solution to one of ore, or in the proportion of three of solution to one of ore, if the agitation is stopped for a few minutes, the finely divided sulphide particles settle to the bottom of the cone, and by simply opening the discharge valve at the cone apex the sulphide may be completely drawn off, together with a small proportion of the non-sulphide pulp. This sulphide product may be run over blankets or similar contrivances and the finely divided concentrate collected, the excess of pulp solution being returned to the agitation tank for treatment. Here is the means of eliminating the use of concentrating tables and obtaining a product of very high value in a very small bulk. Again, the pulp being in a very finely divided state, the pyrite or sulphide portion is not accompanied by quartz or gangue; therefore a clean high-grade concentrate is obtained, a result not capable of accomplishment by the use of concentration tables without the employment of a large quantity of solution with its attendant expenses.

The pulp now in the agitating tank carries the correct alkalinity, this being previously established in the tube mill by adding lime so as to obtain from 0.1 to 0.2 lb. "protective" alkalinity per ton of solution.

The conductivity of the pulp is adjusted by adding common salt until the required voltage is obtained; 20 lbs. of salt per ton of solution, as already stated, will invariably decrease the resistance of the pulp so that the voltmeter will register about from 5 to 6 volts. In the majority of cases a current of about 50 amperes per ton of ore under treatment per day is adequate. It will be easily seen that the expense for electrical energy is not by any means prohibitive. With iron oxide electrodes it is possible to use a current

density considerably over 50 amperes per square foot of anode surface, so that one electrode 3 ft. in length by 3 in. in diameter will be sufficient for the treatment of from 3 to 4 tons of ore per day; in other words, approximately 30 of these iron oxide electrodes would be required for the treatment of 100 tons of ore per day. If the treatment tank is constructed of iron, the tank itself may be used as the cathode. This arrangement would, of course, decrease considerably the cost of installation. The electric generator is the chief item of cost. A low voltage generator, say, a 10-volt machine capable of giving the necessary amperage, can be obtained at any of the electrical supply houses. It is obvious, therefore, that the process may be applied to any existing fine-grinding plant provided with agitating tanks; all that is necessary is simply to introduce the electrodes into the circulating ore pulp containing the necessary chemicals and switch on the current. It is essential in every case to maintain the alkalinity—*i. e.*, protective alkalinity—at about 0.1 lb. alkali per ton of solution, so as to allow of the formation of cyanogen iodide and cyanamidogen iodide. About eight hours' treatment under electrolysis is usually sufficient to obtain the necessary extraction.

REGENERATION OF THE CYANIDE.

After the eight hours' treatment with the current, the pulp solution is brought up to about 1 lb. per ton of protective alkalinity by adding caustic soda, and the cyanide contents regenerated up to about 0.5 to 0.6 lb. cyanide per ton of solution. The regeneration of the cyanide is then accomplished simply by giving the pulp about two hours' more current. It will be understood that the reason for adding the extra alkali is that cyanide regeneration cannot take place in the presence of a halogen compound unless the solution containing sulphocyanides and cyanamide is made alkaline.

It will be seen that the whole value of the process depends upon the recovery of the halogen compound. While I have described in the examples a proportion of two parts of solution to one part of ore, a proportion of three parts of solution to one part of ore may be used with advantage—that is to say, by using three parts of solution to one part ore a much smaller amount of alkaline halide may be used per ton of solution, thus giving the same ratio of halide salt per ton of ore as in the two of solution to one of ore pulp, and consequently a less proportion of soluble halide to be displaced by the water wash in the final slimes cake.

COSTS.

In this necessarily comparatively brief or incomplete description of the process, the use of chemicals and currents have been described, but no mention of costs has been referred to. I will, therefore, take the following solution to represent the typical working solution: 2,000 lbs. of water, containing 1 lb. of cyanide, 2 lbs. of sulphocyanide, 2 lbs. of calcium cyanamide and $\frac{1}{4}$ lb. alkaline iodide. This appears a

formidable mixture when looked at cursorily, but on analysis it does not work out beyond the limits of economic treatment. For example:

1 lb. of cyanide.....	18 cts.
2 lbs. of cyanamide.....	6 cts.
2 lbs. of alkaline sulphocyanide.....	12 cts.
$\frac{1}{4}$ lb. of alkaline iodide.....	35 cts.
Total.....	71 cts.

This would not represent the total cost of one ton of solution, for, notwithstanding the effect of electrolysis, practically all the haloid salt or salts previously added, together with the sulphocyanide, will be found unimpaired at the end of the operation, the cyanide and cyanamide alone suffering the necessary decomposition. It is clear, therefore, that no matter what the proportion of solution to ore, only the consumption of cyanide and cyanamide per ton of ore is to be taken into account. The amount of cyanide consumption on the ore in presence of cyanamide works out at about 1 lb. of cyanide per ton of ore treated. This consumption of cyanide is regenerated at the expense of 3 cts. for cyanamide and, at the very outside, 3 cts. for current (figuring current at 1 ct. per kilowatt hour), making a total cost of 6 cts. per ton of ore.

Added to the above cost is the cost of the electrical energy necessary for the electrolysis of the ore pulp. The cost of electrical energy for this purpose works out at about 10 cts. per ton of ore treated; this, added to the cost for cyanogen-bearing material and regeneration, would make a total of 16 cts. per ton of ore. These figures would represent the total cost, provided that all the solutions were recovered without mechanical loss. From this it is evident that the recovery of the solutions for re-use is a matter of vital importance to this process. The mechanical recovery of the solutions is, therefore, entirely dependent upon the efficiency of the filter employed.

The Moore filter eminently fulfills the process for the recovery of the solution, inasmuch as it gives perfect uniformity of cake both in thickness and porosity, which means perfect resistance, and perfect resistance guarantees perfect displacement. In fact, the business combination of the Moore filter with the Clancy process was originally brought about not from any arbitrary or usual business considerations, but because I, after the most rigid examination of all existing filter processes, learned in my demonstrations that this type of filter was the only one that could be depended on to recover practically all of the solutions.

This filter process is too well known to describe minutely; suffice it to say that when the final step in the cycle of treatment is reached—that is, when the cake has been washed with barren solutions—the moisture (containing soluble salts) saturating the cake at this juncture may be completely water-washed by giving it the requisite amount of water for displacement. The necessary amount of water for displacement is readily seen without calculation by the diminution of the water level in the wash tank; further, this can be adjusted to a nicety at the will of the op-

erator, obtaining results not capable of accomplishment by any other known filter. It has been necessary for me to mention the unique points of this filter so as to show how the solutions can be recovered with a minimum of mechanical loss and with a minimum of water wash. In a properly constructed Moore filter, Type A, the loss of solution under proper manipulation should not exceed 10 per cent of the total solution.

It should be understood that the above figures are based on the treatment of rebellious or refractory ores, and that they would in all probability be much reduced when dealing with non-rebellious ores.

ECONOMY IN CUPOLA MELTING.*

By J. W. HENDERSON.

The mere melting of iron in a cupola is a simple operation. To get the highest efficiency out of raw materials and the necessary equipment, and convert these raw materials into metal most suitable for the work in hand, is another matter.

Although not fully realized by foundrymen in general, there is more to the cupola melting of iron and to economic production of castings from cupola metal than the mere bringing together in a cylindrical stack lined with bricks, certain quantities of fuel and iron at varying temperatures.

Primarily the engineer is interested in the plant arrangement and equipment and their relation to economic production. As the engineers' work consists in planning the relation of the parts of the foundry to the whole scheme, with a view always to efficiency, he must be interested in all factors that assist or retard this result. In this discussion we are considering factors of economy involved in the conversion of pig iron, scrap, etc., into good castings. Examination of a large number of plants in many states, discovers some of these important factors as follows:

1. Relation of the receiving track to the ground and building plan. This is important on account of required storage room for raw materials, space for scales and narrow gauge yard tracks, and means for properly handling the charges of fuel and iron.

2. Weighing facilities that will require careful weighing of each complete charge as a separate unit. This is a sadly neglected part of the equipment in most foundries. A few of the general practices will be mentioned later.

3. The elevator should be placed, in relation to the position of the cupola, so that the maximum loads of fuel and iron charges can be handled with the least interference with the operation on the charging floor.

4. The charging room capacity in small foundries should be sufficient to carry the charges for the total heat, and in larger foundries adjusted so that the immediate cupola laborers will need the minimum assistance.

5. Cupola details. Those of importance are:

Extension of the spout into the foundry.

Height of spout above foundry floor.

Depth of cupola from sand bottom to charging door.

Dimensions of charging doors and the distance between the doors and the charging room floor.

Capacity of the cupola, when lined to its largest inside diameter with safety, always in advance of shop requirements, and with blower or fan capacity and adjustment within a range that will permit the melting speed to be as elastic as the needs of the shop might demand.

These engineering details of plant arrangement and equipment not only influence economies but they also assist or hinder systematic operations and accuracy.

Many years' experience convinces that those engaged in chilled iron work, such as cast iron car wheels, and those making malleable iron from the cupola are to be credited with most of the present development of cupola practice in this country. This has not been on account of their greater intelligence, necessarily, but rather a matter of compulsion. For example consider car wheel iron. When the metal is running close to the lowest limit as to chilling qualities, if the silicon content increases .05 per cent the chill on the wheels will drop below the specification limits, and the wheels made therefrom will be bad so far as their acceptance by the railroads is concerned. If one such wheel among a lot of a hundred is found by the inspector, he will reject the whole lot even though the other ninety and nine might be known to be all right in every way.

Cupola melting of iron is simple until you give personal attention to the actual running of a cupola and try to produce accurate results as to temperature, composition, chill, strength, shrinkage, etc., in any given kind of castings. Some varied experiments may be of interest.

Cupola A. Sand bottom to charging door, 17 ft.; diameter straight lining, 45 in.; diameter of melting zone, 38 in. Cupola started with 1500 lb. bed coke for 1500 lb. first iron charge. This was varied to 1800 lb. bed coke and 2000 lb. first iron charge but good, very hot, iron was not obtained under good control until the bed coke was made 2000 lb. and the first iron charge increased to more than double this or 4400 lbs. Afterwards it was common practice to take out of this cupola at each heat four different mixtures varying in silicon from .75 per cent to 2.50 per cent. Although no extra coke was used to divide them, the four mixtures were produced each day, with the desired differences in composition and separated from each other.

Cupola B. Diameter varied from 48 in. to 69 in. and depth from charging door to sand bottom was 9 ft. 6 in. Weight of pig iron was taken by considering an average weight for the pigs and making up the charges by counting them. Weight of the scrap was left entirely to the guess of the head charger.

*From the Proceedings Engineers' Society of Western Pennsylvania.

Charges were supposed to be 4500 lbs. each and were so shown on the records, but on checking up the work being done, and allowing 50 lbs. as the average load carried each trip of the men from the scrap pile to the cupola, it was found that the charges would vary from 3500 lbs. to 4200 lbs. The coke was forked into the cupola until it reached a measured distance from the charging door. At 25 lbs. per fork found that the total bed coke would amount to 3400 lbs. After taking up melting on the chemical basis these conditions were changed.

Cupola C. Making very thin and small special castings requiring very hot iron of uniform composition. Cupola was 50 in. diameter and 14 ft. in depth. Coke and iron charged in barrows. At one time they found the average barrow load of coke to weigh 120 lbs., so this was still being used to determine quantities charged. When on hand to check up the actual conditions was told by the man in charge that the bed coke each day consisted of 1800 lbs., made up from 15 wheelbarrow loads. Weighing all barrow loads discovered that the total bed actually weighed 2600 lbs. Upon this was put iron charge of 2500 lbs. Single charges of coke varied throughout the heat from $3\frac{1}{2}$ to 5 barrow loads. The product varied in quality with the "luck" which they had in striking the right conditions. Today this shop is securing very hot iron of uniform quality continuously.

Cupola D. Making special chilled work. Melting rate was too slow for the enlarged shop. Cupola 58 in. diameter. With single charges of 4000 lbs. each the output was 8.5 tons per hour. Increasing the charges of coke and iron, but reducing the then coke charge gave a melting rate of 9.5 tons obtained per hour. Thus hotter iron of more uniform composition was obtained, at an increased rate of 1 ton more per hour. The best results were obtained with the iron charges amounting to 7000 lbs. each. It will be well to say here that the character of the raw materials, the diameter of the cupola, and means for mixing the iron after it is melted, must be considered in determining the size and weight of the iron charges.

Referring again to conditions similar to Cupola B given above, I will state that where any attempt is made to make up the charges by weight the proceeding is usually somewhat as follows: Scrap, and the iron by brands, are taken up an incline in wheelbarrows, and piled in the charging room. From these the charges are made up piecemeal on a scale too small to hold a complete charge, and the assembled charge is again piled on the charging floor. Once more this metal is handled and thrown into the cupola through a door that is above the waist line of the men. Altogether the metal composing the charge is handled five times at least.

The coke is also taken into the charging room by barrows and from the piles thus formed is charged by wire, or solid metal basket, or by fork. It is a common thing to find records being kept that show 45 lbs. as the basket load when 40 lbs. is correct, or vice

versa, and 18 lbs. to the fork when 25 lbs. should be the weight shown.

The blower and the fan. Here is the case of a shop melting 400 tons per day. The shop had been increased so that the requirements were beyond the capacity of the power plant, and it was necessary to enlarge the latter or conserve energy at some point. The No. 10 fan was thrown out and a blower put in its place, with the result of saving 35 h. p. and still being capable of doing more than was being accomplished with the fan. There is another case of a continuous shop driving fans to their fullest capacity, and having considerable trouble to keep belts on them. Here the blower saved over 30 h. p. besides removing all belt troubles.

To accomplish the best results, the mixing by chemical analysis is not merely the province of a chemist. As a matter of fact, where the chemist is only a chemist and not a foundryman, and no one else in the shop is acquainted with chemistry or metallurgy, the chances for definite improvement are doubtful.

This will account for the sad experiences of some few of the foundrymen in this district. There is in mind one proprietor in particular who claims that the entry of a chemist into his foundry cost him \$10,000. Where the chemist, who is also an experienced foundryman, has not the co-operation and assistance of the officials and those in direct charge of the cupola and foundry, the results from mixing by chemical analysis will be negative. The chemistry part of it bears about the same relation to the results that the bookkeeping in the establishment bears to the profits at the end of the year.

Again, the chemical analysis of the materials may be perfect, and the weighing of the charges may be accurate, yet the resulting metal from the cupola may be made almost worthless in the operation of melting. Or if these things are carried on as they should be, still the castings may be had from the way the metal is handled and poured, and in special cases in the manner of cooling or shaking out, or in the annealing of the castings.

To the practical chemist and metallurgist the "brands" of iron, as brands, mean nothing except as the name stands for composition and structure. Notwithstanding demonstrated facts which disprove the following claims, it is not an uncommon thing to hear foundrymen express themselves as follows: "Don't the steel go into the sand bottom?" "Don't stove plate go up the stack or out with slag?" "We have to have charcoal iron in our castings in order to get the chill and strength." "Nothing will close up the grain in cylinder castings like old car wheels." And so it goes with the foundrymen visiting conventions, listening to and reading the educational papers, still clinging to the ways of the past century. It does not make any difference what kind of castings you are making you cannot today get away from the benefits derived from chemistry and metallurgy, and generally better castings at lower cost will be made by employing these

in your foundry. Blast furnace practice is not what it was twenty years ago. In those days if the silicon percentage dropped a given number of points the chemist on the particular furnace output could know that there would be an increase in the sulphur content and almost what the percentage would be. A few years later the same blast furnaces were making low silicon iron containing lower sulphur than was thought possible from coke iron furnaces.

Whether making stove plate or ordinary machine castings, automobile cylinders, semi-steel castings, or any other kind, after once determining the composition and at what temperature the best of these has been made, the desire should be to duplicate them continuously. You cannot be confident of duplicating castings today in and out, and take advantage of the market changes in iron and fuel, without chemistry and metallurgy.

Here is the analysis of six pigs taken from the same car load of a well known brand of iron: Silicon, .87, .99, 1.62, 1.70, 1.79, and 1.82 per cent. Chemistry obtained a rebate on this iron, enabled the foundry to use it without loss of castings and prevented future shipments of such mixed iron from the same source to this foundry. Without chemistry the resulting bad castings would have been charged to the coke first, and if that was found not guilty then to anything but this particular brand of iron which had been used theretofore without loss of castings attributable to it.

Knowing these things and that there are frequently even greater variations in analysis between different car loads of the same brand of iron, it is not difficult to realize the lack of uniformity in the castings produced where several hundred car loads of iron are thrown together in one pile. When a number of such piles are found on the property of concerns (generally favorable to scientific methods) the marvel increases.

To be successful, mixing by chemical analysis must be supplementary to that experience which is familiar with general cupola practice and the effect on the results desired, of every operation from the raw materials to the loading of the finished castings for shipment. Given the chemical analysis and applying the personal attention, many foundrymen might work out the results themselves. System in the laboratory has brought the cost of analysis within the reach of even the smallest foundry.

As indicating what is being accomplished, attention is called to the analysis of test pieces from three plants in different parts of the country, shown in Table I.

While maintaining this uniformity the castings are being produced at lower mixture cost than before and the casting losses chargeable to the metal are reduced to a minimum. One more point is that of a company which at the beginning of the financial depression found itself with several hundred tons of iron on hand not suitable for the castings to be made. Business policy dictated buying as little iron as possible at that time and this was accomplished by the chemical

analysis indicating just what the minimum amount of iron would have to be composed of to dilute the elements which were too high and what to make up for those elements which were too low, in the stock already on hand.

Advancement in any branch of the iron industry moves in two lines generally parallel to each other though often converging, the one scientific and the

TABLE I.

Test piece	Silicon.	Manganese.	Sulphur	Phosphorus.
No. 1.....	2.39	.57	.088	.632
No. 2.....	2.36	.58	.076	.612
Same plant, same kind of castings, one year later.				
No. 1.....	2.25	.53	.074	.636
No. 2.....	2.23	.59	.061	.632
Foundry No. 2, Combined Carbon.				
No. 1.....	.55	.53		.61
No. 2.....	.52	.57		.59
No. 3.....	.51	.51		.60
Same plant, same chilled work, several months later.				
No. 1.....	.49	.57		.60
No. 2.....	.55	.61		.54
No. 3.....	.55	.49		.58
Foundry No. 3.				
No. 1.....	2.51	.16	.070	.720
No. 2.....	2.44	.49	.080	.690
Same plant, same kind castings, one year later.				
No. 1.....	2.42	.51	.067	.778
No. 2.....	2.32	.49	.074	.780

other practical. With cupola melting considered so simple that any man with a few hundred dollars can engage in the business with hope of reward, and with cupola practice in the chaotic condition in which we find it today in even many of the large foundries, advancement in economical melting along either scientific or practical lines needs necessity to give it movement.

Fundamentally, the small manufacturer cannot take advantage of, and the large one will not introduce, the latest methods affecting economies or a higher standard, worked out by the application of science to practical purposes.

Neither will concede the time to give the application of new methods the personal attention they require and deserve. Foundrymen will run after some new improvement in cupola design, or the suggestion for changes in the size and weight of the iron and fuel charges, when the same amount of time and personal attention given to the devices and materials they now have would yield large returns in economies to be added to the profits at the end of the year.

The county of Welland and adjoining county of Halimond furnish most of the natural gas in the Province of Ontario, Canada. There are about 500 producing wells, some of which furnish 1,500,000 cu. ft. daily. The depth of the wells range from 550 to 3,000 ft. Prices for gas are 15 to 45 cts. per 1,000 ft., according to quantity used. Gas is piped to Hamilton, Niagara Falls, Ontario, and as far west as Brantford and Paris. Farmers and country villages are supplied at retail prices, while ironworks, smelting works, and other manufactories in Welland are supplied at wholesale rates. It is estimated that over 30,000,000 ft. are produced daily in this district.

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HARRY McCORMACK, Editor.

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NOTES AND COMMENTS.

Industrial Chemistry and Industrial Fellowship.

We published in the March CHEMICAL ENGINEER an excerpt from the address by Dr. Duncan on the relation between chemistry and manufacture in America. The article is an admirable one and calls attention to many of the faults in our present arrangement. It also points out remedies for some of these failures. It seems to the writer, however, that Dr. Duncan fails to make one point which is fundamental in the collation of chemistry and manufacture in any country. That is, that there must be men handling the process who have at least a working knowledge of both chemistry and engineering. No process can be transplanted from the laboratories to the factory without engineering difficulties intervening. It is on account of this lack in engineering training on the part of our industrial chemists that we have such a high death rate among our young and tender chemical plants. The research chemist, however admirable he may be in his own field, is too frequently not familiar with modern engineering practice and fails entirely to grasp the relation between plant construction and the success of the industry. The design of the plant is too many times left to someone who does not know the requirements and the operation of the plant; too often this is an architect, a civil or a mechanical engineer who does not have the intimate knowledge required to bring

about the proper sequence of the chemical operations. The most skilled research of the best chemist is a failure if his results are placed in untrained hands for execution.

Dr. Duncan's fellowships are admirable in conception but too often the results attained under them depend on securing properly trained men to carry on the work. It is the writer's opinion that the fundamental training of these men is of at least if not more importance than the proper provision for other research. This is pointed out very clearly by Prof. Whitaker by his remarks at the conclusion of Dr. Duncan's paper. He says in part: "One of the most important things to be considered in the education of the chemists for manufacturers is the fundamental education.

"Equally important and essential to the young chemist in making himself indispensable to the manager is a knowledge of certain other subjects, such as works equipment, works organization, works labor, and works economics.

"The margin between a profit and a loss in a chemical manufacture is often very small and no matter how perfectly the chemical reaction may be developed, a poorly equipped works may convert it into a losing venture. Again, a losing chemical process may be, and often is, converted into a dividend-payer by improving and developing the organization for its execution."

Dr. Duncan's remarks on the question of the instructor in chemistry engaged in industrial or commercial work seem to the writer to be based on an erroneous conception of what this work usually amounts to.

A somewhat wide acquaintance with the men engaged in teaching chemical engineering and industrial chemistry would lead the writer to make the statement that the live wires in the profession are just the men who have extensive outside interests and that it is these direct commercial interests which keep them keenly alive to the conditions and opportunities in the field of industrial chemistry. In so far as the work of these men "taking the bread out of the mouths of their professional brethren," the writer has never heard any complaint about this. The work done by the instructor is generally of such a nature and needs such an equipment that a commercial chemist is very glad to escape from the necessity of doing it. As to the securing of remuneration, these men usually demand and receive higher pay for their work than the commercial chemist so that the work comes to them as a matter of selection not of cost.

The industrial fellowships of Dr. Duncan fill a need in the chemical industries when the industry is not large enough to maintain a properly equipped research laboratory for its own competent chief chemist. The writer believes, however, that the large companies with their own force and laboratory under their own supervision are receiving the service much more often than they could obtain it in any other way. They also have the satisfaction of knowing that the results attained

are their private property. In the nature of the case no college or university could maintain such a laboratory for special processes as those maintained by some of our largest industries. With all this in mind, however, the writer knows that in some very large industries the research work is as Dr. Duncan says, a failure; primarily on account of incompetent direction. He cannot say, however, where the establishment of industrial fellowships will make the executives of any corporation infallible in the selection of the proper chemist to direct their research work.

Dr. Schott Visits America.

The chemists of America have rejoiced recently in the visit which Dr. Schott of the firm of Schott & Genossen, has been making this country. Dr. Schott and his wife are now on a combined business and pleasure trip through the United States, and it has been the good fortune of the chemists of Chicago to have recently entertained the Doctor and his wife and to have heard from him an account of the founding of the great German Glass Works bearing his name. The account of his early failures and success is certainly an inspiring one to the young chemists of today. No one could have met and overcome greater difficulties than those under which he labored during the beginning of his industry. The present success of the firm bears witness to the successful termination of the early research work as well as to the business ability of the men conducting the industry.

THE CHEMISTS' BUILDING, NEW YORK CITY.

The Chemists' Building, which was inaugurated on March 17th at 50-54 East 41st Street, New York City, is believed to be the first of its kind in the world. It combines the features of a first-class club, including restaurant and members' bedrooms, with finely equipped laboratories for analytical and consulting chemists and for investigators in pure and applied science, not to speak of a carefully planned scientific lecture-room, a large library and scientific museum.

The building occupies 56x100 feet and cost upwards of half a million dollars. The ground floor is devoted to a public entrance hall, as well as to the general foyer of the club, back of which is found the lecture-room, decorated in classic design, whose lecture platform is fully equipped for chemical demonstration. The next story contains the social room and restaurant, and above this are the library, the museum and the trustees' room, and two floors devoted to bedrooms. The five laboratory floors are arranged in such a manner that each laboratory has light on at least two sides, and has its own ventilating ducts and mains for the supply of gas, water, electricity, steam, compressed air, and suction. The floors are fire, water and acid proof, and nothing has been omitted which is found in the best-devised scientific laboratories of educational institutions.

The building is owned by a stock company, whose

shares have been taken by prominent chemists and by individual manufacturers and companies whose business largely depends upon chemical process, and who have realized that industrial progress depends upon scientific research. While it is expected that the building will be more than self-supporting, the shareholders limit themselves to 3 per cent dividends, all surplus to be devoted to the betterment of the building and to the ultimate benefit of the science. The Directors of the Corporation are Dr. Morris Loeb, formerly Professor at New York University; Dr. Charles F. Chandler, Emeritus Professor of Columbia University; Mr. Albert Plaut, of Lehn & Fink; Mr. William H. Nichols, Jr., of the General Chemical Company, and Dr. Leo H. Baekeland, of the General Bakelite Company. Messrs. York & Sawyer are the architects and Messrs. Marks & Woodwell the consulting engineers of the building, which was erected by the W. L. Crow Construction Company; Charles L. Eidlitz & Co. are the electrical, and Evans, Almirall & Co. the heating contractors; the Otis Company furnished the elevators and Black & Boyd the lighting fixtures.

The chief tenant is the Chemists' Club, which occupies the lower half of the building. The five laboratory floors were nearly all let before their completion, some of the tenants being Professor E. E. Smith, Professor Morris Loeb, Dr. William M. Grosvenor, Mr. Oscar L. Palmenberg, Messrs. Eimer & Amend, Mr. S. R. Morey, the Scientific Station for Pure Products, Messrs. York & Sawyer.

The Chemists' Club is a social organization, whose president is Dr. R. W. Moore. It was founded twelve years ago to provide opportunities for social intercourse to its own members and a meeting-place for the New York Section of the American Chemical Society, and before taking new quarters in the Chemical Building, occupied the old Mendelssohn Glee Club room on West 55th St., New York. It has recently grown very fast, and the number of professional chemists has nearly quadrupled in the past 20 years in this vicinity. The American Chemical Society has a larger membership than any similar society elsewhere, and there also flourish the Society of Chemical Industry, the American Electrochemical Society and the Institute of Chemical Engineers, all of which held frequent meetings in the hall of the Chemists' Club and which migrated with it to the new building.

To express the manifold purposes to which the structure is to be devoted, an elaborate program was arranged for the opening exercises, extending from March 17th to March 19th. At the opening exercises on March 17 the speakers included Dr. Edward S. Morley, as Honorary President of the approaching Eighth International Congress of Applied Chemistry; Professor Alexander Smith, as President of the American Chemical Society; Professor William H. Walker, President of the American Electrochemical Society; Professor Wilder D. Bancroft, of Cornell University, and Professor Frank W. Clarke, of the United States Geological Survey. At the conclu-

sion of the addresses Dr. R. W. Moore accepted the keys of the clubrooms on behalf of the Chemists' Club.

BOOKS RECEIVED.

The Corrosion and Preservation of Iron and Steel. By Allerton S. Cushman and Henry A. Gardner. McGraw-Hill Book Co., New York.

A publication which is of interest to the engineer and practical chemist alike is the "Corrosion and Preservation of Iron and Steel," by Dr. Allerton S. Cushman and Mr. Henry A. Gardner. Both authors have contributed so much to the current literature of recent years regarding these matters, that the book will be welcomed by all who are interested in this field. Dr. Cushman has long been a prominent worker in investigations of the natural influences which produce oxidation of iron and steel, and he has done much toward the conservation of these valuable materials by his excellent studies of the prevention of rust formation. Mr. Gardner, on the other hand, has been very active in the study of paints and pigments, and no one is better fitted to write on these matters than he is.

Some of the material which makes up the book has previously appeared in bulletins and magazine articles written by one or the other of the authors, and some has been already published in committee reports of various technical societies. Nevertheless there is much that is new, and great pains have been taken to explain the many laws and principles of physical chemistry that underlie corrosive action. In the Introduction it is stated that one of the objects of the work is to treat these matters in such a manner as to render them clear to manufacturers, engineers, metallurgists, and others who are not familiar with the theories involved. The difficulties arising from an attempt to attain this end are plain to any student of the subject, but in the main very good results have been attained for a work of this size. In a few instances, however—and especially in quotations from other authors, with which the book is replete—material has been brought out which will not be comprehensible to those who have not made an extended study of physical chemistry, and in such cases a little further explanation might profitably have been appended. In most cases, where such added explanation would have proven too bulky to include in this brief work, all mention of the matter involved might probably have been omitted without lessening the value of the book to the majority of readers. For example, in the fifth chapter a quotation from Walker is included, which mentions the silver voltmeter and Ostwald's bromide voltmeter. It is hardly to be expected that the practical man, aside from a few engaged in electrical work, is familiar with an apparatus of this nature, which finds so little application outside the research laboratory.

In other instances analogies have been drawn between the phenomena involved in corrosion and cer-

tain common phenomena familiar to all practical men, for the purpose of making more plain the various theories which are expounded. This practice may be useful at times, but it is likely to produce undesirable results, and false impressions are likely to be brought about in the minds of the uninitiated, which no amount of further explanation will succeed in removing. The unwary beginner in the study of a theory is oftentimes led to extremes in the application of such similes, by his enthusiasm. For example, on page 14, the "driving force" of solution pressure or solution tension is said to be exactly analogous to steam pressure in a boiler, and further on it is stated that as the rising steam pressure tends to resist further evaporation of the water, so the growing number of free particles entering into solution produce a back pressure which tends to resist the entrance of more. The use of this particular example might be excused on the ground that Arrhenius, the father of the electrolytic theory, made use of it also, and that numerous authors have since done likewise. Still, the false impression may be formed in the mind of the beginner that the walls of a vessel of water into which a mass of metal is introduced, are instantly subjected to a bursting pressure due to the solution pressure of the metal. Even though the Helmholtz "double layer" idea be brought out afterward, as is done in this work on page 27, still there remains this unhappy impression, which serves to puzzle the reader in his further study of ion formation.

The first part of the book is given over to a discussion of the problems of corrosion, the various conditions and influences which stimulate and inhibit corrosive action, and the theories which have been advanced to explain the phenomena involved. While the carbonic acid and the peroxide theories receive some attention, greater stress is laid upon the electrolytic theory, and in connection therewith a general exposition of the electrolytic dissociation theory of solution is given. The latter part of the book deals with paints and pigments and preservative measures in general. This portion of the book is written in a very practical way and no user of paints can peruse it without obtaining many valuable ideas.

Considered by chapters, in the order in which they appear, the contents of the book are as follows: Chapter I treats of the corrosion problem in general, and the more common causes of rust formation. Chapter II gives a very good explanation of the electrolytic dissociation theory of solution, which could hardly be improved upon except perhaps by the introduction of a paragraph on the law of mass action and its application to the explanation of hydrolysis. With this addition the theory of indicators might have been brought out more clearly than has been accomplished, by further applying the law of mass action in this explanation. However it is hardly to be expected that these matters could be treated in an exhaustive manner in a work of this size. Chapter III deals with the various theories of corrosion, and gives an

outline of the evidence which is at hand to prove and disprove each one of them. In connection with these matters the very excellent work of one of the authors with the "ferroxyl reagent" is described at some length. Chapter IV treats of the application of the electrolytic theory to the rusting of iron. If, after the third chapter, there still remains any doubt in the reader's mind as to the strength of the author's contention in favor of the application of this theory to rust formation, all such doubt is removed by the fourth chapter. The influence of the constituents of iron and steel on their corrodibility, and the effects of heat treatment on the corrodibility of the product are also given in this chapter. Chapter V deals with inhibition and stimulation of corrosion and the various factors entering into each of these actions. Chapter VI is taken up with the practical application of some of the principles which are brought out in the preceding chapters, and gives a detailed description of the methods of manufacturing galvanized iron, together with an account of various tests of galvanized iron and tin plate materials.

In Chapter VII the question of pigments is taken up, and the tests conducted by the American Society for Testing Materials on the inhibitive action of pigments are described. Chapter VIII describes the field tests of paints made at Atlantic City by the same society, in conjunction with the Paint Manufacturers' Association. In Chapter IX special paints are discussed, especially bituminous paints, while Chapter X takes up acceleration tests of paints, and the design of special paints, several formulas being given for the composition of inhibitive mixtures. Chapters XI and XII give a very good discussion of the properties of pigments and paint vehicles, including a description of the various desirable and undesirable properties of paint materials. The processes of manufacture of the more important pigments and vehicles are also described in these closing chapters, including several that are used for the painting of materials other than iron and steel. The remainder of the book, consisting of nearly one hundred pages, is taken up by an extract from the Transactions of the American Institute of Mining Engineers on the corrosion of the water jackets of copper blast furnaces, and by the complete bibliography of metal corrosion and protection which was issued by the Carnegie Library in July, 1909.

Taken as a whole the book is altogether the best work on this important and involved subject that has been issued thus far, and may be read with much profit by all those who wish to be well informed along these lines.

C. L. PARKER

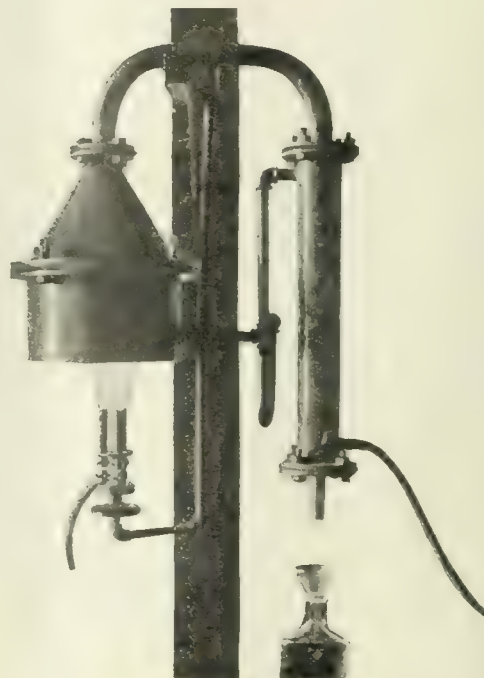
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THE ENGINEERING SCHOOL GRADUATE: HIS STRENGTH AND HIS WEAKNESS.*

By HENRY P. TALBOT.†

So much has been written and spoken of late concerning the success or failure of the various engineering courses in our schools of technology that a reason should be offered for the introduction of this topic at this time. It is to be found, I think, in the general and increasing interest in these matters which is leading the practicing engineers, the manufacturers, the men of affairs, in short, the consumers of the product of the engineering schools, to examine and evaluate the work of these schools. This interest has voiced itself more and more freely in the daily press, the engineering journals, and the occasional address. Some of the comments thus made are harshly critical, some are based upon sadly insufficient knowledge of existing conditions, but many are sane and helpful. It is the duty of those of us who are charged with the conduct of these courses to give heed to these utterances and to avail ourselves of the helpful counsel which many afford; but it is also a privilege which we may sometimes allow ourselves to present the case as it appears to us, and this anniversary occasion seems to suggest both retrospection and introspection.

The complexity of the educational problem is nowhere greater to-day than in the training of the engineer, using that term in a broad sense to include the man who applies his science to concrete ends, whether he be, for example, civil engineer, research chemist or field geologist. The boundaries of all the sciences have been extended at a rate which has far outstripped any reasonable alteration of educational methods to meet these changing conditions; for, over against the charge of undue conservatism which is commonly made with respect to educational practice, should be placed the fact that seven years is the minimum period which must elapse before the ultimate success or failure of an educational experiment can be determined, and since the remodeling of a course or system of instruction to utilize successfully such of the new acquired knowledge as it is possible to include must often be the result of gradually accumulated experience, it is plain that rapid and frequent alterations are both

unwise and unprofitable. Such advances in scientific knowledge, as for example, those relating to wireless telegraphy, the turbine engine, or aeroplanes, which are of such immediate significance as to seem to imperatively demand a place in our courses of instruction, cannot be allowed to displace other older topics of permanent importance, and in many cases these later developments of science are based upon abstruse principles, the proper teaching of which in turn demands much time.

The educational problem has moreover been rendered more difficult of solution by the concomitant increase in the number of men to be educated. It is no longer possible to give to the undergraduate that measure of personal attention from a mature teacher, of strong personality, which characterized successful teaching in the young manhood of our fathers, and resulted in the production of what may be termed "hand-made engineers." And, again, the increased ease with which our young men can now obtain educational advantages is sending to our schools a much larger proportion of students who, while they are earnest to a high degree and constitute a most desirable class of pupils, have not descended through generations of ancestors with scholarly or scientific instincts, and have not been surrounded by an atmosphere which is at all closely in harmony with that of the lecture room or laboratory. That most of these young men meet with success is the more to their credit; that some others meet with only measurable success in the scientific professions, and that distinct limitations, both professional and social, manifest themselves in the post-graduate development of some, is not surprising; but the cause is often mistakenly ascribed to faulty educational method when in truth it is far more a question of raw material than of manufacturing process.

The product of the engineering schools has not escaped the universal demand that all products should advance in quality without increase in cost, which, in this instance, means with little or no increase in time expenditure. One needs only to review the conditions of the last quarter century to realize that an extraordinary change has taken place in the position of the engineer in the community. None of the older professions have been called upon to face such kaleidoscopic conditions and it is not strange that there should be a dearth of men immediately adapted to meet the altered situation, or that many should be found to be

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*Paper presented before the Congress of Technology at the 50th Anniversary of the Granting of the Charter of the Massachusetts Institute of Technology.

partially lacking in the extremely composite training which would lead to complete command of the field. It may not be irrelevant to ask whether the so-called learned professions, so long regarded as superior to the engineering professions, would have fared distinctly better under a like extreme test.

The wholly successful engineer of the day (I do not speak now of the recent graduate) must be a man possessing a capacity for logical, quick and exact thought; a detailed knowledge of some portion and a broad knowledge of the whole of his professional field; and be master of a certain amount of the technique of his profession. He must have the ability to select and guide competent and trustworthy associates and to obtain from them loyal and willing service. He must be strong in his sympathies and generous in his public services, and while quick to enlist desired interest in his enterprises he must be shrewd in detecting avarice or perfidy. He should be a loyal husband and father, and should find opportunity for that enjoyment of art and literature which will afford him present pleasure and ensure the happiness of advanced years. It is a matter for sincere rejoicing that the engineering profession has reached such a commanding position in our national life that only a man of this type can completely fill it; but the imperfect portrait just drawn is evidently that of a man for whom Nature must have done much at the start, and toward whose efficiency many elements must have contributed. Of the need of such men there is no doubt, and it becomes a question of paramount importance to ask how far the engineering schools, as such, or, indeed, how far our entire educational machinery can contribute to the desired end. The most obvious function of the engineering school is to afford a fundamental knowledge and understanding of the principles of the sciences underlying engineering operations. Failure to do this seems to be without excuse, yet it is almost inseparable from another important function, namely, the development of the power to think (for there can be no adequate understanding of principles unless one can think logically in terms of them when considering concrete problems), and it is just at this point that much of the current criticism is aimed. The candid teacher must admit that there is truth in the charge that the graduates are too often lacking both in a capacity for logical thought and in an ability to command the knowledge which they actually possess to the degree needful for immediate or perhaps ultimate success in their vocation. But it should not be supposed that he is indifferent to this state of affairs. It is within bounds to say that it is the supreme desire of every worthy teacher to encourage power of thought rather than mere acquisition of knowledge on the part of his pupils and that he is constantly devising and testing new means to that end, but a moment's consideration will show you how much this depends upon personal contact—now so difficult in even the smallest practicable subdivisions of large classes—and will convince you

that there must also be constant conflict of judgment as between the extent of the field to be covered in a given subject (rarely more than the minimum quantity nowadays) and the time which can properly be spent in that drill which is necessary to develop the powers of the average student; for it is against the average student that the criticism is most valid. I do not make these statements to condone the conditions but rather to show you that the teachers recognize them, deplore them, and are striving against them, but the contest is an unequal one, at best.

Let it be remembered, moreover, that some responsibility for these conditions rests upon our public school system, and also that the sort of thinking which the engineering professions demand is of a kind which is more exacting than is essential in the more common vocations, and that *no* system of education has yet succeeded in training a large proportion of exact thinkers, however much such a result is to be striven for. Let us also admit for our encouragement that, after all, there is a considerable proportion of our engineering graduates who *can* use their brains effectively and do have their knowledge in available form, and my observation leads me to believe that there is a much larger proportion who appear deficient in these respects at graduation who develop unexpected power when they have opportunity to concentrate their efforts in a more limited field. Remember that many of these youths have been in some sort of educational training for a continuous period of 15 to 17 years, during which there has been a constant (but sometimes unwise) increase in the pressure put upon them to cover more ground. Is it strange that they have lacked an opportunity to sort their immense stock, or to become familiar with it? They are, I think, entitled to charitable consideration for a time after entering their vocation, but if as a class they are deficient after three years, the criticism of them or of their training certainly becomes valid.

The public has a right to look to the engineering schools for sound instruction in fundamentals including, of course, physics and chemistry, as well as the mathematics and drawing which must form a part of the equipment of every competent engineer. In addition, they may demand that the fundamental principles and something of the technique of those subjects which are of general application within a given profession shall be thoroughly taught, and that this shall be done with reference to development of power and the inculcation of useful habits, rather than the mere acquisition of information. While this is a demand which no engineering school would desire to evade, let us recognize that it is, of itself, no light task to accomplish.

But in our epitome of the distinctly successful engineer of maturer years was included *breadth* of knowledge within and without his profession, the quality of leadership, which means power of imitation and a knowledge of men, and the ability and inclina-

tion to fulfill the requirements of good citizenship. Are the graduates from the engineering schools, as a class, in line to develop thus symmetrically? Let us admit again that many are not and that that is the occasion of the general charge of "narrowness" and inadequacy which is directed against our courses. But here again I venture to assert, not however in a spirit of complacency—that the situation is more complex than is generally admitted, and that there is a good deal that is encouraging in the situation. Recall once more how short a time it is since the engineer has occupied a position in the community which is recognized to be of equal dignity with that of the so-called learned professions, and recall how recent is that evolution of our industrial system, which has as its most important feature the recognition of the fact that the engineer and the financier, if not combined in the same individual, must be on a parity with respect to influence and authority if efficiency—the watchword of the hour—is to result. Is there not cause for congratulation that some have been found in the engineering ranks capable of meeting this surprising increase of responsibility rather than ground on which to pronounce the general result of engineering education a failure, as some seem inclined to do?

It is well known that the Massachusetts Institute of Technology endeavors to stand today, as it has from its beginning, for the largest measure of breadth of training and education which is compatible with thoroughness of fundamental scientific instruction. An inspection of its courses as prescribed for the various professions shows, that, notwithstanding the pressure resulting from the growth of science and technology about one-eighth of the total hours which a student spends at the institute is devoted to subjects which are cultural studies, using that term to distinguish them from those scientific subjects which may be regarded as tools of trade, although many of these, notably such as physics, chemistry, biology or modern languages, if properly taught, will contribute much to the cultural development of the well-rounded engineer and the useful citizen outside of his strictly professional field. In this respect the institute has been a pioneer in engineering education and its founders took a position far in advance of the times. Nevertheless the institute has not escaped the charge of narrowness and this has sometimes come, alas, from some of her own sons who were not over zealous in availing themselves of the opportunities actually offered during their student days. More specifically, as has already been implied, it is charged that the graduates from engineering schools are not as a class showing themselves capable of development into men who can occupy successfully the commanding positions already described; and again the institute is not exempted. So far as this charge relates to breadth of view within the professions it is the immediate and vital concern of these schools. So far as it relates to those traits which go to make up the accomplished man of affairs it is ser-

ious, and demands earnest attention, but the remedies are less obvious; for these remedies must mean the superimposing upon an already heavy burden a task which should be begun in the home and largely completed there; a task, indeed, which no college has satisfactorily met with respect to all of its professional or non-professional graduates. So far as books can help an added year of student life would seem to afford a remedy and there has been much discussion of the desirability of extending the undergraduate course to five years, and of making the engineering schools the graduate schools. The arguments cannot be reproduced here but it seems clear the added expense incurred and the increased age at which the young man enters his life work militate seriously against the adoption of either of these as a universal procedure. For those to whom such opportunities are open they are likely to prove of great value, and it is interesting to note that each year seems to bring to the engineering schools a larger number of young men who have already graduated from some college, and encouragement is also to be found in the fact that more and more of these men have planned their courses during their college years with reference to later work in the technical school, a procedure which is much more to their advantage than what Professor Jackson refers to as a "butt-joint" between a general college course and a later engineering course. It should also be remembered that this is a recent educational development and that these men have not been tried out.

One serious difficulty which technical schools are encountering has been frequently referred to by recent writers but deserves a mention here, namely, that of securing and holding broad, cultured teachers. Specialization has invaded the teaching profession, especially in scientific lines where the mastery of any large field of knowledge to a degree corresponding to the needs of the expert is rarely possible. The specialist is apt to use the microscope far oftener than the field glass and this habit is partially reproduced in his students. It is encouraging to note that certain schools are now recognizing the need of men who are efficient teachers with a broader outlook, to deal especially with the younger men. They are recognizing that not every eminent specialist or successful investigator is a successful teacher, more particularly in this very matter of breadth of view, and are leaving the specialists greater opportunity for the presentation of their specialties to the older classes, while improving the instruction in the more general courses. It is obvious that these difficulties are enhanced by the larger financial rewards which tempt the broad minded engineer away from the schools—a serious matter which cannot further be discussed here but lies close to the root of much of the cause for criticism. It is interesting to note how even a single instructor, of some engineering experience, who keeps himself and his pupils closely in touch with current events, and leads them to understand that no single human attainment neces-

sarily represents the best that can be done, and that it may well be the privilege and the duty of any one of his hearers to extend the boundaries of such attainment, will give an impetus to successful effort that will be felt in the entire lives of his pupils. It is to be hoped that no one of us is unable to recall with gratitude some such instructor. We need more of them. A single instructor, again, who exemplifies the cultured scholar and gentleman in ease of manner and grace of diction does more for the cause of scholarship and culture than any quantity of sound advice can do; for, I fear that it is utopian to hope that a majority of the students with whom the study of engineering is their main purpose will ever believe that any man is disinterestedly sincere in his advice regarding such subjects as literature, language, art or economics, unless he makes it quite clear to them that these subjects have a distinct significance to him and are a part of his life. Just here lies one of the great obstacles to the elimination of "narrowness."

If the inculcation of breadth of view and love of the refined in life is difficult, the development of qualities of leadership is even more so. That these qualities are largely conferred at birth will, I suppose be generally admitted, but I take it that the criticism of lack of leadership is really directed toward an alleged culpable lack of facility in getting the best from others, of appreciating the point of view of others, or of presenting our own views to others. If this indicates a failure on our part to stir the ambitions of our students to avail themselves of opportunities which come to them, or to plan for themselves a really worthy career, then we are at fault; but if it means that the faculties of engineering schools should further encourage those forms of activity commonly designated as "college life," then I believe that we are on debatable ground. Of the importance of those traits which enable a man to win the confidence and respect of his fellowmen, to "succeed" among men, no one could be more conscious than I. In individual cases they may indeed be more potent factors than accuracy of scientific knowledge in securing preferment, and any man is fortunate who combines engineering skill with ease of manner and persuasive speech. But the real function of these schools is, after all, the training of capable engineers and it is very easy to pass the line beyond which there is grave danger that both the quantity and quality of individual attainment will be lowered because of time and energies devoted to social affairs. Let the schools realize by all means, their responsibilities for the development of men as well as engineers, and encourage by precept and especially by example an interest in all that tends toward a better understanding on the part of our students of their human relations, including prudent encouragement of the so-called "student activities." But let those who lack a realization of the great changes which the student life at our technical schools has already undergone in the last few years, and who therefore constantly clamor

for more of what is called "college life," reflect that one of the greatest assets which a graduate from one of these schools can take with him when he leaves it is the well-established habit of "doing a day's work in a day," of meeting his obligations on time, and let him realize that this cannot be reasonably demanded if the instructors must in fairness accept excuses because of an undue diversion of time and energy to other things. Although the sciences actually owe many of their advances to "grinds," it is probably fortunate that few of our engineering graduates of today belong to that class; but there is little likelihood of an undue increase in the proportion of such over-developed scholars under existing conditions! An impartial survey will, I believe, show that our recent graduates are, as a body, less open to the charge of lack of adaptability, and want of social resources than formerly and that they are improving in this respect as the need of such improvement is more generally realized, and also that there is ground for the belief that this has so far been accomplished without serious sacrifice of professional efficiency.

In what I have just said I have had in mind particularly the business and social relations of the young engineer with his colleagues and superior officers. It is often stated that some or many of the graduates also lack an appreciation of the proper way to deal with those whose labors they must direct. This, again, is doubtless in some considerable measure true, and in fact it can hardly be otherwise when nearly all of these young men pass directly from the public schools to the higher educational institutions. It is not, however, true that no effort is made to bring this phase of their future responsibilities to their notice, for not only is the subject discussed in its general aspects from the lecture platform but the young men are advised to secure summer employment as far as possible to the end that they may learn to know industrial conditions.

In this connection I should like to point out to those in control of our industrial establishments that there is a large store of energy, combined with a desire for opportunity to work and ability to render intelligent and willing service, which goes to waste in the summer because our students are unable to secure temporary positions. This is particularly true in the industries into which the men in whom I am especially interested, the chemical engineers and chemists, will go, I am of course aware, that the net return in value to a concern from this temporary service is not relatively large, especially during the first summer, and that in certain industries there is a risk in trusting to the integrity of these men with respect to information acquired regarding methods. But I cannot avoid the conviction that if the industrial managers would co-operate with the engineering schools in the consummation of an arrangement whereby young men whose ability and character could be vouched for could be given summer employment for two or three of the successive summers intervening during the four years of study,

the concerns thus co-operating would actually find that they would derive appreciable benefit from the plan. That it would enable the schools to add at least fifty per cent to their efficiency so far as these students are concerned, I have no question whatever, and surely no better means could be afforded for the acquisition of a knowledge of the problem of the laborer in the works. Let me add that I do not urge the placing of these young men at once in positions of responsibility but rather in such positions as will afford them working experience under industrial conditions. It seems to me, however, that it is not improbable that say, in a third summer the majority of such men might be utilized to much advantage in the immediate direction of specific processes or operations, they themselves acting under general or specific direction.

Some of us are just now concerned to know how with respect to chemical engineering we can give the young men an opportunity to come into contact with the actual practices of their profession before they leave the school, and the advisability of the equipment of laboratories of chemical engineering is under careful consideration. While it is no doubt true that, from its nature, chemical engineering offers less abundant opportunities for industrial work during the vacation interval in a student's career than many other professions, notably less than civil engineering, and at the same time is a profession the actual practice of which it is exceedingly difficult to reproduce in an educational plant, I suspect that similar general conditions exist in other lines. Here, again, is a problem with no small dimensions or importance with which we are wrestling, and one step toward its solution may be made through the greater co-operation on the side of the industrial managers for which I have just appealed.

If I have dwelt more upon the alleged weaknesses of the engineering school graduates than upon their strength, it is because the latter is attested by the engineering advance of the recent past to which they have contributed to an extent which would not have been possible had not the majority of them received from the schools an education and training which has proved useful, dependable and stimulating.

I believe that the large majority of the engineering school graduates are virile, intelligent, industrious fellows, with sound habits of thought and great capacity for work, ambitious to make the best of themselves, possessing a sincere desire to acquit themselves honorably both in private and public life and with an increasing ability to do so. As such, we their instructors, honor them, and ask your co-operation, advice and encouragement in our efforts to give to them what they deserve at our hands. We ask you also to recognize that while for the moment the rapidly changing social and industrial conditions may have outrun our ability to adapt our educational practice to them we are not lacking in an appreciation of the significance of these changes or of our obligations for the future.

THE CHEMISTRY OF RAW SUGAR PRODUCTION.*

By CHARLES A. BROWNE.

Few commodities of commerce have acquired so wide and general an interest as sugar, and this interest we find exemplified in many ways. It is shown best of all perhaps in the large number of books and publications which have been written upon sugar. These books, a collection of which in the Library of Congress in Washington has been classified together in one catalogue, constitute in themselves a large library; there are books upon the history of sugar, upon the economics of sugar production, upon the agriculture of sugar-producing plants, upon the manufacture, technology and refining of sugar, upon the chemistry and analysis of sugar, upon the nutritive value of sugar, upon the tariff laws and bounties regulating the importation and exportation of sugar and so on and so forth. The importance of sugar in the commercial affairs of the world is so great that some economists have used the per capita consumption of sugar as a means of gauging the wealth and prosperity of a nation. It may be interesting to note in passing that the statistics of 1910 show the following consumption of sugar per capita for several different countries during the past year.

	Pounds.		Pounds.
England	86.30	Germany	43.45
United States.....	81.60	Norway	41.78
Denmark	77.75	France	37.80
Switzerland	64.10	Belgium	32.36
Sweden	53.90	Austria-Hungary	25.14
Holland	43.53	Russia	22.82
		Spain	14.20

The phase of the subject which I have been asked to present to you this morning is the part which chemistry and chemists play in the production of raw sugar. Sugar, *i. e.*, sucrose, although composed of the common elements, C H and O—the two latter in proportion to form water—has not been made synthetically up to the present time. I mention this for we read occasionally in the Sunday newspapers of sensational experiments (paid press notices, no doubt) where steam is injected into an electric furnace containing charcoal at one end and a continuous stream of the purest white sugar flows out at the other. Here is a sample of such sugar, testing 100 per cent exactly as they claim, and here is an illustrated pamphlet telling all about the process. It is simplicity itself, for they show by formula that $C_{12} + 11H_2O = C_{12}H_{22}O_{11}$, and there you are. The most important feature of this, as of all similar announcements is: "The price of our stock is bound to increase rapidly; do not wait till the stock goes higher but buy now."

The process of making synthetic sugar just described is mentioned simply as an example of the countless number of fakes which are continually springing up in connection with the sugar industry;

*Lecture delivered to the third and fourth-year students in chemistry and chemical engineering of Columbia University, Jan. 18, 1911, and published in the School of Mines Quarterly.

there are fake powders which applied to the soil will increase the yield of sugar cane or sugar beets; there are fake chemicals for clarifying and fake processes of boiling and crystallizing. So many people have been deceived by putting money into fictitious schemes, that sugar planters and manufacturers have become in some countries ultra-conservative; they refuse to adopt a really good invention when it does come along. It is one of the chief qualifications of the chemist who takes up any branch of chemical technology as a profession to be able to detect quickly the false and good features of the numerous new processes which are brought to his notice; he will increase his usefulness and earning capacity in this way many times over.

Sugar may, at some time, be manufactured synthetically just as indigo is now made; and when that day arrives it may also happen that synthetic sugar may displace the agricultural product just as synthetic indigo is displacing the product of the indigo plant. But until the day of synthetic sugar does arrive we must be dependent for our supplies of raw sugar upon some one of the sugar-producing plants among which may be mentioned the sugar cane, the sugar beet, the maple tree, the palm tree and various plants of lesser importance as the sorghum and maize.

The oldest and best known sugar-producing plant is the sugar cane. This plant grows only in tropical and semi-tropical countries; it resembles in many ways the Indian corn, producing a jointed stalk varying from six to ten feet or even more in length.

In the long years of its history the sugar cane is probably but little richer in sugar than it was when the soldiers of Alexander first tasted its juice. There is an important reason for this: the sugar cane is propagated almost entirely by planting the stalks, the buds or eyes germinating and producing a new cane resembling the old stalk. It is practically impossible to improve a cultivated plant without the agency of seed production; there must be cross-fertilization and some process of controlled selection. The sugar cane produces seed in a feathery plume at the end of the stalk, but the seed had been regarded as sterile until about thirty years ago when it was found that under favorable conditions some of the seeds could be made to germinate; thousands of new canes have resulted from this discovery and these so-called seedling canes are displacing many of the old-established varieties. In the nurseries of tropical countries, where new varieties of sugar cane are bred, the problem of selection devolves almost entirely upon the chemist, who determines the percentage of sugar in the different canes, the purity of the juices and other factors which have to be considered.

The sugar cane, when the crop is ready, is harvested by cutting off the stalk as close to the ground as possible, trimming off the green tops and stripping off the leaves. These and the other agricultural operations of planting, fertilizing and cultivating require a large amount of labor, the expense for which makes up about

three-fourths of the cost of the raw sugar, the remaining one-fourth being due to the expense of manufacture.

The composition of the stalks and the expressed juice of the sugar cane vary considerably. The general range of the different constituents as compiled from analyses made in different countries is given in the following table.

	Whole Cane. Per Cent.	Cane Juice. Per Cent.
Water	67-75	80-86
Dry substance.....	33-25	20-14
Fiber (cellulose, etc.).....	10-15	
Sucrose	11-16	12-18
Invert sugar.....	0.5-1.5	0.5-2.0
Ash	0.5-1.0	0.4-0.8
Nitrogenous substances.....	0.4-0.6	0.1-0.4
Gums, acids, etc.....	0.2-0.5	0.3-0.6
Wax and fat.....	0.4	0.2

Individual cases may show variations above or below these figures.

The sugar cane after it is hauled to the factory is first passed through mills to remove the juice. The cane mills are of all kinds and types, and range from the crude ox-driven mills employed in the Philippines and other primitive countries, to the high power, steam-driven hydraulic nine and twelve roller mills employed in Cuba, Java, Hawaii, Porto Rico, Louisiana and other countries where the most modern machinery is used. In the best equipped factories the cane is delivered by an endless carrier to huge corrugated crushers, which reduce the stalks to a thick blanket of pulpy fiber, removing at the same time some 50 per cent to 60 per cent of the juice. The crushed stalks pass next through a mill of three rollers, where still more of the juice is removed, and then through a second, third and sometimes a fourth set of such rollers, the hydraulic pressure upon the rollers being increased at each mill in order to remove more and more of the juice. It is also customary to wet the pulp with a thin spray of water between the sets of rollers, the water thus soaked up facilitating the removal of the residual sugar by the succeeding rollers. This process of wetting, of maceration, as it is called, is highly important, but requires to be carefully controlled; the water added must of course be afterwards evaporated and the question which the chemist must decide is when the cost of evaporation begins to exceed the value of the extra sugar recovered. Large numbers of treatises have been written upon maceration and scarcely any two authorities are found to agree upon the details of manipulation. The quantity of water used for wetting the fiber is usually about 15 per cent per 100 parts of normal undiluted juice, although 25 per cent and more is sometimes used. With 15 per cent maceration about 90 per cent of the sugar in the cane is extracted in the juice; with 25 per cent maceration over 95 per cent of the sugar may be extracted. The residue of cane fiber as it leaves the last mill contains about 45-50 per cent moisture and from 3 to 5 per cent sugar, *i. e.*, from 5 per cent to 10 per cent of the original sugar in the cane. This residue of fiber is called "bagasse" and is burned under the boilers; it

constitutes the chief and in some countries the only supply of fuel for operating the sugar factory.

The polarization and purity of the raw juice are the first important factors to be determined in the chemical control of a cane sugar factory. The polarization of the juice will give the approximate sugar content; the dissolved solids in the juice are estimated by means of a floating hydrometer called a Brix spindle. The polarization of the juice multiplied by 100 and divided by the reading of the Brix spindle gives the purity of the juice. Good cane juices run over 90 per cent purity, juices running from 85 to 90 per cent purity are fair, and from 80 to 85 per cent medium. Juices with a purity below 80 per cent are poor and very unsatisfactory to work.

The second step in the manufacture of cane sugar is the clarification or purification of the raw juice. The best clarifying agent and the one that has been used from time immemorial is lime. Nothing better has ever been found, although hundreds of patents have been taken out for substitutes.

Many methods of using lime are practiced and a few of these will be described. Cane juice as it comes from the mill is slightly acid. One method of clarification is to neutralize this free acid of the juice by adding lime to slight alkalinity, and then to heat to boiling. The lime combines with the organic acids and phosphoric acid of the juice and the heat coagulates the albumenoid substances; a thick scum of impurities rises to the surface, this is skimmed off and the hot juice run into settling tanks when the suspended impurities settle out. More often the juice is passed through filter presses and the impurities removed in this way. The residue of impurities called "filter press cake" contains the phosphates and nitrogenous matters of the juice and is returned to the cane fields as a fertilizer.

In many factories the cane juice is sulphured before liming, gaseous SO_2 , produced by burning sulphur, is led into the juice to a certain point of acidity; the free acid is then neutralized with lime and the juice heated as first described. The SO_2 has a favorable bleaching effect upon the juice and the mechanical separation of the impurities is greatly facilitated by its action. Objections against its employment are the increase in scale (mostly CaSO_4) which forms upon the tubes of the evaporators and the contamination of the molasses with sulphites, the latter being forbidden in food products by the laws of some of the states. In some factories phosphoric acid is used with the lime; this gives a beautiful clarification but does not give the bright effect which bleaching with SO_2 brings about.

In some countries, notably in Java, lime is added to the juice to strong alkalinity and the excess of lime then removed with CO_2 . This process of clarification called carbonatation is the only one used in beet sugar manufacture. It works well with cane juices when but little invert sugar is present. If the latter occurs in large amounts the lime forms dark colored soluble

compounds which not only give a dark colored sugar but interfere seriously with the work of evaporation and crystallization. Such juices are said to be lime-burnt. The tendency at present in cane sugar manufacture is against carbonatation and all other methods of strongly alkaline clarification.

The third process in the manufacture of cane sugar is that of *evaporation*. In primitive countries and out-of-the-way plantations evaporation is carried out over the direct fire in open pans or kettles. The juice is either boiled down in one single kettle or passed through a train of pans; when crystallization of the sugar has begun great care must be exercised that the mass be kept in constant motion; otherwise there will be burning and caramelization next to the surface of the evaporator. Such caramelization is in fact unavoidable and all open kettle sugars are characterized by a dark color and by an agreeable aromatic taste which is preferred by many to that of the pure refined sugars. In some countries the cane juice after evaporating to a thick pasty mass is allowed to cool and solidify, just as molasses candy hardens after cooling. This solidified mass is called *concrete sugar* and is ground up in mills and marketed as a coarse lumpy sugar of very uneven composition. This concrete sugar contains of course all the molasses with the soluble impurities of the juice. Such sugar constitutes at present almost the sole output of the Philippine Islands. It is shipped to this country in mats weighing about 50 pounds and comes in three grades which contain all the way from 80 per cent to about 90 per cent pure sucrose.

In other primitive countries, especially in parts of South America, the juice is not evaporated to concrete but only to the consistency of a thick mush; this mush is run into hogsheads having a fine perforated bottom through which the molasses or mother liquor surrounding the crystals of sugar percolates. When as much as possible of the molasses has drained away the residue of sugar is removed and sold as *muscovado* sugar. This is purer than concrete sugar and polarizes sometimes as high as 92 per cent. It is usually quite moist and for this reason very liable to deteriorate.

In the open kettle process of evaporation there is always considerable loss of sugar due to caramelization and inversion caused by the high temperature of heating which may be from 20° to 30° F. above the boiling point of water. To avoid these losses all modern sugar factories employ vacuum evaporators which allow evaporation to proceed at a temperature much below the boiling point of water and at the same time permit the utilization of waste steam from the exhaust of the engines and other points about the factory. Vacuum evaporators are manufactured in many different forms, and are arranged usually in a series of two, three or four sometimes even as high as five or six, the combination being called double, triple, quadruple, quintuple, or sextuple effects. In the first vessel of an effect, a lower in the second than in the third, and so on, the temperature of boiling for each succeeding ves-

sel is thus progressively reduced. The steam which is evaporated from the juice in the first vessel goes to heat the coils of the second vessel, the steam from the second vessel goes to the heat coils of the third, the reduction in temperature of the steam for each vessel being of course counterbalanced by the greater vacuum and lower temperatures necessary for boiling. With a long series of vessels as in a quadruple or quintuple, or sextuple effect, the thin juice in the first body may be boiled under atmospheric pressure or even at a few pounds above this; this is necessary in order to get a high enough temperature to carry sufficient heat through to the last evaporator. The subject of multiple evaporation is a science in itself and exhaustive treatises have been written upon this one single phase of sugar manufacture.

After the clarified juice has been evaporated to a syrup we come to the fourth stage of the process of modern sugar making—the graining or crystallizing of the sugar. This is accomplished in a vacuum pan which is operated in much the same way as one of the vessels of an effect; in the case of the vacuum pan and the same is true with many other effects, the process is assisted by connecting the outlet with a vertical condensing column 28 feet high. A stream of cold water flows through the column and this serves both by rapid condensation of the steam and by the barometric pull of its column of liquid to maintain a high degree of vacuum.

To operate the vacuum pan a charge of syrup is first drawn in; this syrup as it leaves the evaporators has a sp. gr. of about 1.25 or about 50 per cent solids and is boiled down in the vacuum pan to a sp. gr. of 1.50 or about 90 per cent solids. The ebullition in the vacuum pan is violent and unless the sugar boiler is careful some of the syrup may be carried over with the vapor into the condenser; this is called *entrainment* and is a source of frequent losses in sugar manufacture. In all modern sugar factories the chemist makes constant examination of the condensation water, using the familiar *a*-naphthol test with H_2SO_4 , this test being delicate enough to detect one part of sugar in many thousands of water.

The handling of the vacuum pan requires more skill than any other operation of the sugar house; care must be taken to avoid entrainment and care must be taken to build up crystals of uniform grain or size. The usual practice is to boil down the first charge of syrup to what is called "string proof," *i. e.*, to the point when a few drops of syrup withdrawn from the pan will draw out between the fingers in fine strings or threads. When this point is reached a large charge of fresh cold syrup is drawn into the pan; the sudden cooling of the supersaturated contents starts the formation of innumerable fine crystals. These first crystals constitute the foundation so to speak of all the sugar obtained in a given boiling or strike of the pan; the boiler aims to build up these crystals without forming new ones; he aims from now on to avoid supersatura-

tion and to avoid sudden chilling through drawing in too much syrup at one time. He controls his process by drawing out samples every few minutes and examining these upon glass against a light; if he sees fine new crystals appearing among the old ones he reduces the vacuum a little, thus raising the temperature and dissolving this false grain as the fine crystals are called. By skillful manipulation, which only comes with long practice and experience, the sugar boiler is able to build up his crystals to any desired size; the usual practice is a crystal about the size of ordinary granulated sugar; in certain localities, however, a large crystal is favored, as for example, in Peru, where the sugar is boiled slowly and for a long time, thus building up a very large grain.

When the vacuum pan is filled with a thick magma of sugar crystals, of about the consistency of mortar, the steam is shut off, air is admitted, the bottom of the pan opened, and the entire contents dumped into a mixer, which keeps the mass in slow movement by means of revolving arms. This mixer is situated over a row of centrifugal machines; the mass of crystals called sometimes *masse cuite* from the French or Füll-mass from the German is drawn off gradually in successive charges into the centrifugals. The inner walls of the latter are lined with fine brass meshing and as the drums are rotated the *masse cuite* is whirled against the meshing which retains the sugar but allows the molasses to pass through. After spinning for a few minutes until as much of the molasses is removed as possible, the revolving mass of sugar may be sprayed with a fine spray of water or a jet of steam in order to remove more of the film of molasses which remains adhering to the crystals; the amount of spraying depends upon the whiteness of sugar desired. In Louisiana a very pure, white sugar is made by spraying with several sprinklings of water; such sugar is over 99 per cent pure sucrose, the remainder being mostly moisture. In Cuba and Porto Rico they aim to make a 96 per cent sugar. In Hawaii and Java a sugar testing about 97 per cent is desired. Spraying will, of course, dissolve some of the sugar, so that the process is one which must be carefully controlled.

After the sugar is spun out, the centrifugals are stopped and the sugar emptied through the bottom of the drum into a conveyor by which it is carried to the bagging department where it is prepared for shipment. The raw sugar from the centrifugal contains considerable moisture and in some countries the sugar is dried in revolving drums before being bagged. This drying is advantageous for two reasons: (1) The excess moisture is removed, thus saving the transportation charges upon water, and (2) the sugar is sterilized and protected against the attacks of ferments and bacteria. The drying of raw sugar is not practiced in Cuba, Porto Rico or Louisiana, but is carried out in Java and the Hawaii Islands where the sugar has to be shipped for long distances to this city and other

ports for refining. The storage of undried raw sugar for long periods of time is a risky operation, as many speculators in sugar have found to their cost.

The sugar which is made from the pure juice of the cane is called "first sugar" and the molasses drained from this sugar is called "first molasses." The latter still contains a large amount of sucrose and various processes are used to recover as much of this as will crystallize. The first molasses is sometimes boiled down again in the vacuum pan and a second crop of sugar crystals obtained; this is the second sugar and the molasses obtained from this the second molasses. The second molasses may be boiled even again and a third sugar obtained, the molasses from which is the third or final molasses. Of course, as the sugar is removed the impurities become more and more concentrated in the molasses, until finally a thick stringy mass is obtained which will no longer crystallize. Such a molasses may still contain, however, 30 per cent sucrose; there is also present about 30 per cent invert sugar, 8—10 per cent of ash, and 8—10 per cent of gums, organic acids, amido-compounds, etc. This residual molasses is unfit for human food, although attempts are made to bleach it, dilute it with glucose and make it otherwise presentable. The waste molasses is valuable, however, as a cattle food and is also used by distilleries for making alcohol.

The tendency of modern methods in cane sugar manufacture is against the repeated boiling of molasses; the aim is to get as much sugar as possible in one operation. Many processes have been devised to attain this end. One method is to take the molasses from the first strike of sugar, draw this into the vacuum pan with the syrup for the succeeding strike and boil the two down together. The *masse cuite* from this mixture is then run while still hot into large tanks, called crystallizers, where it is kept in slow motion by means of revolving arms; as the mass cools and thickens more molasses is drawn in to keep the proper degree of fluidity. When no more sugar will crystallize as determined by analysis of samples, the contents of the crystallizer are spun out in centrifugals and the molasses withdrawn from the factory. There are many modifications of this process, each sugar manufacturer having usually a pet scheme or method of his own.

With this very hasty description of the sugar cane industry we will pass to the great rival of the cane, the sugar beet. The history of the sugar beet industry extends back only about one hundred years, but though centuries younger than the cane industry the story of the development of beet sugar manufacture is in many respects more interesting to the chemist. The sugar beet may be called the foster child of chemistry; in its wild uncultivated form, even as we find it growing today, the primitive sugar beet is very deficient in sugar. The beet, however, belongs to a much younger family of plants than the cane; its habits are not so firmly fixed; it accommodates itself more easily to new conditions. The propagation of the beet by seeds, in-

stead of by buds as with the cane, has rendered it easy to build up the sugar-producing capacity of the plant until it has reached and in some cases even passed beyond that of the cane.

I ought to mention here the important historical event which led to the founding of the beet sugar industry; it was the great continental blockade established by the fleets of England against the whole continent of Europe during the wars with Napoleon. The nations of Europe shut out from their supplies of cane sugar were driven to devise a substitute; in some countries, as in Bohemia, they made sugar from the maple, in other countries they crystallized dextrose from the juice of the grape, but these sources were inadequate. The final outcome of the matter was the birth of the beet sugar industry. It was one hundred years ago in 1810 that the first loaf of beet sugar was made and presented to Napoleon; and the centenary of this event has just been celebrated in France.

It is doubtful if the beet sugar industry could have made much progress, had an important piece of apparatus, the polariscope, not been invented about this time. The polariscope offered a quick and rapid method for determining the amount of sugar in beets. There was thus established a means of selecting beets for the production of seed—mother beets as they are called. A small boring is taken diagonally through the heart of the beet and the percentage of sugar determined in this by the polariscope. The beets of highest content are planted for seed; the same process is carried out with the second generation of beets grown from the seed of the first and so on generation after generation. In this way the original sugar content of the beet of only a few per cent has been raised to 14, 16 and even 18 per cent. The process of seed selection is still kept up and in the seed nurseries chemists are kept constantly employed polarizing mother beets for seed production. The work demands constant vigilance; a plant easy to improve will deteriorate even more easily; if the sugar beet were left to itself it would revert in a few generations to the old condition of low sugar content.

Extremely large beets are usually deficient in sugar and are not desirable; a medium sized beet is most preferred. Many beet growers place considerable stress upon the spiral twist which the root sometimes assumes; this factor, however, is probably of little significance.

Before taking up the details of manufacture we should look for a moment at the composition of the sugar beet and its juice. The average range in composition is given in the following table:

	Sugar Beet, Per Cent.	Sugar Beet Juice, Per Cent.
Water	75 —85	78 —84
Dry substance ..	15 —25	16 —22
Fiber (cellulose, etc.).....	4 —6	
Sucrose	12 —16	13 —17
Invert sugar.....	0.6 — 0.3	0.6 — 0.3
Ash (salts).....	0.8— 1.5	0.6— 1.0
Nitrogenous substances...	1.5— 2.5	0.8— 1.5
Gums, acids, etc.	0.4 — 0.2	0.3 — 0.6
Wax, fat, etc.....	0.2	

It will be noted as compared with the cane there is much less fiber in the beet and more water; there is also more ash (or salts) and more nitrogenous matter, but much less invert sugar than in the cane. These differences in composition have an important bearing upon the differences in process of manufacture.

In the first beginnings of the beet sugar industry the beets were crushed and the juice pressed out in much the same manner as with the cane; this process, however, soon gave place to the process of diffusion. The beets, after they are dug and deprived of their green tops, are hauled to the factory; they are first washed to remove adhering dirt and then passed through revolving knives which reduce them to fine slicings or chips.

The fine slicings are next carried by a conveyor to the diffusion battery, which consists of a series of tall boiler-shaped cylinders called cells. These cells are connected together by pipes, the outlet from the top of one cell passing downward into the bottom of the next and so on around. Each cell is filled with beet slicings through a manhole at the top and when full is tightly closed with a cover which is clamped into place. Twelve cells connected in series usually constitute a battery and when all are filled, warm water of about 80° C. is passed through the system. The water circulating through each cell removes the sugar from the beet slicings and of course becomes richer and richer in sugar with each succeeding cell. Heaters are placed between the cells so that the circulating water is kept always at the right temperature. When the water has made a complete circuit through the twelve cells of the battery the slicings in the first cell are practically exhausted; this cell is then thrown out of circulation, emptied of exhausted chips, refilled with fresh slicings and reconnected with the system, while the second cell undergoes replenishing. The process is thus a continuous one; ten cells are always in circulation while one is always being emptied and one always being refilled. This goes on continually during the beet campaign, night and day, Sundays and holidays; interruption at the diffusion battery upsets the work of the whole factory.

The exhausted slicings from the diffusion cells are dried by the heat of the flue gases from the boilers and are then sold as a cattle food.

The diffusion juice as it leaves the last cell of the battery contains from 12—15 per cent sugar and is then ready for clarification. The juice is first treated with a considerable excess of lime; and the dissolved lime precipitated by leading in a stream of CO_2 . This process, called "carbonatation," has already been described.

The burning of lime is an important operation in every beet sugar factory, the lime being used for saturating and making saccharate; and the kiln gases, which contain on the average about 30 per cent CO_2 being pumped off for carbonating.

After the first treatment with lime and CO_2 the pre-

cipitated CaCO_3 , and other impurities are filtered off in filter presses and the juice subjected to a second carbonatation at a higher temperature. Less lime is used in this second carbonatation, the final alkalinity being only a few hundreds of a per cent CaO .

The juice from the second carbonatation is again filtered, when it is evaporated, grained, and centrifugaled, these processes being carried out exactly as described for cane juice.

There is a great difference in the physical properties of raw cane sugar and raw beet sugar. Raw cane sugar has usually a pleasant, fragrant odor and a delightful taste which many prefer to the refined product; this is due to the presence of invert sugar and the slight caramelization which this invert sugar has undergone during manufacture. Raw beet sugar, on the other hand, has a very objectionable odor, due to the nitrogenous substances of the beet root and a very bitter taste due to the mineral salts taken up by the beet root from the soil. Beet sugar is not a fit article of food until it has been refined. Attempts have been made to render raw beet products more palatable by subjecting them to a slight inversion and caramelization but this method has been only partially successful.

Many processes have been devised for recovering the sugar contained in low grade beet molasses. In the short time at our disposal we must pass over the osmose process, barium process and other methods of desaccharification and describe briefly the saccharate process of Steffens. In this process the molasses is diluted with water to about 12—14 per cent solids and the solution treated in the cold with very finely powdered lime, using all the way from 80—150 parts of lime to 100 parts of sugar. Constant agitation of the solution is necessary to prevent the formation of clumps of lime and care must be taken to keep the temperature down. The insoluble tri-saccharate of lime which is formed is filtered in presses and washed with cold water to remove impurities. This tri-saccharate is then mixed with hot washings from the filter presses and added directly to the diffusion juice for saturating; the saccharate is decomposed by the CO_2 and the liberated sucrose worked up with that of the juice. In some factories the saccharate sugar is treated separately; there are many modifications of this process and nearly every factory has a method of its own.

A peculiar impurity found in sugar beet products is the sugar raffinose, which has a polarizing power over 50 per cent greater than sucrose. A very small percentage of raffinose will introduce serious errors in the control of factory operations and the chemist may wonder why his yield of crystallizable sugar does not come up to his polarizations. The sugar refiners in England have been much agitated the past year by finding large amounts of raffinose in raw beet sugars from Germany and a congress was held in Berlin to arbitrate the question.

In addition to raffinose there are gums and pectinous

substances in the beet which affect the polarization and introduce errors in the calculation of the sucrose balance. Similar gums are also found in cane products. Every well-organized cane sugar and beet sugar factory maintains a strict account of all the sugar which enters the factory in the cane or beet and all the sugar which leaves the factory in raw sugar or molasses. The difference between the two should equal the determined losses of manufacture, such as sugar lost in the press cake, sugar lost in the spent beet chips, etc. If there are any undetermined losses the chemist is called upon to explain; the papers and discussions upon undetermined losses would alone fill several volumes every year.

In the present condition of the sugar industry the chemist has to understand not simply the chemistry of sugar; he must also be perfectly familiar with the chemical properties and behavior of all the non-sugars which do and may occur in his raw products. A German chemist, Rümpler, has written a book upon the non-sugars of the beet; the list of compounds runs up into the hundreds. Every technical sugar chemist should master the contents of this book. It is a pity that no such work has been written upon the non-sugars of the cane.

The application of strict scientific principles has been followed more energetically in beet sugar manufacture than in cane sugar manufacture and this we would expect for the beet industry was developed in the highly civilized European countries where skilled labor was abundant and the cane industry was maintained in primitive tropical countries where labor was shiftless and unskilled. This condition of affairs existed for many years and gave beet sugar for over two decades a strong ascendancy over the cane. In 1852, taking the whole supply of the world, there was six times as much cane sugar manufactured as beet sugar; beet sugar kept gaining, however, until 1884 the production of the two kinds of sugar was about equal, a little over two and one-half million tons each. Beet sugar then took the lead until in 1899 the production of beet sugar was about five and one-half million tons and that of cane about three millions. Then came the ending of the Cuban war and with it the introduction of the scientific principles worked out for the beet industry to the old dilapidated cane factories of Cuba and Porto Rico. Modern methods of manufacture had also been busy reorganizing the cane industry in Hawaii, Java, Australia and other countries. The production of beet sugar in 1901 reached nearly seven million tons and since then has remained about stationary; but the increase in cane sugar production has gone forward each year by leaps and bounds. In 1907 the cane had caught up to the beet, the production being a little over seven million tons apiece. In 1908 cane sugar was over 600,000 tons ahead and in 1909 nearly 2,000,000 tons ahead out of a total world's production of about 15,000,000 tons. This supremacy the cane seems destined to maintain.

There are several reasons for the belief that the future development of the sugar industry will be more along the cane lines. There is first the cheaper cost of producing cane sugar; the average cost of producing one pound of raw sugar being about 1.75 cents for the cane and 2.25 cents for the beet, a difference of 0.5 cent in favor of the cane. There is second, the much greater yield of sugar per acre which the cane gives over the beet. Under the best systems of cultivation and manufacture such as are found in Hawaii and Java, over four tons of sugar can be produced per acre. Germany, which leads in the production of beet sugar and in the yield of this per acre, can raise barely two tons of sugar per acre. Then there is third, the preference which the refineries of England give to cane over beet sugar. Yet notwithstanding these advantages possessed by the cane industry, the production of beet sugar seems destined to hold its own for many years to come; and indications are that the present year will see a record-breaking beet sugar crop of 8,500,000 million tons. The consumption of beet sugar, however, will be limited largely to those countries where it is produced; it cannot compete with cane sugar in foreign markets.

In the following table I have arranged some twenty leading sugar-producing countries of the world in the order of their production for the year 1909-10.

	Tons.
1. Germany	2,027,000 beet.
2. Cuba	1,894,349 cane
3. Austria	1,257,000 beet.
4. Java	1,200,618 cane.
5. Russia	1,145,000 beet
6. France	801,000 beet.
7. Hawaii	462,613 cane.
8. United States.....	450,595 beet.
9. United States.....	335,000 cane.
10. Porto Rico.....	308,000 cane.
11. Brazil	253,000 cane
12. Belgium	250,000 beet.
13. Mauritius	244,597 cane.
14. Holland	198,000 beet.
15. Mexico	160,000 cane.
16. Formosa	160,000 cane.
17. Peru	150,000 cane.
18. Australia	134,000 cane.
19. Argentine	125,000 cane.
20. Philippine Islands.....	120,000 cane.
21. Demerara	101,843 cane.

During 1910 Chile produced 7,705,535 Spanish quintals more nitrate than during 1909, with a prospect of as great an increase for 1911.

The recent discovery of tungsten in Nova Scotia has greatly interested the steel and metal manufacturers both here and abroad. The mines are located on Moose River, about 30 miles from Halifax, and it is claimed that the veins in number and richness exceed those of any other discovery. So far the mines have been merely prospected, with a yield of scheelite amounting to 75 tons, valued at \$30,000.

The new Technical College at Dunfermline, Scotland, is now ready for formal opening. The institute has been erected at the joint cost of Andrew Carnegie, the Carnegie Dunfermline Trust, and the Dunfermline school board.

METALLURGY.

EXPLORATION OF CUBAN IRON-ORE DEPOSITS.*

By DWIGHT E. WOODBRIDGE.†

During April, May and June, 1910, I was in charge of an examination of the greater part of the Moa iron-ore area in Oriente Province, Cuba, on the north coast, near the east end of the island. My instructions, on arrival at the properties, were to check former estimates of tonnages and grades, and to re-examine the ore comprised in claims covering 44,727 acres. This work included the running of lines dividing the properties into co-ordinate planes, the boring of many thousand feet of holes spaced at the intersections of these co-ordinates, the taking of samples of the ore penetrated, the analysis of these samples for their various constituent minerals, and the determination of the results as to tonnages, depths, and grades, both for individual properties and for the entire group. Each section of every one of the thousand holes drilled was to be compensated for depth and grade with every other, a series of simple arithmetical calculations of no slight magnitude, the mere mechanical labor of which consumed much time, but finally resulted in giving a complete average of all the essential facts for the entire area of 18,000 hectares.

Articles descriptive of the discovery and development of a tonnage of 600,000,000 tons of commercial iron ore in the Mayari field by the Spanish-American Iron Co., have been published from time to time. Subsequent to these discoveries and their exploitation, the red soil at Moa was recognized as an iron ore, and researches were immediately instituted to determine the quantity and quality of this ore. These investigations commenced in 1906 and had been carried on almost continuously with a varying degree of vigor up to the time of my own examination, in 1910. The tonnages of this new district proved to be greater than those of Mayari, while the quality was found to be quite similar. The resemblance in grade was but natural, since the origin of ore in these two fields was precisely the same and the breaking down of the ore-bearing rock has proceeded at Moa in a manner analogous to that process at Mayari. More than 50,000 acres of land were examined and drilled, the district was mapped, and thousands of drill samples were analyzed. It was found that the general area of these Moa fields was superposed upon about 60 sq. miles, and that the ore beds extended directly to the Atlantic

shore, forming a blanket more or less continuous from the sea to the summit of the island, the height of land between the Atlantic ocean and the Caribbean sea.

A precipitous range of rugged hills is practically continuous along the north coast of Oriente Province. These hills attain an altitude of from 2,000 to 2,500 ft., and approaching by sea, form the distinguishing feature of the landscape. At points the slopes reach the water's edge, elsewhere they are some miles from the shore. Numerous bays break the coast; some large enough for harbors for ocean-going ships, while others are constricted in area and shallow in depth. A series of coral reefs extending for many miles along the coast protect it from the constant sweep of the Atlantic surge, which is hurled in by the steady NE. trade-winds. Occasionally these reefs are cut by broad and deep entrances, easily distinguished by the break in the otherwise uninterrupted line of white water that is like a foamy stripe, elongated on either hand until it ends, a mere ribbon upon the blue. These reefs, awash at low tide, are covered at high tide, so perfect a protection do they form that the decrepit, poorly-rigged, flat-bottomed fishing boats of the natives are safe inside, no matter how fiercely the combers may smash upon the reefs beyond.

The *ciudadita* of Baracoa is 35 miles east of Moa, and its history extends back to the time of Columbus, for it was here that he first landed on Cuban soil. The town was founded in 1500. To the west, 50 miles, is the capacious bay of Nipe, where are situated the works and shipping piers of the Spanish-American Iron Co., the sugar mills of the United Fruit Co., and a terminus of the Cuba railway. Between Baracoa and Nipe bay there are no settlements worthy the name—only an occasional fisherman's hovel, where a cocoa palm grove comes down to the sea, or where there are a few roods of cultivable soil. So much of the scanty earth along this stretch of coast is iron ore that arable ground is hard to find and is in high request.

Roads scarcely deserve the name in this section of Cuba. While there is the *Camino Real*, the so-called King's Highway, it is impassable for wagons, and from Moa to Baracoa a pack-mule cannot get through, even with an empty saddle. In seasons of high water the roads to Sagua and on to Nipe bay cannot be traversed at all, and communication is almost entirely by boat. The poor transportation increased the difficulty of securing provisions and supplies, of getting and keeping competent men, and of handling the mails.

No surface of soil exists over these ores; indeed, the ore itself is the soil, upon which grow either pine for-

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ests or a characteristic tropical jungle. On the lower elevations and in the better drained of the upland interior, pine predominates; inland, where the rainfall may be heavier, and wherever it remains more permanently after falling, the verdant jungle enters. It closely resembles the jungles of northern South America, with its tough, cord-like creepers, its strange arboreal growths, and its dense poisonous and prickly shrubbery. It is hard to penetrate unless one has in his hands that omnipresent weapon, the *machete*. In the belief that a thin capping of surface soil and humus might lie above the ore in these jungles, I took a number of samples in these woods at varying depths, which showed on analysis that, when found at all, the ore extended to the surface, whether timbered or not. No stripping of these ore bodies is necessary to fit them for mining, and during the dry seasons a lighted match may be applied to the forest floor and the fire will clean off all organic matter above the ore, leaving it free and fit for immediate mining by the steam shovel or other means of excavation.

Scattered about the surface of these deposits are boulders, flat sheets, pellets, and nodules of hard iron ore, somewhat dehydrated, and varying from masses of many tons to pieces the size of minute bird-shot. Natives call the pellets *tierras de perdigones*, or "shot soil," a name warranted by their appearance and by the use to which they sometimes have been put, both in peace and in war. While the upper inch or two is occasionally composed entirely of this material, it is usually carried in a matrix of soft ore, and it was the original design, at the time of discovery, to wash this hard ore from the surrounding red soil and ship a product of indurated iron ore. This scheme, however, was impracticable, owing to the expense of collecting the hard ore, which is spread over a great area in a comparatively thin layer or appears in isolated deposits and pockets; moreover, the matrix contains so much clay that washing was slow and difficult. During the course of experiments having in view the washing of this material, it was found that the soft ore matrix was as good ore as the hard, and it was not until this fact was fully realized that the great size and vast importance of these deposits were appreciated and their possibilities realized.

It has been considered by some engineers that these shot ores cemented into masses occur in layers and bedding planes, and so form a persistent sheet covering a large continuous area. In proof of this they point to the hard boulders frequently found underground in the progress of drilling operations. Basing my opinion on the results of a drilling campaign greater than that of any concern aside from the Spanish-American Iron Co., at the Moa and Mayari properties, I cannot agree with this theory. I believe the hard ore found underground in drilling to be blocks and boulders of this cemented material, and not often of large size. Also, that the horizontal outcrops of cemented nodules, at times found along the sides of

erosion cañons, are not original, but have assumed their present condition since they became subject to the changes incident to surface action; and this is the case whether they are directly upon the top of the ground or near to it. Contrary to statements made in occasional reports, there are in these deposits no definite layers of ore of varying degrees of induration, color, or class. The deposits are homogeneous masses, and the harder ores found so frequently are the result of heat, the action of the elements, and the infiltration of iron salts as a cementing material; while the variations of color and texture are the result of a more or less hydrated condition and a more or less complete disintegration of the original rock, all due to local favoring or retarding causes. I took careful note of the depth reached by nodulized ore and found it to average a few inches, while the extreme depth was 24 ft. This latter depth for nodules was rare; in such cases their proportion of the mass was very slight.

The deposits constitute a surface mantle varying in thickness from a mere film to 121 ft., which, I believe, is the extreme depth ever drilled in ore in Moa. This hole was bored by men in the employ of the Juragua Iron Co. The greatest depth which I attained was 81 ft., said to be the second deepest ever bored there, and the deepest ever put down by an ordinary crew of two men. There is an average thickness of from 18 to 20 ft.; the results of work under my supervision, covering an area of more than 8,000 hectares of ore drilled, showed an average of 18.83 ft.; Mayari ore, I understand, is a trifle thicker. The thickness of the ore mantle is affected by local causes, assisting or delaying the breaking down of the serpentine rock (which experts agree to be the mother of this ore), erosion by streams, and other causes. The ore lies directly upon the serpentine, and mining will be somewhat unfavorably affected by the fact that the gradation from ore to rock is not at all regular, but very rough, so that in cleaning the bottom of an ore body with any sort of automatic machine, chunks of serpentine are liable to be broken off and lifted with the ore, unless care is constantly exercised. This irregularity is shown plainly at the mines of Mayari, and shipments from these openings to Nipe piers sometimes contain serpentine broken from the floor.

Torrential mountain streams are frequent in this area; a square of 225 hectares was measured for check work in which were no less than three large rivers with deep gorges, each one worn well into the underlying rock. In this particular area about 25 per cent of the total was barren of ore. But, while there are many streams, this special case was abnormal and cannot be duplicated in the entire district. In spite of a brief rainy season and a long dry period, waters flow with surprising volume throughout the year. But erosion at the present time is exceedingly slight and entirely negligible so far as tonnage of ore is concerned, for the *arroyo* slopes are hard and smooth, and, even

in flood, the rivers bear comparatively little material in suspension.

One peculiarity of this ore is that it stands indefinitely without caving. On exposed vertical faces, open to storm and sun, there is no appreciable sloughing-off of the sides. I have seen pits dug years ago, that have been open to the action of the weather, the vertical walls of which still retain marks of picks and other tools of the diggers. This ore is very clayey, representative and composite assays showing Al_2O_3 , 13.34, and SiO_2 , 3.36 per cent. Derived, as it undoubtedly is, from serpentine, the proportion of alumina is naturally very high.

By reason of the character and condition of these ores exploration can be carried on by a process that is simple, accurate, rapid and cheap. Ordinary 2-in. auger-bits are forged on one end of 4-ft. sectional rods, the other end being fitted to receive a sleeve nut, 5 or 6 ins. long, into which another 4-ft. section may be screwed. As a hole is driven down by the auger bit additional threaded sections are screwed on the rod, making it any desired length. On each end of each rod, except where the bit is shaped, is a backing nut screwed down hard, in order to prevent the rods from working too tightly into the sleeve nuts when turned into the resisting ground, which would render it difficult to release quickly. In most cases ore can be bored by this simple tool with comparative ease, and when hard blocks and boulders are encountered underground, they are sometimes cut by the substitution of a cutting chisel bit for the auger point; in other cases the men will move a few feet away and drive another hole, experience having shown that a very short distance will usually be sufficient to avoid a boulder. The hole is started through the drier top soft ore or nodules on the surface, a little water is poured in, the bit lifted and driven down by the combined strength of two men, and then turned in the ore. The work is a combination of churning and boring. Every few feet the tool is lifted, the ore adhering to the bit is cleaned off by pressing a stick into the point of the bit and then revolving the tool, and saved for analysis, and all sludge that has collected above the bit is scraped off. If the hole is sampled in sections, all ore taken out of each section by the bit is saved to make a full sample; but if the hole is sampled as a whole, the ore is all piled upon a cloth and afterwards mixed and quartered down with the ever-ready *machete* to make a suitable sample. When sampling was in sections it was found best to adopt 5-ft. lengths, both for general convenience and to ease the work of the calculator of averages. The drilling is hard work in deep or difficult holes, or where nodules are frequent—as hard as any labor that a man can comfortably endure. It is done almost entirely by Spaniards, mostly from the province of Galicia, who become very expert and earn good pay. It is all task work, and the going rate of contract wages varies with the depth of holes as well as with the character of country rock. Each pair of

boring-men is accompanied by a water carrier and a sample marker, both paid by the day; the sample marker acts as the representative of the employer. He measures the holes and sees that bottom is reached before the drilling is stopped. The deeper the work the more difficult it is, and there is on the one hand a tendency on the part of drillers to shirk, and on the other to allow themselves extra measurements. They will stop in ore if it is hard drilling, marking *piedra*, or “rock,” on their last sample, if there is no one to check them. Were it not for the peculiarity of this ore of standing without caving, this system of drilling would be impossible, and it would be difficult for the engineer to follow and check the depth of holes by dropping down a measuring rod, or by inserting a bit with which to test the material at the bottom. It is not uncommon to check grades of properties previously drilled by inserting bits in the old holes and reaming out a sample from the sides of the hole. If the original hole has been protected at the surface by plugging it with a piece of sapling, it is very unusual to find the hole caved or destroyed.

The price paid the borers begins with from 1.5 to 2 cts. per foot for the first 10 ft. of depth, and increases by the addition of a like sum per foot for each succeeding 10 ft. of progress following. In ordinary ground each borer will earn from \$2.50 to \$3 per day; in other words, a pair of borers will complete from 10 to 13 holes, averaging 20 ft. deep, per day. Sometimes, when work is unusually difficult, or when it is desirable to get special results on check work, it is necessary to pay by the day at the rate previously earned on contract, or to give some sort of bonus for depths. Working with one of these drills, two men in my employ drove a hole 81 ft. deep, although it took them two long days to complete it. This hole was drilled at a spot where I thought that the ore was thicker than the original testing, or my own first check, had shown it. The original record was 22 ft. and was marked “rock bottom”; my own check was 20 ft. and was likewise marked “rock.” But the third attempt went down four times as far before it really bit the serpentine, though located less than 10 ft. from either of the others. Evidently both former holes had cut into an ore-boulder that the men thought was bottom, or that they did not desire to penetrate. In the third effort to reach bottom 10 ft. of hard ore was cut by the use of a chisel bit between the 20- and 30-ft. levels. A fact that was somewhat of a surprise to me, in connection with this hole, was that the bottom section, from 75 to 80 ft., showed ore as high in grade as that in any other part of the boring, and slightly above the average. The borers acquire great facility, and work rapidly and hard. If the ground is easy of entry they complete the holes quickly, and race each other from one location to the next in order to lose as little time as possible. They regard themselves as of a type of laborers higher than the average, and feel pride in their occupation.

In no other way is it possible to explore such an area except at great expense and in a long time. No system of tunnels, pits, or other openings is so well suited to this work. It is well enough to sink pits occasionally, to check by actual observation certain facts that seem patent from the drilling, or to answer questions that may arise. In this manner of drilling there have been bored on that part of the Moa area explored by my men more than 50,000 ft. of openings in ore, counting the work of original explorers and my own check work.

By this rapid and inexpensive method of boring the ore blanket it will be possible to determine in advance of any actual mine operation the precise quality of product to be expected from any given area, and thus to regulate grades won, or to produce any quality within the chemical limits of the ore body. And it will be a simple matter to ascertain in advance the general topography of the underlying rock, and thus to bring mining work for years to come under an assured and definite plan and system. All this, of course, means a greatly reduced cost of mining.

To those accustomed to vein mines or to the great replacement deposits of the Mesabi iron range, borings varying from 100 to 300 m. apart may seem utterly inadequate to prove grades and tonnages. But a consideration of the origin of these fields and of their necessarily quite homogeneous character answers this objection in part. The answer is made definite and conclusive and the customary methods proved safe by results secured in actual practice. In early examinations of the Mayari field original borings were spaced every 100 ft., but as the work proceeded the ore was found to be so regular in analysis, texture, and thickness that holes were gradually spaced at intervals up to and even exceeding 1,000 ft. There was some variation in the essentials, but the averages proved so closely as to be accepted as perfectly competent evidence. The results reached by these more widely separated borings have been since abundantly proved and confirmed by intermediate holes spaced as close as 50 ft. from each other; while in actual mine operation over the same ground, shipments also check these distant original holes. My own intermediate lines, run between coordinates, were a further proof. Engineers and others accustomed to narrow veins and comparatively small tonnages may be startled at such figures as this work presents, secured, as these have been, on data that may seem absurdly insufficient, but study and examination will convince them of the reasonableness of the assumptions made.

One interesting peculiarity of this ore is that often its appearance is no guide to its analysis. Naturally one might expect the deep reddish soft ore to be of better grade than the coarser, yellowish ore containing grains of quartz, etc. But this lighter yellowish ore, when dried at 212°, is as high in iron as the heavier red-colored ore, and its discovery in a hole is little or no guide to the probable depth of that hole, although it

is a fact that this class of ore is found more frequently near the base of the beds than in the higher levels.

It is very important, for this and many other reasons, that any serious attempt at the examination of these ore fields be assisted by a chemist in the field. About 2,500 samples were analyzed during the course of my work on this examination, most of them in a field laboratory. It was impossible to maintain an equipment in the hills sufficient for the determination of chromium, nickel, phosphorus and the like, but all iron assays were made there, and were kept as close to the daily returns from the drillers as was practicable. With the crude equipment at hand, one chemist, assisted by two Spanish grinders from the district, assayed as many as 50 samples in a day. Our laboratory was housed in a palm-thatched hut, one side open to the breezes, with its foot-thick roof inhabited by snakes, scorpions and rats, and with myriads of flies, fleas and gnats swarming about us as we carried on our calculations or weighed out our samples. The NE. trade winds that come into such a laboratory after sweeping over thousands of miles of sea are freighted with dampness, and it was found that a slight delay in weighing a dried sample caused it to absorb moisture so rapidly as to affect the results. So careful were my selected native assistants in their work of marking samples, both in the field and in the grinding shed, that of all the samples brought in for analysis less than half a dozen were marked or misplaced.

This limonitic ore carries an excessive amount of hygroscopic moisture and is light in weight, varying between 18 and 21 cu. ft. to the ton. At an average of 20 cu. ft., which has been assumed as a safe unit for computation by all explorers in that field, the ore will weigh 5,382 tons per hectare-foot. When the area runs into thousands of hectares, and the average depth to more than 18 ft., it may be seen readily that the estimated tonnage will give an enormous aggregate.

The presence of nickel and chromium has been noted. The former is found in quantities increasing towards the floor of the deposits. In the analyses of several hundred samples for this element, the highest percentage found was 1.28 and the lowest 0.44, with an average not far from 0.80. I need not emphasize the economic importance of an iron ore averaging 43 per cent of iron, and carrying 0.80 per cent of nickel. Several hundred tests for chromium showed an average of 1.75 per cent, a serious matter if it were not that a simple metallurgical process will eliminate this element at one stage of the reducing operation. These ores are of Bessemer grade, slightly lower in silica than the average Mesabi, and not higher in kaolin than some Mesabi ores. Phosphorus exists in very slight proportion. Sulphur is negligible. At Felton, on Nipe bay, the Spanish-American Iron Co. operates a large works for the beneficiation of this ore by drying it in cylindrical, rotating, horizontal kilns heated to a high degree, which reduces, by 33 per cent, the weight of raw ore charged. Against this cost of nodu-

lization, which may be given at about \$1.25 per ton of product of the kilns, are to be placed the saving in freights and duty, and the advantage to the furnace man of receiving a partly prepared material for treatment.

With no over burden to be removed, the deposit situated close to the sea, with stream valleys cutting through the ore beds and running directly to deep water, and with an average thickness suitable for about one shovel cut, these ores should be mined at low cost by ordinary steam shovel. The steam shovel is referred to here as though its advantage for this work were unquestionable, but this is not so certain, since some other type of machine excavator may be better. The drag-line excavator has been tried, and has advantages, especially if the deposit of ore is comparatively thin and the floor quite rough. Also, its radius of action is far greater than that of a steam shovel, which must be moved frequently. There is no question of the relative efficiency of the two machines if the shovel can get one or two full cuts in clean ore, but such opportunities are comparatively rare. One block of 75,000,000 tons, assaying several percentages better than the average of the district and of a thickness of about 70 ft., can be connected with deep water by a railway 4,000 m. long, without excessive gradients. Ore so situated can be delivered on board ship at an actual operating cost not to exceed 20 cents per ton. The average cost of mining and rail transport to the sea for the entire tonnage in sight should be but little more than this amount, if operations are conducted on a scale of magnitude commensurate with the importance of the undertaking.

Iron ore is transported from Cuba to American Atlantic ports at 85 cts. per ton. It is carried in British and Norwegian tramp steel ships of from 3,500 to 6,000 tons cargo capacity, usually equipped with two cargo hatches forward and two aft. These vessels do not compare with the great lake freighters of from 10,000 to 13,000 tons capacity and with from 20 to 33 hatches. To be sure, a lake ore carrier would not live in the weather to which these boats are subjected, but there is no doubt that reasonably large staunch carriers can be so constructed as to afford rapid loading and unloading at each end of the route. With a ship of this trade, more money could be made at 70 cts. per ton than the lake boats make at 85 cts. per ton. Adding duties at 75 per cent of the foreign import rate, incidentals, administration expense, nodulizing, and all other charges, a nodulized 54 per cent. Bessemer ore can be delivered from these mines at American Atlantic ports at a cost of about 5 cts. per unit of iron, and at Pittsburg at a cost of 8 cts. per unit of iron, the additional 3 cts. being due to the freight from the seaboard to Pittsburg. The raw ore can be delivered at the same points at 3.5 and 7.2 cts., respectively. Of course Lake Superior ores have a counter and equivalent advantage at Pittsburg as this Cuban product has at

ocean ports, due to the cost of the rail haul between that city and the sea.

Various important steel making concerns are interested in the Moa region. The U. S. Steel Corporation has a number of men in that field; the Pennsylvania Steel Co. and the Bethlehem Steel Co. are also well represented by their subsidiary companies—the Spanish-American Iron Co. and Juragua Iron Co.; and other eastern and western interests are identical with the field. No mining has as yet been started at Moa, but it is probable that operations will not long be delayed.

THE ELECTROLYTIC SYSTEM OF AMALGAMATING GOLD ORES.*

By ELMER ELLSWORTH CAREY.

I am convinced that a revolution in mining methods is imminent, due, to a great extent, to the work of those engaged in research work along electrochemical lines. In submitting this paper, I will have the opportunity to refer to a very valuable paper, read at the Dec. 16, 1910, meeting of the New York Section of this society, by Mr. John Collins Clancy,† on his electrochemical cyanide process. Some comments on Mr. Clancy's paper appear below.

The increasing demand for gold has turned the attention of miners and metallurgists to new fields and new methods, and every year brings down the cost of mining and milling; new extraction methods are being tried, and every effort is being made to save values in low grade and refractory ores. The tailing piles of yesterday are now being reworked; and tomorrow the tailings piles of today will yield further values, as the march of progress discloses new methods, systems and appliances.

The favorite ore of the miner is the so-called "free milling." In such ore the gold particles are comparatively large, and are generally imbedded in quartz ore, free from sulphur, arsenic, etc. Values in such silicious ores may be recovered by the standard system of amalgamation. Then there are other ores where crystals of iron pyrites (sulphides) are found in the quartz, and within the pyritic crystals may be found gold particles in a finely divided state. The values in such crystals cannot be recovered on the usual mill plate, as the particles of gold are coated with various substances, preventing amalgamation. The present practice is to separate the pyritic crystals from the crushed ore by various types of concentrating tables. This so-called concentrate is then sent to a smelter, or it is ground in tube mills (100-200) mesh and delivered to the cyanide tank, where the extraction ranges from 85 to 96 per cent.

Concentrates can only be sent to a smelter where the values are high enough to pay transportation and reduction charges; the cyanide process is expensive

*Paper presented at the 19th General Meeting of the American Electrochemical Society, in New York City, April 6-8, 1911.

†Mr. Clancy's paper was printed in our last issue.—Editor.

and unsatisfactory, unless we admit Mr. Clancy's claim that he has solved the cyanide problems by his system of electrochemical cyanidation.

Another method calls for fine grading (100 mesh) and electro-amalgamation. An ideal plant for this process consists of some type of rotary crusher, with outside screening, possibly a secondary crushing device to finish the work of the first crusher, and from the crusher the pulp, ground sufficiently fine to release all economic values, is passed over electrolytic amalgamating devices. The released values are recovered in the form of amalgam. Concentration, cyanidation and smelting are unnecessary and the metallurgy of gold is reduced to its lowest terms.

The theory of electrochemical amalgamation has been before the mining world for half a century; one of the earliest authoritative papers on the subject may be found on p. 205, of Vol. I of the Proceedings of the (London) Institution of Mining and Metallurgy. Early investigators found as many difficulties as the pioneers in the art of aviation. With no information, no reserve of text book knowledge, no authorities to consult, no works on electrochemistry, it is not strange that the electrolytic method of gold recovery made slow progress. All the problems of current density, anode troubles, forms of construction, etc., had to be laboriously and expensively worked out. To obtain working data regarding the system will cost an independent observer several thousand dollars and a year or two of time; besides he will draw heavily on his stock of good nature, patience and perseverance.

In electrolytic amalgamation, the sole function of the electric current is to deposit hydrogen, sodium, potassium or ammonia in the mercury. The sole function of sodium or potassium is to de-oxidize water, and the final work in the chain of reactions is the liberation of nascent hydrogen at the surface of the mercury. As the particles of gold sweep over the electrolytically excited mercurial surface, all substances usually preventing amalgamation are automatically and almost instantly destroyed or rendered inert. Grease is saponified by the caustic soda; oxide coating on gold particles is reduced by hydrogen; in a word, gold particles are cleaned and amalgamation quickly follows.

It has been known for some decades that in some way amalgamation was intensified in the presence of the electric current; the problem was to devise a machine to utilize the electrical action to the best advantage. Many complicated devices for this purpose have been patented, but as is usually the case, the successful apparatus is simple—almost childishly simple. So simple is the device that extracts all released values, and so broad are the claims made for it, that the builders of our metallurgical temple have persistently rejected this stone. I will follow the scriptural allusion no farther, but I earnestly suggest that in dealing with any difficult extraction problem, the claims of electro-amalgamation be considered.

Hungarian mercury wells are described in Vol. II of Prof. R. H. Richards' Work on Ore Dressing; by making the baffle in such wells an anode, using suitable material, and employing an 8 to 12 volt current of high amperage, we have an electrolytic amalgamating device. The objections to the use of such mercury wells disappear when they are electrified, and their extraction efficiency is greatly increased. The interior of such wells must be lined with some very refractory substances. The ideal amalgamator provides a method by which the gangue is brought closely and intimately into contact with a mercurial surface, which acts as a cathode; provide these conditions, and arrange for suitable capacity and durability, etc., and the low grade and refractory ores of the West soon yield their gold.

Today the mining industry is at a standstill because new conditions call for new methods. The old style mining engineer must give way to the metallurgical and chemical engineer. There are no problems in metallurgy that cannot be solved by the application of the proper electrochemical principles. Electrochemistry has devised profitable methods of extracting the useful metals from other ores, and electrochemistry will also find methods for releasing the noble metals from Nature's refractory grasp. There is more gold in the West than has been mined. Lying in plain view in tailing piles, low grade veins, desert sands, beach and river deposits of the West, there is sufficient gold awaiting the electrochemical engineer to pay the combined national debts of the world. There will be no great advance in the mining industry until the technical schools have furnished a supply of electrochemical metallurgists; for the problems of the mining industry today can only be finally solved by the electro-metallurgist.

In Mr. Clancy's paper above mentioned, the author takes occasion to speak rather disparagingly of the electrolytic system of gold recovery. He states: "One of the most severe criticisms on electrolytic processes is where the direct precipitation of the values from the ore pulp is concerned, the objections probably being due to the scouring of the plates by the circulating ore and the consequent loss of finely divided gold amalgam in the ore pulp."

With the use of the electrolytic sodium-mercury cells, as above mentioned, there is no scouring of the plates, and no loss of finely divided gold amalgam; on the contrary, mill pulp after passing the electrolytic sluices, contains only encased values. The aqua regia test shows no free values. It is true that electrolytic plate amalgamation, to be successful, requires considerable experience in the art, but it is possible to arrange electrolytically excited amalgam plates so that the dire results mentioned by Mr. Clancy do not occur. Electrolytic amalgamation will give extraction results equal to electrolytic cyanidation; and when cost of installation, operation and maintenance are considered, all comparison ceases. Scores of investigators in the last 25 years have given testimony as to the efficiency

of electrochemical cyanidation; the method of regenerating the cyanide solution and the recovery of values not amenable to the straight cyanidation have frequently been referred to in the technical and mining journals. Mr. Clancy deserves credit for calling attention anew to the utility of electrolytic lixiviation, and his standing is such that many engineers will doubtless turn their attention to electrolytic processes. However, I regret that Mr. Clancy should have found it necessary to disparage electro-amalgamation, which is now slowly being recognized by progressive mining engineers as an important factor in milling. But let no one imagine that he can run a few wires from a power line into a solution tank and have a successful electrolytic cyanide plant; and let no one imagine that he can connect his battery plates with a low voltage generator and thereby greatly increase the extraction. This would be on a par with the village mechanic who purchased working drawings of an aeroplane and set out to build a flying machine.

A simple, but very interesting experiment which may throw some light on the wonderful activity of hydrogen-sodium amalgam and the value of electrolytic amalgamation is made by placing a piece of plastic sodium amalgam the size of a pea in a test tube and adding an ounce of water heavily saturated with ammonium chloride. Test the amalgamating powers of the very curious resultant amalgam by copper and iron wire.

Those who wish to investigate the records of the patent office for progress in electrolytic amalgamation will find the various types of electric amalgamators described in the following U. S. patents:

526,099	579,211	592,793	418,134	370,366
492,711	669,058	548,265	307,081	285,523
641,360	590,524	328,532	757,557	947,958

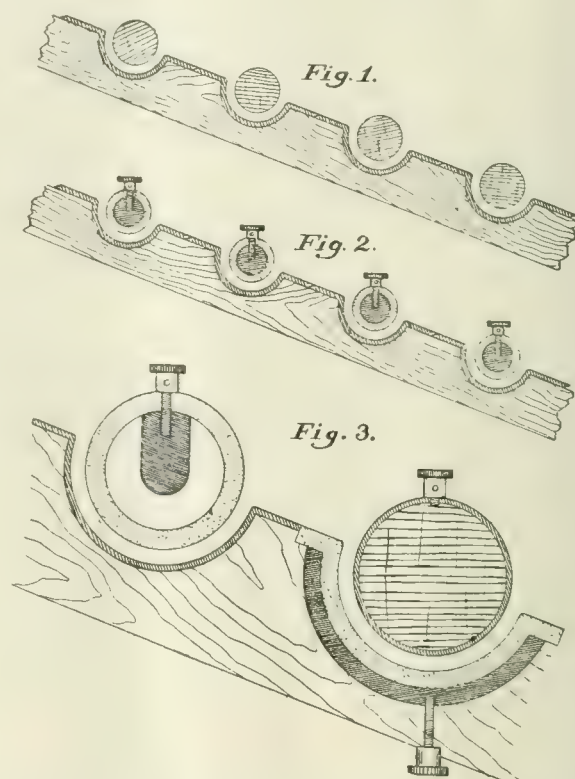
Copies of the above patents may be had from the Commissioner of Patents, Washington, D. C.

An electrolytic amalgamating apparatus should be so constructed that there is a constant and regular electro-deposition of sodium (or some similar element) in the mercury, and provision must be made for passing auriferous pulp or sand over the mercurial surface, so that every particle of gold is forced into intimate contact with the mercury. With the automatic deposition of sodium, and suitable mercurial contact, no free values can possibly escape; and the ideal milling plant of the future will consist of some type of rotary crusher producing a 100-mesh product, followed by a series of electrolytic amalgamators. Such an arrangement will extract all values shown by any free gold test.

The ideal electrolytic amalgamator consists of a series of electrolytic sodium-mercury cells, followed by another electrolytic amalgamating device in which the values are recovered on silver-plated copper plates, of suitable construction. With the mercury wells, high amperage may be employed, liberating large volumes of hydrogen, which removes all coatings (oxides, sulphides, grease, talc, silicious coatings, etc.) from the

microscopic gold particles; in the secondary amalgamator the plates offer a large cathode area so that all portions of the pulp are forced into contact with a highly active mercury surface. The supplementary amalgamating device above mentioned consists of a silvered copper plate of suitable width; this plate contains parallel, transverse, semi-cylindrical depressions (grooves, pockets or riffles) the full width of the plate; into these grooves project cylindrical terra cotta cylinders, leaving a quarter-inch clearance, from three to four inches in diameter; these cylinders contain a graphite core, one inch in diameter, connected as an anode.

The pulp passes under the cylinders, and sweeps gently over the curved (cathode) amalgam plate, while gravity, the force of the water and centrifugal



force tend to drag each gold particle into contact with the highly excited mercury surface. With such a series of amalgamating riffles, supplied with a low voltage current of proper density, we have an amalgamating device of wonderful activity. It is not too much to say that this simple device will extract 99 per cent of the values which may be saved by any system of lixiviation, the cost of extraction being insignificant.

Fig. 1 shows a section of a new type of mill plate which I have devised. The sides are not represented. The silver-plated copper plate (being of the usual length of amalgamating plates) contains transverse parallel semi-cylindrical grooves into which fit solid cylindrical wooden baffles; these baffles may also be made of standard piping or casing, from 3 to 4 ins. in diameter, with the ends closed. The clearance between the plates and the baffles is from 3/16 in. to 1/4 in. One piece of copper plate may be used, with

the proper depressions pressed therein, or, preferably, a number of overlapping plates may be used, each plate containing one or two semi-cylindrical depressions. The plates are held in position by their own weight, fitting closely to the sides, and can be quickly removed from the device for cleaning up. The device may be given any desired grade, and the plates are dressed and operated as the usual mill plate. A mill plate arranged as described will make a better extraction on the average ore than the usual type of amalgam plate, there being no loss of amalgam.

In Fig. 2 is illustrated an electrolytic mill plate, constructed similarly to the one shown in Fig. 1, which will not only recover all values saved by the standard types, but in addition, all free values in silicious pulp and slimes, all values in placer material, beach, sand and all black-sand values are also recovered. The cylindrical baffles are made of terra cotta, and each contains a graphite core connected to the positive lead of a low voltage generator; the amalgam plates are connected to the negative lead of the generator.

In the first groove or riffle shown in Fig. 3, the water and pulp passes over the amalgam plates as in Fig. 2; in the second riffle of Fig. 3, the amalgam plate, connected to the negative lead forms a casing for the baffle, and the pulp stream passes *under* the mercurial surface, thus bringing the surface of the water into intimate contact with the electrically excited mercurial surface; or a copper cylinder, silver plated, may be used as a baffle, and at the same time act as an amalgamating surface. A gold saving device may consist of a series of such riffles as shown in Fig. 3, arranged alternately; such an arrangement is particularly useful in treating pulp containing gold in a finely divided form, or for recovering values in slimes or in solutions. By screening placer material to 10 or 12 mesh, and passing the undersize over an electrolytic amalgamating sluiceway of suitable length, all fine, rusty, float, coated and greasy gold is recovered.

With the standard system of mill-plate amalgamation, many difficulties are encountered; with the device just outlined, all the usual amalgamation troubles disappear. The plates may be dressed by hand in the usual manner once a day; by adding a mercuric solution to the water, the proper amount of mercury will be deposited electrolytically to keep the plates in excellent condition. Electrolytic sodium amalgam containing gold is soft, yet tenacious; it is plastic and arrests every particle of passing gold, yet such amalgam never crumbles. Such an apparatus will extract free gold from material having any kind of gangue, clayey, sulphurous, arsenious, etc., and there is no fouling, no formation of sulphide coatings, no discoloration by tellurides, arsenides, etc.

In certain classes of base ores it may be necessary as a preliminary measure to treat the pulp for 30 minutes by electrolytic pan amalgamation before passing the pulp over the electric sluice; and for ores containing gold in chemical combinations (sulphotellu-

rides, etc.) a preliminary roasting may be required.

In the near future we may look for a greatly increased production of gold, due to the application of electrochemical methods in mining and milling operations; were electro-amalgamation and electro-cyanidation today in general use in other mills now in operation, the gross output of gold would be increased 25 per cent. The great increase in the future supply of gold, however, will come from vast low-grade deposits and ledges which cannot now be commercially mined. A new field containing fabulous treasures awaits the command of the modern Aladdin—the electrochemical engineer.

THE CONSERVATION OF OUR METAL RESOURCES.*

By ALBERT E. GREENE.†

My theme before this congress is the conservation of our metal resources. Just as we are conserving the timber in the forests; utilizing the land we have heretofore called desert; using the exhaust steam of steam engines and greatly increasing their power; so also in the metallurgy of our base metals we are beginning to conserve a 1-10 of a per cent of metal here, a 1 per cent of metal there which have hitherto gone to waste. Most of you are aware of the great progress made in recovering copper or silver or gold out of low grade ores that only contain a very small percentage of these metals. Now in another field there is a loss which has been going on before our eyes with apparently no attention and it is a loss which amounts to millions of dollars per year. It is effecting not only large corporations but it is effecting small producers even to a greater extent. The loss to which I refer is the loss of metal in the processes of making steel and other metals from ores.

In the production of the 20,000,000 tons of steel per annum in this country there is a loss of metal by oxidation during the conversion process which probably aggregates 1,000,000 tons. Most of this metal is lost in the slag and some of it can be reduced again by smelting the slag in the blast furnace, but in the Bessemer process a large part is blown out of the vessel in fine dust which is difficult to collect. The loss of metal by oxidation in the Bessemer process alone probably aggregates over 500,000 tons per year.

Another significant fact is that a very considerable part of the metal lost by oxidation consists of elements which are more valuable than iron—such as manganese and silicon. These elements are usually required by specification in the finished steel and although the iron ore usually contains enough of them, and the blast furnace process usually reduces them into the iron in sufficient quantity to meet specifications without any

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additions, yet in the Bessemer and open-hearth processes they are almost invariably oxidized again and practically lost. When one remembers these facts and that it costs to reduce these metals into the iron and that after they are oxidized out they have to be replaced by expensive alloy additions to the steel—it seems as though this were an excellent opportunity to apply conservation theories to advantage.

My object in this paper is to point out how such losses as these can be and are beginning to be diminished, and another object is to show how the application of physical and chemical principles to these problems is one of the greatest aids we have in accomplishing this end. If I can do this I shall feel that, in some small way at least, I shall have served my alma mater, to whom I feel very greatly indebted.

It has been my privilege to have followed one of the newer fields of development influencing the conservation of our metals. This is the field of high temperature chemistry which the advent of the electric furnace has opened up. It has been said by prominent steel men that in the next ten years the greatest developments in that industry will be those effecting increased metallurgical efficiency. I venture to say that in this one industry by the application of electricity together with improved chemical processes there will result a saving of metal which will amount to hundreds of thousands of tons per annum.

I wish to try to show how the electric furnace can influence conservation in such a way. It is not simply because the electric furnace provides a means of getting extremely high temperature but rather because by means of electrically developed heat we can simultaneously control the temperature and the chemical reactions wholly independently of one another. This is a most important fact, and it opens up a whole new field of chemistry, especially where the gaseous treatment of metal is involved.

For sake of comparison let us consider the methods of heating now in use. Almost invariably heat is obtained by the combustion of fuel. The fuel is usually burned in the same chamber with the material heated so that gaseous products of combustion come in contact with the material. These products of combustion are themselves of an oxidizing nature since they always contain oxygen, and furthermore, to obtain high temperatures efficiently, it is necessary to burn the fuel with excess of oxygen which makes the atmosphere still more oxidizing. Thus we see that as a general thing high temperatures are obtained only under more or less oxidizing conditions whether such conditions be desired or not. For example where steel scrap is melted in an open-hearth furnace the steel takes up more or less oxygen in some form or other and the melted steel must be deoxidized by use of silicon and aluminum or other agents. Here much metal could be saved if the atmosphere could be controlled. In another example such as the open-hearth process where pig iron is being converted into steel, the oxidizing atmosphere aids in oxidizing the carbon but the trouble

is that it oxidizes the metal, too, and this is the important cause of the great loss of metal referred to above. Even though the atmosphere may be modified slightly by allowing the gas to enter the furnace chamber next the charge and beneath the air, yet this does not prevent the loss by oxidation. And so we are practically driven to use the electric furnace if we desire to control temperature and atmosphere independently of one another.

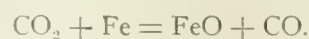
Let us now see what can be accomplished by means of such independent control of temperature and gases and let us consider in this connection the meaning and use of the term "neutral atmosphere."

One often hears of maintaining a neutral or a non-oxidizing atmosphere in an electric furnace to prevent oxidation. If we have a bath of steel and an iron oxide and lime slag on top of it, a neutral or non-oxidizing atmosphere would not necessarily prevent oxidation of elements in the steel. A neutral atmosphere would be in equilibrium with iron and an oxide of iron. If we are going to make use of the atmosphere to prevent oxidation it must be controlled and not kept simply neutral. It ought to be reducing in certain cases and oxidizing in others. And this leads me to the consideration of an idea whose application in practical processes I believe is new. It is this: We can by controlling the temperature and also the composition of the atmosphere acting on a charge at a given temperature, effect a reduction of one substance and an oxidation of another substance at the same time by the same atmosphere. As far as I am aware, the application of this idea to the oxidation of an impurity out of a metal without oxidation of the metal itself and even with the reduction of the metal oxides is new. Its application to the conversion of iron into steel makes it possible to oxidize the carbon without the very great loss of metal I have already referred to.

Inasmuch as I was led to a clearer conception of such a process as this by a careful study of the chemical principles involved, it may be of interest to apply one or two of these principles to a simple case, for it is the application of these principles which I believe is of great value in improving the efficiency of such processes.

Suppose we are working with pig iron and can react on it in a furnace chamber with an oxidizing or reducing gas or mixtures of these gases and at any temperature and pressure. The physical conditions to be controlled in such a case are the temperature and the pressure of gas in the furnace chamber; the chemical conditions are the reactions between the different elements and compounds present.

The two laws particularly applicable in this case are the Mass Action Law and the Phase Rule. Let us choose a simple reaction which might take place in the above example. The CO_2 gas present may act on the iron as follows:



The law of Mass Action tells us that if such a reaction goes on in a closed vessel at a given elevated temperature and pressure, a condition of equilibrium will be reached and the CO produced will bear a certain relation to the CO₂ present. For the given temperature and pressure the ratio of the amount of CO and CO₂ will have a particular value. The Mass Action Law also tells us that if we force more CO₂ into the closed vessel the oxidation will proceed further and if CO is forced into the space the oxidized iron will be reduced.

Now suppose we replace the mixture in the vessel by a mixture of CO and CO₂ in which the CO is in excess of the amount present under the equilibrium conditions just referred to; the result will be that some iron oxide will be reduced and CO₂ formed, and equilibrium will tend to be restored, but the amount of solid or liquid iron oxide remaining will be less than at the start. We can make this replacement continuous so that the gas in the chamber is always reducing toward iron oxide and we therefore have a means of carrying the reduction of iron oxide to completion and preventing the further oxidation of iron, it being remembered that the amount of solid FeO present does not influence the equilibrium. And furthermore this can be done with a gas containing the oxidizing agent CO₂. By keeping enough CO₂ present it is possible to oxidize carbon without oxidizing iron, since carbon oxidizes more easily than iron at high temperatures.

With respect to temperature and what the control of it enables us to do, the Phase Rule tells us among other things what the effect of temperature will be on the equilibrium we have just considered. It tells us that the tendency of oxygen to separate from iron oxide has a certain definite value for every temperature. Increasing the temperature increases the tendency to separate or dissociate. This tendency is called the dissociation pressure of oxygen for the particular oxide and it has been measured for a few compounds. For our purposes, at present at least, we can and must get along without knowing what these actual pressures are because they have not, except in a few cases, been determined; it serves our purpose, however, if we know what ratios of reducing gas to oxidizing gas will prevent the undesired oxidation for given temperatures. Thus we need only to know, first, what ratio of CO is needed with a given per cent of CO₂ in a gas in order to prevent oxidation of iron at a given temperature and second how this ratio changes with the temperature.

The Phase Rule tells us the same thing about other elements, for example silicon, manganese, carbon, phosphorus, etc., these being the ones with which we are concerned in steel processes, and we find that at a given temperature these different elements as a rule have different tendencies to oxidize. Since the temperature affects different elements differently we find also that at certain temperatures two elements may

have the same tendency to oxidize, and the so-called critical temperature of the Bessemer process when both silicon and carbon have an equal tendency to oxidize is one of these temperature points.

To sum up a few of these considerations we have seen how by continuously controlling the atmosphere acting on a given charge we may carry out any particular reaction to completion or how we may prevent a given reaction for an indefinite time; how an atmosphere which is reducing to one metal at a given temperature may not be reducing, but may even be oxidizing, to another element, and that the term "neutral" is indefinite unless something more is specified.

And finally I think you will see how the electric furnace, by reason of enabling us to control temperature and, at the same time, oxidize one element while keeping another reduced, makes it possible to do what the present steel making processes cannot do, that is, oxidize carbon without the tremendous loss of other elements I have already referred to. It enables us to oxidize carbon and not iron or manganese or silicon; it enables us to oxidize phosphorus without oxidizing iron or manganese; it enables us to separate iron from copper by oxidizing the iron without oxidizing the copper; and these are only a few of the things that the combination makes possible.

The important question is: Can such processes be carried out economically, can they compete with present processes? When we started to develop these processes we had first worked them out theoretically and we, too, wondered if they would really work in practice. We carried out an extensive series of tests on a small scale and as these tests proved that it was possible to accomplish the oxidation of one element without oxidizing others, we then designed and built larger furnaces to try this process on a practical scale. Our first practical furnace was of about 300 lbs. capacity for making steel for castings. I have just presented a paper before the American Electrochemical Society at New York, on the 5th of this month, which goes into more detail in regard to this process, but it may be of interest to make a brief comparison with the converter process used by many small steel foundries in making high-grade steel.

In the converter process, molten low-phosphorus pig iron is poured into the converter and blown with a blast of air usually through tuyers near the level of the metal, the air burning out the silicon, manganese, carbon and considerable iron. In these small converters such as used in steel foundries the loss is very high; it is often more than 18 per cent of the weight of the molten pig iron started with. True about 4½ per cent, which goes to make up that 18 per cent, is carbon and silicon which have to come out anyway, but there remains a loss of 13 or more per cent which is so much waste of metal. In a plant making only 20 tons of molten steel per day, the value of this 13 per cent of steel (for it would be steel if saved) would at \$30

per ton amount to over \$23,000 per year, and in this loss is included manganese and silicon which have to be put back again by adding alloys, and their addition entails further losses. These alloys, usually ferro-manganese and ferro-silicon, serve two purposes; they partly combine with the oxides in the iron which have come from the air blown through the iron and separate out as slag, and part of them remains in the steel and raises the percentage of these elements enough to meet the specifications.

The process we have developed and which we are calling an electric convertor process is carried out in an electric furnace instead of a Bessemer convertor, and the molten metal is blown with gas of regulated composition instead of with air. The vessel must be electrically heated because the reactions of the gas on the metal do not supply much heat, as the corresponding reactions in the Bessemer process do. We use a gas which may be obtained from a cupola, or a gas producer or from a blast furnace. This gas would contain 6 or 8 per cent of carbon monoxide, CO , and 12 or 15 per cent of carbon dioxide, CO_2 . When blown into molten pig iron this gas does not oxidize iron or manganese and as pig iron usually contains manganese in sufficient quantity to meet specifications, this process avoids replacing that element by expensive alloy additions. The metal is heated to about $1,450^\circ$ Centigrade and blown with this gas and the carbon is taken out by it. The carbon is oxidized, but not the iron nor the manganese. There is a small loss of manganese by vaporization, but this is so small that after the carbon is out, the percentage of manganese may even be larger than at the start.

In the small furnace which we have used to make various grades of steel, we have been able to convert pig iron into steel with a total conversion loss of only about 2.5 per cent plus the weight of carbon burned and including metal spilled in handling and left in the furnace and this compares with a total loss of 15 per cent plus the carbon burned by the Bessemer process in small converters. This furnace was heated by induced electric currents in the metal. The process is now installed and under test in a two ton furnace, and we believe that the developments we have made are a step toward the saving of that immense loss which is going on today.

One of the most interesting applications of the process has been in the production of manganese steel. As is well known, manganese steel containing about 12 per cent manganese has very valuable properties of strength and resistance to wear, which render it very useful for such articles as railway crossings, rails, crusher-jaws, safes and many similar things. The scrap steel such as heads and gates, etc., from castings made of manganese steel contains a valuable amount of manganese but in the melting up of manganese steel scrap in cupolas much of the manganese is lost, and when the melted mixture of scrap and iron is blown

in the converter, all the rest of the manganese not already lost is oxidized. This tendency of manganese to oxidize has made it practically impossible, at least from a commercial standpoint, to make low carbon manganese steel and the properties of such steel are almost unknown. By applying our process it has proved practical to melt manganese steel scrap into pig iron and then remove the carbon without practically any loss of manganese, and it has further proved practical to produce very low carbon manganese steel in this way. The commercial saving resulting from the utilization of the manganese that has heretofore gone to waste is a most important consideration to the manufacturer and this saving can be accomplished at a cost very much less than the value of the metal saved.

These examples of losses in steel processes are characteristic of similar losses in the metallurgy of various other metals; in the process of converting copper matte for removing the sulphur and the iron from the copper there is an oxidation of copper which totals up very high. Likewise in the metallurgy of lead there is a similar loss and in the case of many other metals. These losses can be prevented by the use of electric heat for controlling temperature and simultaneously controlling the composition of the gaseous reagents.

The commercial practicability of such processes depends on many factors but we believe that the losses which can be prevented will in a great many cases prove to much more than offset the cost of electric heating.

The field of chemistry opened up by the electric furnace is a most fertile one from the standpoint of scientific research as well as on the practical side. We need exact knowledge about all the metals and elements concerned in high temperature metallurgy, such as the dissociation pressures of various metallic oxides and the variation of these pressures with the temperature. Such data will be of the greatest value to the manufacturer so that there shall be an additional incentive to its collection by scientific workers. By means of such data and by the application of simple physical and chemical principles in practical processes I believe the chemist has in his power one of the greatest means today of aiding the cause of conservation of our metals.

And to our Institute of Technology, for the things she stands for; for what she has done for us and is doing for many others today, in training, in preparing us to put up a good fight for things worth while; for her inspiration and encouragement in helping us to see even in a small way how the tasks she placed before us would help in time to come, for all these gifts I feel that I owe a debt to our Alma Mater that only the greatest effort can pay even in part. For the honor and privilege of being one of her sons I am profoundly thankful.



FUSIONS.

By H. C. PRITHAM.

It is not the writer's intention to claim anything new in the following article, but merely to give his method of fusing any material for analysis in the laboratory. From observation I have found few chemists using any such method and furthermore have had the pleasure of instructing quite a number of very excellent analytical chemists in the details of the same.

I have noticed in visiting many laboratories from Boston to San Francisco that the majority use the old method of fusing at a high heat and then running the fusion up around the sides of the crucible. Some allow the fusion to settle and cool somewhat, then put the bottom of the crucible in water and allow the button to "pop" loose. While I do not question the accuracy of such methods, I do claim that there is an unnecessary waste of time. In a commercial laboratory or a factory one for that matter, time is costly—very costly. The most insistent demand on the head chemist is "When can we have results?" and the telephone is kept busy until they get them, too.

Any method of running a fusion around a crucible to cool takes time and care to perform the operation and in the subsequent digestion in hot water the fusion dissolves slowly and if, as sometimes happens, considerable of the fusion collects in a hard mass in the bottom of the crucible, it certainly takes time and patience to get it dissolved out. With the following method all this is avoided. The crucible is almost entirely clean from the fusion and the dissolving of the bottom is very fast. The minimum of time is required to transfer the liquid fusion to the digestion in hot water. Many times in a commercial laboratory or a cement plant one has to fuse 20 or 30 samples of clay or other material in the morning and as a rule has about four platinum crucibles to do the work in. The sooner the chemist can make a fusion and get a clean crucible back the more time he will have to sleep that night. Moreover the wear and tear on the platinum is much less. The sudden chilling of a crucible in cold water or the squeezing in of the sides in order to loosen a fusion is not beneficial; the platinum gets brittle and loose. Considering the present price of \$43 an ounce it is of interest to take care of the crucibles.

My method is to take a shallow 5-in. evaporating dish and bend up a clay triangle so as to set in the center of this. In the triangle put a 50 cc. nickel crucible so that it is held solid and leave about an inch space between the bottom of crucible and dish. Fill the dish

with cold water up to the triangle and you are ready for business.

When the fusion is finished raise the heat for a minute or two, remove the cover and at once pour the fusion into the nickel crucible by taking up the platinum crucible in the forceps on one edge. This can be done very quickly and easily with practice. Almost all the fusion will pour out and will solidify at once. Have a casserol ready half full of hot water, pick up the nickel crucible in the forceps, tap it lightly on the desk, and dump the button into the casserol. Keep the casserol covered. The action of the hot water on the hot button will disintegrate it. Add the platinum crucible and cover and after a few minutes take them out and rinse off. If necessary, rinse in acid into a beaker using this acid when the fusion is made acid for evaporation. Acid should not be added, however, until the fusion is well dissolved. The button will leave the nickel crucible clean, but if desired it can be rinsed in hot water. If this is done be sure to have the crucible perfectly dry before pouring the next fusion. A little water will cause rather startling results when the hot fusion is poured in on it.

In conclusion I would call attention to the fact that a blast lamp is not necessary for fusions, contrary to most textbooks. A good Bunsen will give all the heat needed except in a few iron ores. The heat should be low until the fusion is well under way. In no case should the practice of turning on a blast-lamp for fifteen minutes be put up with.

The German sales of potash for 1910 are valued at \$36,500,000, against \$13,600,000 in 1900, \$6,500,000 in 1890, and \$4,300,000 in 1880.

An article in the German Medical Weekly gives a statistical survey of the medical profession for 1910. Those practicing numbered 32,449, or 5.01 doctors for each 10,000 of population, an increase of 480 over 1909. A still further growth may be expected, the number of medical students having risen from 9,239 in 1909 to 11,125 in 1910. The increase in number of physicians was mostly in the cities; of the 480 additions, 329 went thereto. For each 10,000 inhabitants Berlin has 12.32 doctors, Wiesbaden 22.6, Munich 16.6, Stuttgart, 9.7, Dresden 9.2, Leipzig 8.3, Chemnitz 5.6, Plauen 4.8, and Gelsenkirchen 4. The number of female doctors has risen from 55 in 1908 and 69 in 1909 to 102 in 1910; of these Berlin has 32, Munich 6, Frankfort 6, Dresden 6, and Hamburg 4. The number of female medical students has risen from 371 to 512.

A VOLUMETRIC METHOD FOR ANTIMONY IN ALLOYS.*

By GEORGE S. JAMIESON.†

The object of this paper is to describe an application of L. W. Andrews' iodate method¹ to the determination of antimony in alloys, particularly "hard leads" and solders. The method is satisfactory because it is not interfered with by copper and iron, metals which frequently occur in small quantities in these alloys, and because it is rapid and accurate.

Two other iodimetric methods have been compared with this by the writer with less satisfactory results. The first, which is based upon getting the antimony into the pentavalent form in dilute hydrochloric acid solution, adding potassium iodide and titrating with sodium thiosulphate, gave good results in the absence of copper and iron, but was unsatisfactory in the presence of these metals. The other method, depending upon getting the antimony into the trivalent state in a sodium bicarbonate solution and titrating with iodine was found to give poor results, particularly with alloys containing much lead, as even when tartaric acid is used, the lead carbonate precipitate appears to hold antimony, thus causing low results.

Andrews showed¹ that his iodate method was satisfactory for antimony by applying it to pure tartar emetic. The method has been further tested by the writer by dissolving weighed quantities of Kahlbaum's pure antimony in concentrated sulphuric acid and titrating it as described beyond. It is to be observed that this titration is carried out in the presence of 15-20 per cent of actual hydrochloric acid, which gives iodine monochloride as the end-product, according to the equation $2\text{SbCl}_3 + \text{KIO}_3 + 6\text{HCl} = 2\text{SbCl}_5 + \text{KCl} + \text{ICl} + 3\text{H}_2\text{O}$. The potassium iodate solution which was used throughout the entire investigation contained 3.5667 grams of KIO_3 per liter, corresponding to 0.00400 gram of antimony for 1 cc.

The following results were obtained:

Sb. taken.		
Gram.	KIO_3 used.	Sb found.
0.1000	24.85	0.0994
0.1000	24.90	0.0996
0.0490	12.20	0.0488

The following method of analysis for alloys has been worked out: Take 0.5 gram of alloy in the form of drillings or chips² in a 200 cc. Erlenmeyer flask. add 10 cc. of concentrated sulphuric acid and heat

until the alloy is entirely decomposed.³ Boil the solution gently for about 2 minutes after the lead sulphate has become white, allow the solution to cool to room temperature, dilute with 15 cc. of cold water, allow to cool somewhat, add 15 cc. of 1 : 1 hydrochloric acid, shake thoroughly and filter off the lead sulphate on a Gooch crucible, washing with small quantities of the same hydrochloric acid. Transfer the filtrate to a glass stoppered bottle of about 250 cc. capacity, add 5 cc. chloroform, 15 cc. of concentrated hydrochloric acid,⁴ and 5 cc. of iodine monochloride solution.⁵ Shake the titration bottle, let it stand for about five minutes and then titrate the liberated iodine with standard potassium iodate solution⁶ until the chloroform is just decolorized after thorough shaking, which should be repeated in about a minute to make sure of getting the true end point. To make a second titration most of the liquid may be poured off, leaving the chloroform ready for use.

It should be observed that, as Andrews¹ has shown, the strength of the hydrochloric acid solution in which the titration is made is of much importance. The directions, therefore, as given above should be closely followed in regard to the strength and amounts of hydrochloric acid used.

The following results were obtained with a number of alloys:

No.	Weight taken.	c. c. KIO_3 used.	Per cent. of Sb.	Per cent of Sb by thio-sulphate method.
1A	0.2000	6.70	13.40	13.43
2A	0.2000	6.70	13.40	
1B	0.2000	6.00	12.00	12.07
2B	0.2005	6.03	12.03	
1C	1.0000	2.70	1.08	1.04
2C	1.0000	2.70	1.08	
1D	0.5000	5.90	4.72	5.11 ⁷
2D	0.5000	5.90	4.72	
1E	1.0000	2.40	0.96	0.96
1F	0.5000	2.85	2.28	2.32
2F	0.5000	2.85	2.28	
1G	1.0000	2.80	1.12	1.16

The first two alloys are antimonial leads and the others are commercial solders, some of which were of poor quality on account of the antimony present.

The time required for an analysis was about an hour.

Arsenic is rarely present in appreciable quantities in the alloys under consideration, but it is to be noticed that, if present, it would be titrated with the antimony. The following results were obtained by dissolving metallic arsenic in concentrated sulphuric acid and proceeding exactly according to the method that has been given.

As taken.	KIO_3 used.	As found.
Gram.	cc.	
0.0100	4.10	0.0102
0.0100	4.10	0.0102
0.0109	40.60	0.1015

In a case where an alloy contains an appreciable amount of arsenic it is best to carry out the process exactly as directed as far as filtering off the lead sulphate and washing it with 1 : 1 hydrochloric acid. Then pass in hydrogen sulphide to precipitate the arsenic, pass air through the liquid for half an hour or so to remove the hydrogen sulphide and to oxidize

*From The Journal of Industrial and Engineering Chemistry.

†Sheffield Laboratory, New Haven, Conn.

1J. Am. Chem. Soc., 25, 756 (1903).

²If the alloy contains less than 2 per cent of antimony, it is better to take 1 gram or more, while with alloys very rich in antimony 0.1 or 0.2 gram will suffice.

³If the flask is covered with an inverted porcelain crucible cover, the dissolving and boiling may be carried out without the escape of any disagreeable quantities of fumes, so that the hood need not be used.

⁴To allow for dilution with the standard solution.

⁵To prepare this solution, dissolve 10 grams of potassium iodide and 6.44 grams of potassium iodate in 75 cc. of water, add 75 cc. of concentrated hydrochloric acid, then add a globule of chloroform in a glass stoppered bottle, and adjust exactly to a faint iodine color by shaking and adding dilute potassium iodide or potassium iodate solution as the case may require.

⁶If the volume of potassium iodate solution used is much over 15 cc., it is advisable to add more concentrated hydrochloric acid to keep the strength near the 1 : 1 point.

⁷Solder D contained a little copper or iron, which caused the thiosulphate method to give high results. Also alloys A and B were found to contain traces of copper.

any iron that may be present, filter, wash with 1:1 hydrochloric acid, and titrate as usual. This process was tested by the use of alloys mixed with known quantities of pure metallic arsenic with the following results:

Alloy taken	As taken.	Per cent	Per cent.
Gram	Gram.	Sb found	Sb in alloy.
0.5000	0.0050	4.72	4.72
0.9560	0.0118	4.77	4.72
0.2000	0.0460	12.00	12.00
0.2000	0.0520	12.10	12.00
1.0000	0.0097	1.04	1.08

THE DETERMINATION OF MANGANESE BY THE SODIUM BISMUTHATE METHOD.*

BY PAUL H. M.-P. BRINTON.

In looking over the tables of analyses furnished by the Bureau of Standards at Washington with its analyzed samples of steel, I noticed that the figures for manganese obtained by those chemists who used the sodium bismuthate method were in nearly all cases a little lower than the general averages obtained by all other methods. The differences were small, generally from 0.02 per cent to 0.03 per cent, but as I had noticed the same thing in using the bismuthate method myself, I had a curiosity to learn the cause of this tendency of the process. I have made a little series of analyses and experiments, the results of which may be of some interest, especially to those who have not yet adopted the method in regular work.

As is well known, the method depends upon the oxidation of manganese to permanganic acid by adding solid sodium bismuthate to the cold nitric solution of the sample. The excess of the bismuthate is filtered off on asbestos, a measured excess of ferrous ammonium sulphate solution added to the filtrate and the latter then titrated back with potassium permanganate.

The method was originated by Schneider,¹ who used bismuth tetroxide, and subsequently developed by Roddrop and Ramage,² and by Brearley and Ibbotson.³ Blair⁴ states that the sodium bismuthate method is not only remarkable in its simplicity and ease of manipulation, but that for samples containing not over 2 per cent manganese it is the most accurate process known. The purpose of my work was to check up Blair's results (and those of the earlier workers also) and to confirm, if I could, his statement as to the accuracy of the method.

My analyses were made on various samples of the Bureau of Standards steels. I think we may take the averages given out by the bureau, coming as they do from a variety of independent sources, as figures which represent the highest attainable accuracy in practical work. The methods and solutions were used, in the analysis of the steels, just as they are given by

Blair. In the ferrous ammonium sulphate solution half the sulphuric acid was replaced by phosphoric acid, as suggested by Dr. C. B. Dudley. The end point is rendered a little sharper by this modification as the color due to the iron is less intense.

The solution of manganous sulphate for standardizing the permanganate was carefully analyzed, the manganese being determined in weighed portions of the solution both as sulphate and as pyrophosphate. The results by these two methods were concordant.

The permanganate was standardized against the manganous sulphate solution, and also against pure Sørensen sodium oxalate, as well as against the "Sibley" standard iron ore issued by the Bureau of Standards. In standardizing against the ore, the iron was reduced with amalgamated zinc in a Jones reductor. The titer obtained by the sodium oxalate was identical with that by the standard ore.

Five of the standard steels were analyzed, the last one being a vanadium steel. Table I shows a comparison between the results obtained by taking the strength of the permanganate as determined by the manganous sulphate method, and by taking it as determined by sodium oxalate or by the "Sibley" ore. In these determinations the theoretical conversion factors were used: $\text{Na}_2\text{C}_2\text{O}_4$ to Mn 0.16024, and Fe to Mn 0.1967.

TABLE I.
Per cent. manganese.

Bureau of Standards averages	0.412	0.916	0.760	0.568	0.669
Found with MnSO_4 as primary standard.....	0.410	0.922	0.757	0.557	0.663
Found with $\text{Na}_2\text{C}_2\text{O}_4$ or with iron ore as primary standard	0.411	0.920	0.758	0.562	0.665
	0.397	0.892	0.732	0.539	0.642
	0.398	0.890	0.733	0.544	0.644

From these figures it is seen that the results come somewhat low if the permanganate is standardized against sodium oxalate or against an iron compound, and the theoretical conversion factor used. It is interesting to note that the results in this table obtained by the latter method agree very closely with those obtained by the analysts who used the sodium bismuthate method in the analyses published by the Bureau of Standards with these samples. (The vanadium steel is an exception.)

The results obtained by standardizing the permanganate against the manganous sulphate solution are highly satisfactory, and a great many determinations which I have made in addition to those given in the table have convinced me that the degree of accuracy there shown can be regularly attained; and the process is so simple that no unusual manipulative dexterity is required to insure correct analyses. In fact it is hard to go wrong with the methods, after the correct titer for the permanganate is once obtained.

While standardizing against a weighed quantity of manganous sulphate solution is the most accurate method known for the purpose here desired, it is also the most tedious. It requires a gravimetric determination of the manganese in the solution, and upon this result depends the accuracy of the whole process. A

*Paper read at the Minneapolis meeting of the American Chemical Society.

¹Dingl. Polytech., 269, 224.

²J. Chem. Soc., 67, 268; see also H. Ramage, Chem. News, 84,

209.

³Chem. News, 82, 269; 84, 247.

⁴J. Am. Chem. Soc., 26, 793; also "Chem. Anal. of Iron," 7th Ed., 121, et seq.

slight error made at this point introduces serious discrepancies in the final results.

The simplest method of standardizing is against pure sodium oxalate; but since the figure obtained in this way brings the results too low we must use an empirical factor. The ratio $5\text{Na}_2\text{C}_2\text{O}_4 : 2\text{Mn}$ calls for the figure 0.16024, but to obtain correct results I find that the factor 0.1656 must be applied.

Since permanganic acid has a certain tendency to decompose in dilute nitric acid solution, particularly in the presence of much ferric nitrate, it was deemed advisable to investigate the rate of decomposition. Referring to the solution after the excess of bismuthate has been filtered off and when it is ready to receive the ferrous ammonium sulphate and be titrated back with permanganate, Blair¹ says: "At a temperature of 5° C. the solution will remain unaltered for several hours, but at 40° C. fifteen minutes will show an appreciable change." The temperature of a laboratory is never so low as 5°, nor so high as 40°, so to see what danger from this decomposition would be if artificial cooling were not resorted to, I made the following series of determinations on the 0.760 per cent manganese steel from the Bureau of Standards. The temperature of room and solutions was 21° C.

TABLE II.

	Found. Per cent Mn.
1 Run strictly according to Blair. The filtration took 4 minutes, and the ferrous ammonium sulphate was added and excess titrated immediately.....	0.760
2 Process same as before, except that after filtering, an interval of 12 minutes was allowed before adding ferrous ammonium sulphate and titrating	0.758
3 Same, but an interval of 25 minutes was allowed..	0.755
4 Same, but an interval of 55 minutes was allowed..	0.750
5 Same, with an interval of 1 hour and 30 minutes..	0.740

From these figures it appears that a delay of ten or fifteen minutes is practically without effect under ordinary conditions. Twenty-five minutes causes a slight lowering of the results, and a longer time than this has a decidedly unfavorable influence.

It may be mentioned in passing that a large excess of bismuthate is inadvisable, since it not only increases the cost of the process, but also tends to clog the filter rapidly so that fewer filtrations can be made on the same asbestos felt. Just enough bismuthate should be used so that a slight excess is plainly visible in the bottom of the flask or beaker.

As regards the application of the method to iron ores low in manganese, I find the method advised by Blair, *i. e.*, treatment of the sample with 10 cc. water, 4 cc. H_2SO_4 and 10–20 cc. HF in a platinum dish or crucible, somewhat tedious; and that it requires so large an amount of platinum if many samples are to be run is a further disadvantage. I prefer to dissolve the ore, about 1 gram, in 12 cc. of concentrated hydrochloric acid in a 4-oz. Erlenmeyer flask, evaporate almost to pastiness, and then add 4 cc. of concentrated sulphuric acid. By boiling down to heavy fumes over a free flame, manipulating the flask in a holder, the for chlorides can be obtained with silver nitrate. The hydrochloric acid is so completely expelled that no test

residue is then taken up with 50 cc. of nitric acid, sp. gr. 1.135, and finished as usual. The process is quite rapid and the results very accurate. A few ores will not yield all their manganese to this treatment, so if the residue appears dark after taking up with nitric acid, it should be filtered off, fused with a very small amount of potassium bisulphate and added to the main portion. It should be noted that this same treatment of the residue may be necessary with some ores even when using the hydrofluoric method of attack. The fact that the method as above given does not eliminate the silica is not really a drawback, since the silica is obtained in a form which does not seem to clog the filter much.

Attacking the ore by fusing with sodium peroxide was also tried and it proved fairly successful. Were it possible to procure iron crucibles free from manganese this would be the ideal method. Since that seems to be impossible at present, I had to use nickel crucibles. It should be noted that some nickel contains traces of manganese, but not, I believe, as a rule. In using this method, 1 gram of ore is fused with about six grams of sodium peroxide at as low a heat as possible. A minute or two in liquid fusion is all that is necessary. The crucible and contents are treated with water in a covered beaker, and when the action is over the crucible is removed and one-third the volume of concentrated nitric acid added. The solution is boiled until all hydrogen peroxide is decomposed and everything dissolved but some manganese dioxide which has been precipitated. Ferrous sulphate is added until the manganese is reduced, the lower oxides of nitrogen are boiled off, and the analysis is finished as usual. The crucible is considerably attacked, but if the heat is kept low a dozen fusions may be made in a crucible before it is unfitted for further use. The color, due to the nickel, tends to obscure the end point with permanganate somewhat, but after a little practice the point can be readily seen.

SUMMARY.

1. Corroboration of Blair's statement that for small amounts of manganese the bismuthate is the most accurate method known.

2. When the potassium permanganate is standardized against sodium oxalate or iron, and the titer theoretically calculated, the results come out too low. A gravimetrically standardized manganous sulphate solution is the correct primary standard, but as this method is more inconvenient than the sodium oxalate method, it is suggested that pure Sørensen sodium oxalate be used, and that the empirical factor 0.1656, and *not* the theoretical factor 0.16024, be used in the conversion of the sodium oxalate figure to that for manganese.

3. The decomposition of ores by the hydrochloric and sulphuric acids method is suggested as being fully as accurate, more rapid and perhaps more convenient than the hydrofluoric and sulphuric acids method. Fusing the ore with sodium peroxide is recommended as a method suitable for refractory ores.

¹Loc. cit.

METHOD OF TESTING GALVANIZED IRON TO REPLACE THE PREECE TEST.*

By WALTER A. PATRICK AND WILLIAM H. WALKER.

Some time ago one of us¹ published a paper on the testing of galvanized and other zinc-coated iron, in which it was shown that the so-called Preece Test was unsatisfactory from many points of view, although no alternative method was shown to be equally available. This study has been continued, and it is now believed that a satisfactory substitute can be given.

The Preece Test consists in placing the piece of galvanized iron to be tested, in a solution of copper sulphate under standard conditions, and observing the number of one-minute immersions which can be made before copper in a bright adherent form will plate out on the article. The accuracy of the test depends upon the following assumptions: First, that the zinc will pass into solution and be replaced by copper at a definite rate, and hence the *time* taken to dissolve a galvanized coating will measure its thickness; second, that the galvanized coating is homogeneous and that the speed of the reaction between the coating and copper sulphate is constant, that is, the rate of solution is uniform; third, that when the iron base is reached the copper will plate out on the iron in a bright adherent film, in contradistinction to the black spongy form in which it appears upon the zinc coating, and that no bright copper will be seen until the iron base is thus uncovered.

STRUCTURE OF ZINC COATING.

The correctness of the first two assumptions depends upon a homogeneous structure of the galvanized coating. Although a detailed description of the structure of the different forms of zinc-protected iron now on the market will form the subject of a later paper, we will point out at this time that this structure is such as would almost certainly preclude the possibility of the Preece Test being a valid one.

The coating of an ordinary hot dipped sheet or wire is made up of three well defined parts, namely, the iron base or foundation upon which the coating rests, then a layer of an iron-zinc alloy, and next to this the layer of zinc. The thickness of the alloy depends upon the temperature of the galvanizing bath, the length of time the iron is in contact with the molten zinc, and upon the flux which is used. In a previous paper, one of us² stated that this layer of alloy was electro-negative to the iron. This was an error introduced by the use of measurements of absolute potential, and which we herewith desire to correct. The alloy is much less electro-positive than zinc, but is not electro-negative. Since the rapidity with which the

zinc or zinc alloy will pass into solution, and an equivalent weight of copper be precipitated in its place, is a function of the difference of potential between the two metals, it will be seen that the rate of solution must of necessity change as we pass from the zinc to the alloy. The so-called sherardized iron and some kinds of electro-galvanized iron consist so largely of an iron-zinc alloy that any test or measurement based upon an assumed uniformity in rate of solution is liable to grave error and untrustworthy.

SPEED OF SOLUTION.

When a piece of clean zinc is placed in a reasonably concentrated solution of copper sulphate, the rate at which zinc will pass into solution and an equal number of copper ions be precipitated as metallic copper will depend upon the concentration of the copper ions in the solution in the immediate vicinity of the metallic zinc. If for any reason the solution become depleted in copper ions, the speed of this interaction will decrease. This is just the condition which obtains when a one-minute immersion is used in the Preece Test. For the first few seconds the reaction is very rapid, but as the spongy copper forms on the surface of the zinc it becomes more difficult for the zinc ions formed to get away, and fresh copper ions to reach the metallic zinc, so that at the end of a minute the reaction has practically ceased. When the sponge of precipitated copper is removed and the test piece replaced in the solution action again begins vigorously but again falls off. It can easily happen, therefore, that the iron base will be practically exposed at, say, the end of the second minute, and yet no bright copper will be seen until the sponge is removed at the end of the third minute. The test will thus be classed as a three-dip piece, while in reality it is but a trifle over two-dip, and an error of 33 per cent is thus introduced. In fact, we have seen galvanized iron which showed areas of *no coating* whatever. The Preece Test would indicate such wire as one dip when in reality it was zero-dip wire.

APPEARANCE OF BRIGHT COPPER.

It has been proposed to provide against the above difficulty by using a number of ten-second immersions instead of one-minute immersions. But a new difficulty here arises. As is well known, when a piece of iron is placed in a neutral copper sulphate solution iron passes into solution and copper plates out in a bright adherent film. The appearance of this bright copper is taken in the Preece Test as an end point, and as a proof that the zinc has all been dissolved and the iron base exposed. The fundamental error here introduced can be easily shown by immersing a piece of smooth spelter or sheet zinc to a number of ten-second immersions in the Preece copper sulphate solution. In a short time a point of bright copper will appear which as the test proceeds will spread over the entire surface, and were the specimen being tested a galvanized article instead of one of pure zinc, the totally erroneous conclusion would be reached that the iron base had been

*Journal of Industrial and Engineering Chemistry.

¹Walker, Proc. Am. Soc. Testing Materials, 9, 431.

²Loc. cit., p. 432.

uncovered, and the test concluded. This layer of bright copper is formed more easily on the zinc-iron alloy than it is upon pure zinc. Care must be taken, therefore, in carrying on the Preece Test to avoid conditions which would subject the test specimen to a dilute copper sulphate solution, such as obtains when the specimen is frequently rinsed with water, and to avoid a burnishing or rubbing action when the specimen is cleaned.

We attempted to show the change in rate of solution as we passed from the zinc layer, to the zinc-alloy layer, by plotting the loss in weight of coating against the time of immersion. Owing to this irregular tendency of the copper to remain attached to the coating and to more or less protect the coating underneath from solution, the results were altogether misleading. Although this abnormal appearance of bright copper is much less liable to occur when one-minute immersions are employed, no dependence can be placed upon the end point as thus obtained.

CAUSTIC SODA METHOD FOR TESTING COATING.

We have already pointed out that hot concentrated caustic soda will dissolve the zinc coating when placed in contact with iron. Under some conditions an alloy very high in iron is left upon the iron base. Altogether the method is not a convenient or a rapid one, and therefore we desire to propose the following substitution for the Preece Test, devised by Mr. Patrick.

BASIC LEAD ACETATE METHOD.

When a zinc-coated iron article, free from the products of corrosion, is placed in a basic lead acetate solution at ordinary temperature, the coating passes into solution and an equivalent amount of metallic lead is precipitated in a loosely adherent form upon the specimen. This lead is easily removed and the coating determined by measuring the loss in weight of the test specimen, or the lead precipitated. The reaction is retarded by the precipitation of the lead and therefore when a heavily galvanized piece is being tested this lead must be periodically removed. Either a volumetric or a gravimetric method for determining the coating may be employed, according to whether a balance is available or not. If the volumetric method is used, this lead must be preserved, while if the specimen be weighed the lead may be discarded, unless on account of the size of the specimen it is found desirable to collect and weigh the lead. When the iron-zinc alloy is thus dissolved, the iron of the alloy passes into solution and existing as basic iron acetate colors the solution red as it oxidizes. If desirable this iron can be determined by weighing as oxide, or by titration with standard oxidizing solution. When the coating has all been removed and bright iron only is visible, either the lead is quantitatively determined, or the iron base is dried and reweighed.

SAMPLE.

The size of sample must of necessity depend upon the material under investigation. In the case of sheets

or plates a piece 2 by 2 inches, and for wire a piece 3 to 6 inches is desirable, according to the gauge. The size of the wire and sheet should not vary more than 1-64 inch plus or minus. The longer the test piece, the smaller is the percentage error of measuring. The specimen should be weighed if the gravimetric method described below is to be employed, to three decimal places if possible.

SOLUTION.

The solution should be made up as follows: 400 grams of crystallized lead acetate are dissolved in one liter of water and four grams of finely powdered litharge added. The mixture is shaken until most of the litharge is dissolved. Any insoluble residue may be allowed to settle to the bottom from which the clear solution is later decanted and diluted to a specific gravity of 1.275 at 15.5° C. This produces a solution of basic lead acetate which dissolves zinc and zinc-iron alloy but does not attack the iron base.

METHOD OF IMMERSING.

Enough of the lead acetate solution should be employed to completely cover the specimen, and, on the other hand, the specimen should be so placed in the containing vessel that the zinc is completely exposed to the solution. This is a matter of detail that will vary with the form of the specimen, but wire should be placed on end and sheets on an edge, in order that the zinc acetate formed may readily pass off into solution. Tall, narrow, glass cylinders are desirable for the wire, while beakers or thin glass water tumblers may be used for sheets or other forms.

TIME OF IMMERSION.

The test specimens should be allowed to remain in contact with the solution for three minutes. The adherent lead is then removed from the specimen with the hands, a rubber "policeman," or a stiff brush (into preferably another beaker or tumbler, if the lead is to be determined). A burnishing action must be avoided as under some circumstances closely adherent lead may be plated out on the zinc. The specimen is then replaced in the lead acetate solution and after another three-minute immersion the lead is again removed as before. This is repeated until the bright surface of iron is exposed. Four three-minute immersions are usually sufficient. The bright iron base differs so radically in appearance from both the spongy lead, or even lead in the form of a bright adherent film, that a *lack of uniformity* in the thickness of the coating is readily detected. This is an important point in testing wire. A thin place on the wire may frequently show the bright iron surface after the first immersion; we have found some specimens of galvanized wire which over considerable areas carried no coating at all. On such places no lead is precipitated, and the bare iron easily recognized.

DETERMINATION OF COATING.

(A) If a balance be employed the bright iron piece is now thoroughly washed with water, rinsed in alco-

bol, and dried and weighed. The difference between the first and second weighings gives the weight of coating on specimen employed. Suitable factors for converting this into weight per unit area are given later; but, for example, if exactly 6 inches of wire be used, the weight of coating in grams as above determined, multiplied by 23.31, gives the weight of coating in pounds per mile of wire. This value can be readily calculated into pounds of coating per ton of wire if the weight of the wire per foot be known. If the weight of the test specimen be very large with respect to the weight of the coating, small errors in weighing invalidate the accuracy of the test. In this case it is best to collect the lead either on a Gooch crucible, and wash with boiled water and weigh, or to gather it together into a lump and weigh after washing with boiled water by decantation.

(B) If no balance be available the weight of zinc per mile or per unit area can be determined volumetrically as follows: The accumulated lead from the specimen is washed by decantation several times with hot water previously boiled, until free from lead acetate. This is very readily accomplished. The metallic lead is dissolved in a small quantity of hot concentrated nitric acid, for example, two cubic centimeters. This solution is then neutralized with a slight excess of ammonia and without filtration re-dissolved in acetic acid. A standard solution of potassium chromate (K_2CrO_4) is prepared containing exactly 12.624 grams per liter. Potassium chromate reacts with lead acetate, giving yellow lead chromate. On the other hand, silver chromate is deep red. So long as there is any lead acetate in the solution, the potassium chromate will unite with this lead acetate, so that if a little of this solution be placed on a paper containing silver nitrate no reaction will be observed. The moment enough potassium chromate is added to entirely precipitate the lead, the first drop of excess of potassium chromate will color silver nitrate paper red and thus determine the end-point of the reaction. By measuring from a burette, therefore, the exact number of cubic centimeters necessary to give the first indication of red with silver nitrate paper and using the following factors, the weight of zinc per unit area can be directly computed.

FOR WIRE.

Length of test piece taken in inches.	Factor for converting cubic centimeters into lbs. coating per mile.
1	0.60
2	0.30
3	0.20
4	0.15

FOR SHEETS.

Size of test specimen.	Factor for converting cubic centimeters into ounces coating per square foot.
1 sq. inch	0.0213
4 sq. inch	0.0053

It is apparent that working charts may easily be prepared from which the data desired can be read off directly. The following formulæ express the relationships most frequently used:

Let X = pounds coating per mile of wire.

Let Y = ounces coating per square foot of sheet.

Let N = cubic centimeters potassium chromate solution used.

Let L = length of test piece in inches.

Let S = area of test piece in square inches.

Let G = loss in weight in grams.

$$X = \frac{139.86 \times G}{L}$$

$$Y = \frac{5.090 \times G}{S}$$

The manufacturers of both sheets and wire publish small handbooks which may be had for the asking, giving the gauge, relation of weight to area and length and other data for further calculations.

The following results obtained by this method on three samples of No. 9 gauge wire are fairly representative of the accuracy attainable:

LENGTH OF TEST SPECIMENS, THREE INCHES.

Sample No. 1.	Loss of weight.	Lbs. coating per mile of wire.	Lbs. coating per ton of wire.
Analysis 1.....	0.1030	4.81	31.07
Analysis 2.....	0.1010	4.75	30.68
Analysis 3.....	0.1025	4.79	30.94
Analysis 4.....	0.1035	4.86	31.39

Sample No. 2.	Loss in weight.	cc. K_2CrO_4 used.	Lbs. coating per mile of wire.	Lbs. coating per ton of wire.
Analysis 1....	0.1845		8.55	55.23
Analysis 2....		42.7	8.54	55.17
Analysis 3....		42.6	8.52	55.04
Analysis 4....		42.8	8.56	55.29
Sample No. 3.	Loss in weight.	cc. K_2CrO_4 used.	Lbs. coating per mile of wire.	Lbs. coating per ton of wire.
Analysis 1....	0.1050		4.93	31.85
Analysis 2....		24.8	4.96	32.04
Analysis 3....		24.6	4.92	31.78
Analysis 4....		24.7	4.94	31.91

DETERMINATION OF IRON IN ALLOY OF COATING.

The iron which is in combination with zinc in the coating as an iron-zinc alloy will pass into solution as iron acetate and can be determined as follows: The combined lead acetate employed for the sample, together with the wash waters from the metallic lead, is heated to boiling and the lead precipitated with a slight excess of sulphuric acid, the iron oxidized with nitric acid, and precipitated with ammonia, washed and weighed as iron oxide. Or if no balance be available, it may be washed, re-dissolved in sulphuric acid, reduced with zinc, and titrated with standard potassium permanganate solution. An idea of the amount of iron in the different kinds of zinc-coated iron may be obtained by the following results obtained from products found in the open market:

Kind of material.	Iron in coating. Per cent.
Ordinary hot dipped sheet.....	2.26
Sherardized plate	11.70
Wet galvanized (electro deposited) sheet.....	trace
Wet galvanized (electro deposited) sheet.....	7.46

NOTES.

(1) In order to insure the absence of free acetic acid, and to eliminate any danger of the iron base being attacked, enough lead oxide is added to render the resultant solution slightly basic.

(2) It is important to immerse the test specimen

for periods not shorter than three minutes at a time. Otherwise a coherent deposit of lead may be formed.

(3) In the case of articles that are weathered, or have been subjected to corrosion, it is necessary first to remove the layer of zinc salts from the surface by immersion in dilute hydrochloric acid. In such cases the gravimetric determination by difference must needs be used to obtain accurate results.

(4) In washing the metallic lead free from lead acetate, it is necessary in very accurate work to use water previously boiled to free it from oxygen and carbon dioxide.

(5) In order to obtain exact lengths of wire, it is well to cut the specimen a little longer than desired and file to exact measurement.

(6) The presence of a deep red color in the lead acetate after a test must not be taken as indicating a large amount of iron. Traces of iron as basic ferric acetate produce a strong color.

(7) If the iron is determined by titration with potassium permanganate, it is necessary to first remove it from the acetic acid, as with acetic acid potassium permanganate titration is not accurate.

(8) The size of the specimen which can be taken, if the volumetric method be used, is limited by the amount of potassium chromate solution required for reacting with the precipitated lead. Three inches of wire are all that are necessary, except that greater accuracy in measuring is obtained by using a longer piece.

(9) Even when employed in the same way in which the Preece Test is used, the lead acetate solution has important advantages over copper sulphate in this: that a bright copper surface on the zinc cannot be distinguished from the bright surface on the iron base. In the lead acetate solution, bright adherent lead may be precipitated, but cannot be mistaken for the bright iron. Hence, if it is desired to make a rough test for uniformity of coating on a wire, one-minute dips may be employed, and the uniformity with which the bright iron appears indicates the evenness of the coating.

The authorities of Amsterdam are considering the establishment of a factory to make briquets of street sweepings and rubbish. The product to be worked up amounts to 140,000 tons a year. The factory will cost \$200,000, and will require \$100,000 a year for operating expense. It is believed it will yield a profit of over \$20,000 a year.

The rubber shipments from Mexico during the five months ended November, 1910, aggregated in value \$4,970,000 gold, against \$2,148,000 in the same period of 1909, and only \$1,480,000 in the same period of 1908. This was exclusive of guayule rubber, which totaled \$2,550,000 in the five months last year, compared with \$1,700,000 and \$564,000 in the respective periods of 1909 and 1908.

PREVENTION OF UNNECESSARY WASTE OF NATURAL GAS FROM WELLS.

Governor Johnson of California has approved the act suggested by State Mineralogist Aubury, and passed by the California Legislature at its recent session, which provides a penalty for permitting the unnecessary waste of natural gas from wells. The date of approval was March 25, 1911. The act is therefore in effect. The text of the measure is as follows:

Section 1. All persons, firms, corporations and associations are hereby prohibited from wilfully permitting any natural gas wastefully to escape into the atmosphere.

Sec. 2. All persons, firms, corporations or associations digging, drilling, excavating, constructing or owning or controlling any well from which natural gas flows, shall upon the abandonment of such well cap or otherwise close the mouth of or entrance to the same in such a manner as to prevent the unnecessary or wasteful escape into the atmosphere of such natural gas. And no person, firm, corporation or association owning or controlling land in which such well or wells are situated, shall wilfully permit natural gas flowing from such well or wells wastefully or unnecessarily to escape into the atmosphere.

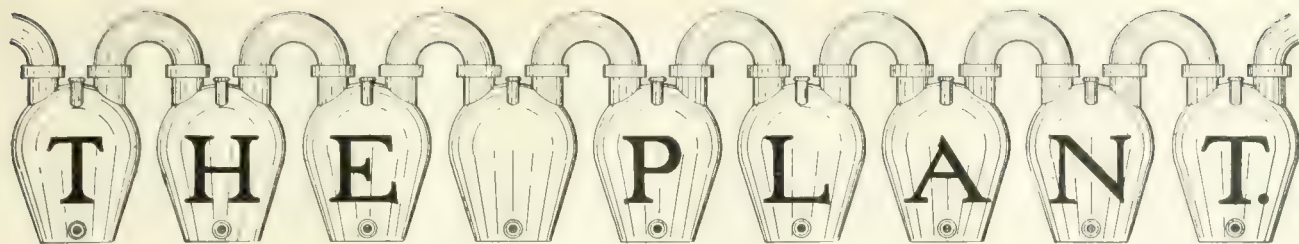
Sec. 3. Any person, firm, corporation or association who shall wilfully violate any of the provisions of this act shall be deemed guilty of a misdemeanor, and upon conviction thereof, shall be punished by a fine of not more than one thousand dollars or by imprisonment in the county jail for not more than one year, or by both such fine and imprisonment.

Sec. 4. For the purposes of this act each day during which natural gas shall be wilfully or unnecessarily allowed to escape into the atmosphere shall be deemed a separate and distinct violation of this act.

Sec. 5. All acts or parts of acts in conflict herewith repealed.

Sec. 6. This act shall take effect immediately.

The necessity for a measure of this kind arose from the fact that at different places in the state where wells have been drilled for oil and other purposes, only gas has been met with. Where it has been possible to utilize this gas for domestic or other purposes, it has been done, but there are many instances where the wells have been abandoned and countless millions of cubic feet of gas have been allowed to go to waste in the atmosphere and no attempts were made to cap the wells. Some of these wells have been flowing for years. Demonstrations that gasoline can be profitably extracted from natural gas have been made in Ohio, West Virginia and Pennsylvania, where a number of plants have been installed. It is reported that a large plant is soon to be established in Kern County on the Honolulu Gas Well, where it is expected to handle 4,000,000 cu. ft. of gas daily, which is expected to yield 8,000 gals. of gasoline per day.



APPLICATION OF HIGH PRESSURE GAS TO FURNACE USE.*

Many years' experience in the employment of coal gas for heating purposes have led me to the conclusion that the methods commonly employed—that is, taking the gas from the mains and consuming with some form of Bunsen burner—are not only wasteful of gas, but are quite incapable of giving anything like a constant temperature, where that is desired, or, of heating any given kind of oven, or other apparatus to the same temperature in the same time in consecutive operations. With the ordinary burner and gas of the same composition, the variation in time of heating up a given furnace to a certain temperature may vary in consecutive days as much as two hours. It is impossible for the best of workmen to judge of the temperature or heating power of a flame by its appearance, or to regulate the air supply to a gas supplied directly from the main in which the pressure is constantly changing. Many times I have observed the most erratic behavior of gas-fired furnaces using raw gas from a main—variations of from 1,200 to 1,600 cu. ft. per hour not being at all uncommon, with now and then an explosion, owing to irregular air supply. When such an accident occurred, several neighboring furnaces or ovens were generally put out of action.

To obviate these irregularities and fluctuations, I employ a high pressure gas capable of working up to 160 ins. of water. High pressure gas lighting has a limit of about 60 ins. water pressure. Air is drawn in at ordinary atmospheric pressure in both cases.

A rotary exhaustor and compressor draws the gas from the mains and supplies it at any desired constant pressure to the burners; any surplus gas is passed back again, and therefore does not interfere with the pressure as delivered from the burner nozzles, nor is it registered again in the meter. Variation of pressure in mains has no effect. Until gas is supplied from the works at high pressure, a small apparatus similar to this is necessary if high pressure gas is to be employed for heating.

Although some towns have high pressure gas mains, the gas does not appear to be used for heating, but merely for transmissional purposes, and is governed down to some ordinary pressures. In Birmingham, Dr. Davidson, the chief chemist to the corporation, and Mr. Barrett, the lighting superintendent, seems to be alive to the possibility of its use for heating

purposes, and a high pressure main is now being installed in that city. The gas meters for high pressures seem to present some difficulties, however.

Assuming the maximum consumption of gas in a small laboratory plant to be 500 cu. ft. per hour, and that 100 ins. pressure is sufficient to meet all conditions, the valve acts directly the machine is put to work, and if 300 cu. ft. of gas is used, the surplus 200 ft. is by-passed back into the inlet main. The pressure would therefore be a constant one, whether one foot or 500 were consumed. The gas pipe need be only one-half the usual size, or half the size of the compressor outlet. The inlet pipe to compressor should be of full size. Having fixed the compressor, any temperature can be obtained by simply turning on a gas tap. Assuming the highest temperature required be 2500° F. or thereabouts, 100 ins. should suffice, and that can be the standard pressure. Supposing there are four furnaces in which different temperatures are required. The first furnace is to be heated to say 2600° F. and the gas taken directly from the standard pipe. The second furnace requires 2000° F. only; 70 ins. pressure will suffice for this, and can be obtained by reducing tap or governor, care being taken in the first instance to have a gauge to show what the pressure actually is. The third furnace requires 1000° F.; for this 35 ins. pressure should suffice. Again, a pressure gauge is necessary, so that the tap at the burner can be fully open and not throttled. The fourth furnace may be, say, for melting lead only, for which 16 to 20 ins. pressure is sufficient. Under the above conditions, the same temperatures will be obtained in the same length of time daily, and after the various burners have been once adjusted there will be the same consumption of gas. Quantities of about half a pound of copper have been melted in 6½ minutes, and remelted in somewhat less time. If larger quantities have to be dealt with, a second or third burner is turned on, as I do not recommend interfering with the standard pressure, although it is quite possible that by raising the pressure, more gas and air would be used, and the second burner found unnecessary. But it is generally better to maintain a standard high enough for the particular purpose without constantly raising or lowering the pressure.

It might be argued, why not 100 ins. for 1,000° F. as well as 2,600° F.? It has been found that at very high pressures crucibles are liable to be melted or distorted, and for certain purposes it is better to work with gas under known conditions rather than increas-

*From the Journal Society of Chemical Industry

ing the temperature too rapidly and overheating some part of the article. With 100 ins. pressure it is better to work with gas under known conditions rather than increasing the temperature too rapidly and overheating some part of the article. With 100 ins. pressure it is possible to soften the ordinary plumbago crucibles in 30 minutes. There are of course cases where the pressure has necessarily to be raised and lowered. In the manufacture of glass, for instance, 10 cwt. or 20 cwt. crucibles should be treated at 60 ins. pressures

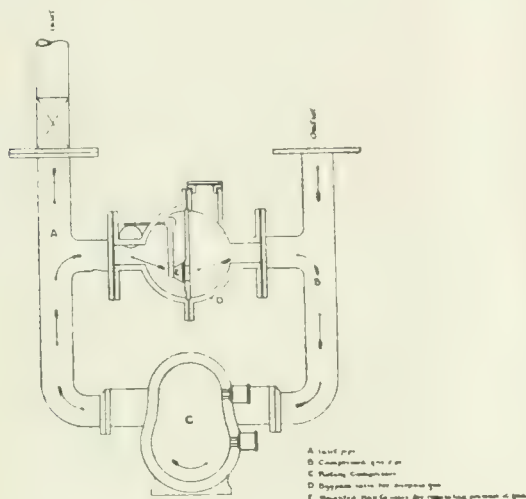


Fig. 1.

for 12 hours to melt the composition; 90 ins. for 6 hours are required to refine the glass, followed by a drop down to 40 ins. for emptying. This process has been found to work by the clock perfectly, and only the cost of town's gas has prevented its general adoption commercially. If gas at 30 cts. per 1,000 cu. ft. could be obtained, glass could be made more cheaply by its aid than with coke at 15 cts. per ton.

In the melting of lead in cast iron pots, 20 ins. pressure is quite sufficient, but if it is obligatory to use 100 ins. pressure, the heating must be by radiation and not by playing directly on the iron vessel.

Another application of high pressure gas heating is in the tempering of long steel bars, such as axle trees. These are suspended vertically, four or more together, and heated at 60 ins. pressure for 30 minutes.

At the expiration of that time, the burners are shut off for five minutes to allow the temperature to equalize. The axle trees or bars are then removed for tempering, and a new batch inserted. All burners are relighted by opening one stop-cock and the process continued, so that it is quite possible to temper 120 axle trees per day in a furnace 2 ft. in diameter and 7 ft. high.

Tempering, case hardening and annealing furnaces as described by model need not have large combustion chambers. In fact by this method there is no need for large combustion chambers as the gas is directed more closely to the object, and only sufficient air for complete combustion is used. These furnaces have a division down the center so that the gas is confined to each side, and so closely is the heat exerted, that fur-

naces have been equally heated all over with less than 10° F. difference in any part.

It is quite possible by these methods of restricting the supply of air and gas to the object to be heated, to reduce the consumption of gas from 600 cu. ft. to 200 ft. as I have done already. Warming plates, instead of consuming 180 cu. ft. per hour, now consume 25 ft. Metal melting pots requiring 1½ hours to melt with gas at 80 ft. per hour now require gas at the rate of 45 ft. per hour, and melt in 25 minutes, after which the gas is shut down to the rate of 25 ft. per hour. These are a few cases out of a large list, and will indicate the reason for drastic changes in our methods of heating.

It is easy to take a high pressure flexible pipe any distance, and heat small or large furnaces built to suit any article from 1 in. diameter to 10 ft., so that any part of a large object may be case-hardened.

One more case, that of warming buildings, may be mentioned. It is well known that a large proportion of the heat generated in a gas fire or stove passes up the chimney with the products of combustion without useful effect. If the highly heated products of combustion are carried through a metallic flue pipe passing through the room for a sufficient distance, nearly

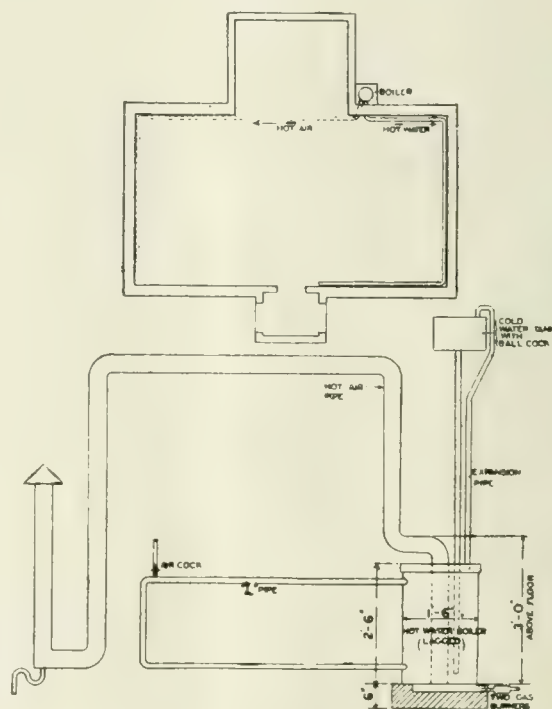


Fig. 2.

all the heat generated may be rendered available for warming the room. With an ordinary gas fire this can only be done to a limited extent because the flue will not draw owing to the cooled gases having little or no ascending power, and there is also the objection that an unlined metallic flue will at its base become dangerously hot. It is, therefore, desirable to give the flue gases momentum, so that one part is not unduly heated above another.

In order to accomplish this Colonel Bagnold, C. B.,

R. E., Superintendent Building Works, Royal Arsenal, conceived the idea that, granted sufficient initial pressure, either in the gas supply or in the air supply, the injector-like action of a high pressure burner would effect this. Under Colonel Bagnold's direction, experiments have been carried out which completely

end of the flue pipe only slightly warmer than the air in the room. Following the example of modern steam boiler installations, this system is really one of "forced draught," the draught being caused by the velocity of the gas from the nipples. This installation has been in use for six months, and has given no trouble whatever. It possibly has an efficiency of about 90 per cent.

In conclusion, I may say that the objects I had in view throughout the experiment described in this paper were to simplify our present system of gas heating, and, by obtaining a higher efficiency and effective flame control, to ensure definite results.

This has been so far achieved that the temperature in a furnace or crucible can now be controlled to such a nicety that it is merely necessary to open or close a

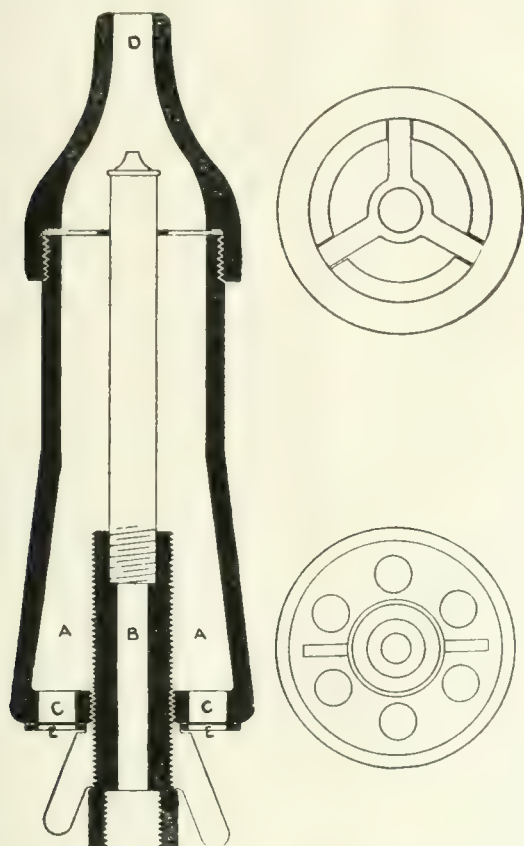


Fig. 3—High Pressure Gas Burner.

demonstrated the soundness of this theory; by means of two high pressure burners fed with gas at 60 ins. pressure, a room 750 ft. super-floor space has been heated 31° F. above the outside temperature, the consumption of gas being 80 ft. per hour. The installation is simple, and may be readily understood by reference to the diagram. The gas supply is taken from an adjoining building, and is on the same service as the lighting. Outside the building is a small combustion chamber lined with fire bricks into which play burners. From the combustion chamber a steel flue is led through the wall and around the room, and again through the wall to the outer air. On leaving the heating chamber (which attains a temperature of $2,000^{\circ}$ F.) the flue passes through a boiler from which are led water circulating pipes; thus part of the room is heated by hot water and part by a hot flue pipe. The flue pipe on leaving its water jacket only retains a temperature of about 250° F., as the water jacket is made long enough to prevent the flame passing into the pipe outside the jacket, and therefore is not considered dangerous. The gases pass out at the

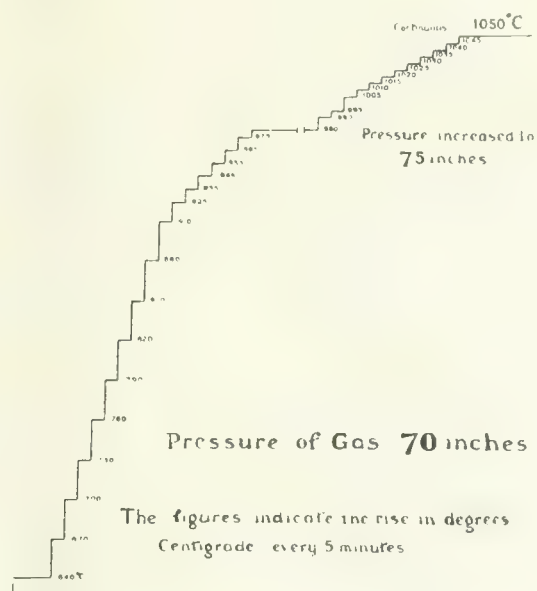


Fig. 4—Temperature in a High Pressure Gas Furnace, Onslow's Patent.

gas tap and wait with certainty for the result. By this system explosions due to imperfect mixtures of air and gas are avoided, a more direct contact of the temperature is ensured, and a small flame is used, its heat radiating from the furnace walls, in place of the large, limp, and uncertain flame that has to a great extent been the practice hitherto. These improvements, in addition to giving better heating results, naturally tend towards economy and a lower cost of production, an end which is furthered by the use of smaller combustion chambers and the simplification of installations.

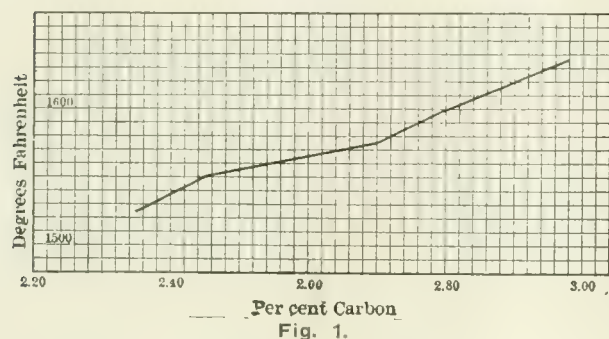
In 1909 the British imports of scientific apparatus amounted to \$3,965,000, of which \$1,018,870 came from France, and \$1,056,220 from the United States. In 1910 the imports amounted to \$7,595,000, of which France furnished \$1,538,986 and the United States \$3,804,400 worth.

ANNEALING MALLEABLE CASTINGS.*

By W. P. PUTNAM.†

To emphasize the importance of exercising greater care in manufacturing malleable castings and to point out some sources of the errors that are so frequent in foundries leaving the annealing process to the judgment of some individual with a "trained eye," the work of this paper was undertaken. There is no branch of the foundry business that needs closer supervision than the process of annealing iron; no greater source of economy; no one department where greater skill is required. And yet it is safe to say that very few foundries recognize the importance of equipping their annealing ovens with proper appliances to insure a uniform product.

To call attention to the variations that can be produced in the annealing process, a set of test bars



was made and sent out to 11 malleable iron foundries to be annealed in accordance with their usual practice. The bars were all made from the same metal and an attempt was made to obtain a bar free from shrink or draw in the center by casting the bars with large ends. In all of the bars that were broken but very little draw was found. The analysis of the metal poured into the bars was as follows:

	Per cent.		Per cent.
Silicon	0.74	Manganese	0.24
Sulphur	0.041	Combined carbon.....	2.70
Phosphorus	0.148		

The results obtained on the annealed bars are given in Table I. It will be noticed that in each case under too high temperature or a long period of annealing, the carbon was nearly all removed. Either practice is wasteful and produces poor malleables for some purposes.

The annealing process should be conducted to produce definite results. Metal intended for parts where machining is necessary should be given a different treatment in the annealing process from that of metal that does not need to be machined.

To arrive at the proper annealing temperature the critical and proper annealing temperature was ascertained on a number of hard iron specimens ranging from 2.35 to 3 per cent total carbon. These tempera-

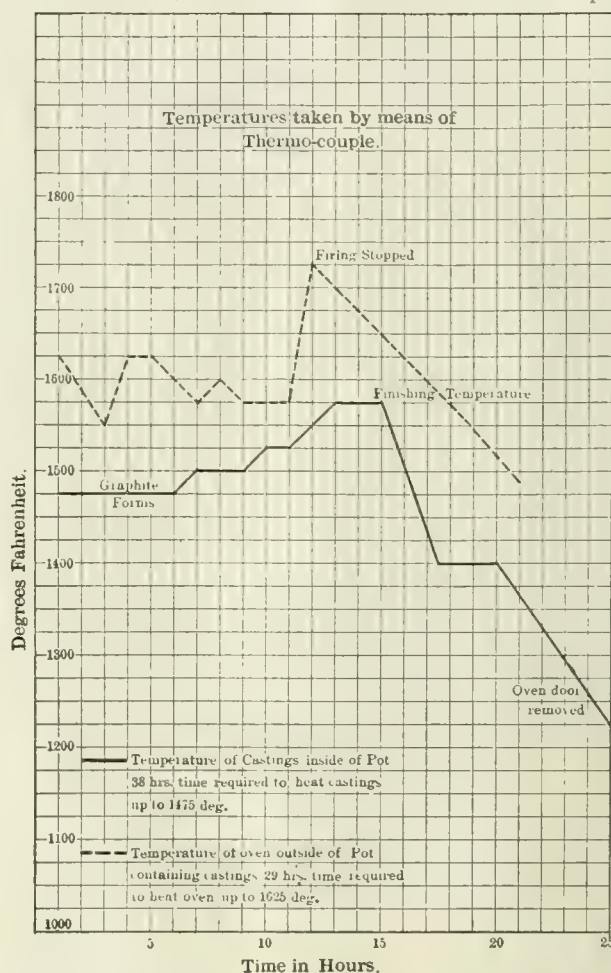


Fig. 2—Curve of a Typical Annealing Cycle.

tures are diagrammatically shown in Fig. 1. They are accurate within 10° F.

In Fig. 2 is shown the curve of a typical annealing cycle in an oven containing 14 tons of castings. The carbon in this lot of iron was 2.65 per cent. After reaching the annealing temperature there is no further rise in the temperature of the metal until all of the

TABLE I—RESULTS OF ANNEALING BARS OF THE

Foundry No.	Total carbon.	G. C.	C. C.	El L. Lb. per sq. in.	T. S. Lb. per sq. in.
1	1.49	1.49	Trace	20,823	40,996
2	1.94	1.94	Trace	34,134	44,034
3	1.47	1.47	Trace	29,276	39,638
4	0.86	0.77	0.09	31,128	48,489
5	1.74	1.74	Trace	30,960	45,760
6	1.71	1.71	Trace	28,326	40,408
7	1.32	1.18	0.14	32,224	49,107
8	0.80	0.80	Trace	33,333	47,336
9	1.58	1.58	Trace	30,833	39,862
10	0.40	0.40	Trace	41,792	50,849
11	1.32	1.32	Trace	35,042	45,213
Hard iron.....			2.70		50,846

*Paper read before the Detroit Foundrymen's Association.

SAME COMPOSITION AT DIFFERENT FOUNDRIES.

Elongation in 2 in.	Reduction in area.	Kind of packing.	Time of annealing. Hours.	Annealing temperature. Deg. F.
8.59	12.13	Slag	112	1,450
10.15	13.87	Mill scale	144	1,700
8.59	17.63	Mill scale	101	1,620
8.59	19.18	Not given	Not given	Not given
8.59	14.88	Not given	Not given	Not given
8.59	15.00	None	72	1,650
8.59	13.23	Not given	Not given	Not given
7.60	17.27	Slag	208	1,620
7.03	19.51	Mill scale	168	Not given
7.45	20.92	Mill scale	96	2,200
7.80	12.56	Slag	144	1,700

†President Detroit Testing Laboratory.

carbon has changed into the graphitic condition. As soon as the change has been completed the temperature will rise in accordance with the temperature of the oven. It has been found that to heat the castings much above the proper annealing temperature for a long period is not only unnecessary but a waste of fuel, and produces poor castings for some purposes. After the carbon has all been changed the oven need not be fired for more than six hours. When the metal has gradually cooled to 1200° F., the cooling can be hastened as rapidly as desired without harming the quality of the castings.

Some of the reasons why hard and brittle castings are occasionally found may be given as follows:

1. Under annealing (which may be caused by too low temperature or by holding the castings at the proper temperature for too short a period).
2. Too rapid cooling of the castings after being annealed.
3. Heating annealed castings above the critical temperature to straighten without subsequent annealing.

That there is great need of the systematic handling of the annealing process is clearly evident from the results tabulated in Table I. The variations shown there are typical of castings in general.

The following statistics show the imports of ferrosilicon into the United States from the time the Payne-Aldrich tariff went into effect up to September 30, 1910. Under the Dingley tariff, ferrosilicon bore a duty of \$4 a ton, the same as that of pig iron. The new tariff made the duty on ferrosilicon containing not more than 15 per cent of silicon \$5 a ton, and on ferrosilicon containing more than 15 per cent of silicon 20 per cent ad valorem. For a portion of the third quarter of 1909, or from July 1 to August 5, the old duty was in effect. In that time the importations of ferrosilicon of all classes were 3,717 tons, advance purchases being made in view of the impending increase in duty. The figures given below for that quarter are for the portion between August 5 and September 30:

IMPORTS OF FERROSILICON TO THE UNITED STATES.—GROSS TONS.

Quarter ending—	15 per cent silicon or less.	More than 15 per cent silicon.
September 30, 1909.....	616	880
December 31, 1909.....	2,050	1,373
March 31, 1910.....	400	1,752
June 30, 1910.....	750	2,273
September 30, 1910.....	425	2,120
Totals	4,241	8,398

No doubt the importations, as shown by the above figures, particularly of ferrosilicon containing more than 15 per cent silicon, are less than is commonly supposed in the trade. From the Canadian plant producing this metal the importations for a year or more have been at the rate of 4,500 tons of 50 per cent ferrosilicon a year.

THE CONDENSATION OF ZINC VAPOR FROM ELECTRIC FURNACES.*

By F. T. SNYDER.

The development of every technical industry is characterized by change from a qualitative basis, in which progress depends on experiment and the trade skill of individuals, to a quantitative basis in which progress is made by calculations from accurately obtained data. On the first basis, progress is slow and costly. The number of independent variables in each technical problem is large, and the choice of improper proportions for some of them often disguises the true effect of a change in the particular one under observation. On the quantitative basis, which may be called the engineering basis, progress is rapid and financially efficient, as the proper proportions of each group of variables may be mathematically equated between the groups, and the waste of time and capital in false construction avoided.

The condensation of zinc in the standard retort smelters is an illustration of the first condition—the basis of trade skill. The limiting technical factor in retort smelting is not the condenser, but the material of the retorts. This fact has shielded the design of the condenser from the commercial pressure for progress, the pressure for improvement in retort smelting being now on the retort. In its white-hot operating condition, the comparatively small mechanical strength of the retort material limits the load of ore a retort can carry to a few pounds. The melting point of this material is only a few degrees above the temperature required for smelting zinc ore. This puts a low limit to the rate at which heat can be forced through the walls of the retort. As only a small amount of heat arrives inside the retort per hour, only a small amount of ore is smelted and only a small amount of zinc vapor released to be condensed by the condenser. Through many years of experience, retort condensers have been proportioned, on the basis of experiment and trade skill, to handle this slowly arriving volume of zinc gas with high efficiency. A fairly-well operated retort condenser will condense from the gas delivered to it something more than 99 per cent of the zinc which is condensible under the temperature conditions at which the condenser is run. If a gradual increase could be made in the mechanical strength of retort material, the size of the retorts would be gradually enlarged, and it is probable that by a process of "cut and try," the proportions of condensers would keep pace with the demand for increased capacity up to, possibly, double their present capacity. The discovery of a retort material of very much better heat conductivity than the present clays would not materially increase the rate at which zinc gas could be delivered to a condenser, as the limiting factor in this direction is the low heat conductivity of the ore

*Paper presented at the 19th general meeting of the American Electrochemical Society, in New York City, April 6 to 8, 1911.

itself. To make heat flow faster into zinc ore than it does in retort smelting means that the outside of the ore must be heated hotter. If we heat it much hotter than in standard retort practice, the ore will begin to melt. If a retort material should be discovered that combined high heat conductivity with high mechanical strength when white hot, we could change the shape of the retort, and by spreading the ore out in a thin layer reduce the average distance the heat had to travel through the ore and at the same time increase the relative area of the heating surface. This would enable us to smelt more ore in a retort in a given time with the present temperature range and so deliver zinc vapor faster to the retort condenser. If this improvement in retort material should be very great, so that many times the present flow of zinc vapor could be delivered to a single condenser, the pressure of progress would be diverted from the design of the retort to the design of the condenser. If, in addition, the discovery of this hypothetical retort material of high capacity should come about suddenly, it is apparent that the trade-skill basis of progress in design would break down and there would be the necessity for engineering in the design of zinc condensers. This has been shown in the several cases where zinc retort furnaces have been built in which fifty to seventy-five retorts of standard size have been arranged to deliver their zinc vapor to a single common condenser. On checking the dimensions of some of these condensers on an engineering basis, it immediately develops that the proportions, as determined from trade-skill basis, are such as to make their commercial operation hopeless.

The electric smelting of zinc ore has produced a technical condition very much like that which would result from the discovery of such an improved retort material. With electric smelting of zinc ore the retort is eliminated, and with it disappear those limitations of size and temperatures which are inherent in the retort. The ore can be melted; can be heated, subject to certain commercial limitations, up to a temperature that will drive the necessary heat into the ore at a very rapid rate. It is now practical and convenient with electric smelting to deliver zinc vapor to a condenser at a rate three hundred to four hundred times the rate at which such gas reaches the standard retort condenser. It is at once evident that the trade-skill basis of design is inadequate and that the opportunity exists for the application of engineering methods to the design of zinc condensers. The metallurgical advantages of electric zinc smelting, outside of the question of condensation, are such as to bring a large and growing commercial pressure toward activity in this direction. The desirability of directing this activity in a fruitful direction will justify us in considering some of the engineering that may be applied to the design of zinc condensers.

GENERAL STATEMENT OF THE PROBLEM.

There are two standard engineering methods that

may be applied to any problem. One is to work out with as great scientific accuracy as we may the detailed steps of a process and then follow it through mathematically. The other is to adopt a set of data which we know is operating satisfactorily under its conditions and, with the use, either consciously or unknowingly, of broad generalizations such as the conservation of energy or the persistence of motions, to transform this data to the conditions of the problem to be solved. We may with profit consider this condensation of zinc by both methods, taking up the latter method at this time and reserving the former method for a subsequent occasion.

The dimensions of the standard retort condenser and the conditions surrounding its use supply us with a set of reliable data for our immediate purpose. As the dimensions of condensers and the practice of condensation vary somewhat from plant to plant, it will be clearer if we adopt a typical average condenser

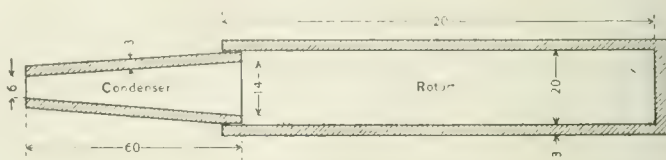


Fig. 1.

and average practice as to temperatures and rate of smelting. Figure 1 gives the dimensions of such typical condenser and its retort. The dimensions are given in centimeters. Cross sections are circular. The top of the condenser is hotter than the bottom. As this difference is small compared with the average temperature, we shall be within an allowable variation if we use the average of the top and bottom temperatures. The temperature of the gas delivered to the condenser varies considerably from hour to hour, but we shall be on the safe side in adopting the average temperature, as the condenser actually handles the maximum heat flow. The temperature of the condenser also varies with the row the retort is on, but the average for a furnace will be on the safe side for capacity data. While the rate of flow of the zinc gas also varies from hour to hour, the same factor of safety for the average exists, as the condensers successfully handle the maximum rate of flow.

In this connection it may be pointed out that it has been commonly thought, on account of the difficulties met in trying to design large condensers on the trade-skill basis, that for some unknown reason it was more difficult to design a continuously operating condenser than one to operate intermittently. A little consideration will make it evident that it should be easier to design a condenser for a relatively stable set of operating conditions, than for varying conditions. As the condenser, to be commercial, must in any event handle the maximum flow of gas and the maximum input of heat, it follows that the condenser can be smaller for a regular flow than for any vary-

ing flow delivering the same weight of zinc per day. One trouble with all the large condensers designed on the trade-skill basis has been that they have had only a small fraction of the condensing capacity they should have had. The electric smelting of zinc delivers a flow of gas that is quite regular in volume and temperature throughout twenty-four hours. If the flow had been irregular, for the same daily capacity, these undersized condensers would have been less able to handle the periods of maximum flow than they were the regular flow.

If, further, in arriving at our factors for engineering design, we adopt the condition that all the gases which are not involved in the reduction of zinc oxide (the water vapor, the carbon dioxides, sulphur dioxide and hydrocarbons from the reducing material), are driven off during the heating period, before the condensation of zinc begins, we shall be on the safe side with regard to area of cooling surface and also with regard to the volume available for diffusion.

DETAILED CONSIDERATION OF THE PROBLEM.

The average temperature at which the mixture of zinc and carbon monoxide gases leaves the retort and enters the condenser will be taken as $1,050^{\circ}$, and the average temperature at which the uncondensed gases leave the nose of the condenser as 600° . These are the temperatures along the axis of the condenser. In order to condense zinc and deliver it as liquid, the temperature of the inner surface of the condenser at the end next to the retort must be below the liquefying temperature of zinc vapor under the pressure conditions existing, and the inner surface of the small or nose-end of the condenser must be above the melting temperature. The melting temperature is 420° , and we can take the inside surface temperature of the condenser at the nose end at 450° . The boiling temperature of zinc at standard atmospheric pressure is 930° . To get at its liquefying temperature at the retort end of the condenser we shall have to consider the pressure conditions, as the boiling temperature varies with the pressure. Figure 2 gives the curves of the variation of the boiling temperature with changes of pressure as determined by Barus (U. S. Geological Survey, Bulletin 102, 1893). These curves have the advantage of having been determined experimentally directly from zinc vapor and so do not depend on the constancy of Boyle's law, as do the curves that are obtained by transposition under Groshen's law (Pogg. Ann. LXXVIII, p. 112, 1849) from the vapor pressure curves of mercury. The mixture of gases, as delivered by the retort to the condenser, is substantially under atmospheric pressure. The flow of gas through the condenser is slow, so that the drop in pressure required to produce this flow is too small to be considered in connection with our immediate purpose.

Before we can determine what proportion of the atmospheric pressure is carried by the zinc, it will be necessary to determine the proportion of zinc in

the mixture. For this purpose we can take the average charge of roasted ore to a retort as 20 kilograms carrying 60 per cent or 12 kg. of zinc. With residues amounting to 50 per cent of the weight of the ore in the charge, and analyzing 7 per cent, the unreduced zinc left in the charge amounts to 0.7 kg. The vaporized zinc amounts to 11.3 kg. The corresponding amount of carbon monoxide produced by the reduction is 4.9 kg. Of the zinc reduced, 0.1 kg. on the average is lost through the retort by leakage and breakages, and 0.4 kg. is absorbed by the material of the retort, leaving 10.8 kg. which is delivered to the condenser. As the relative volumes of each gas are inversely proportionate to their molecular weights, the zinc will carry 48.8 per cent of the atmospheric pressure or 372 mm. At this pressure, by the Barus curve, zinc boils at 863° . We will take the average temperature of the retort end of the inner surface of the condenser as 850° , corresponding to the average temperature of 450° for the inner surface at the nose end. It is to be noted that these are the average

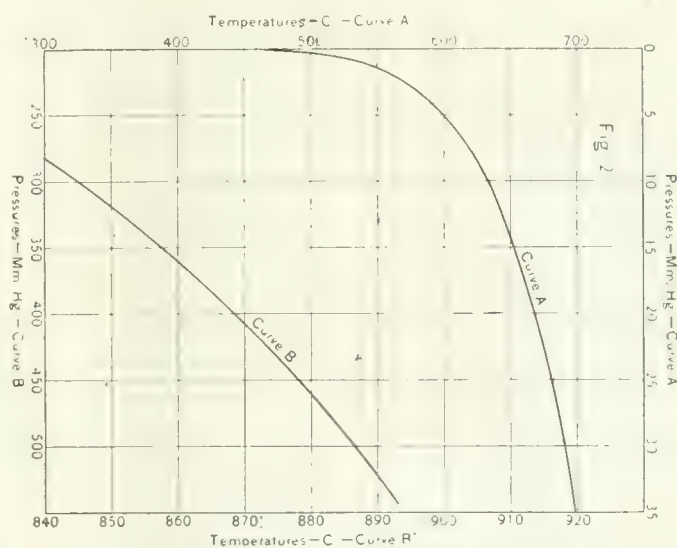


Fig. 2—Vapor Pressure of Zinc (Barus).

temperatures of the surface of the condenser itself, and not the temperatures of the gas near the surface. The average temperature of the whole inner surface of the condenser will be the sum of the products of the circumference and temperature at each end, divided by the sum of the circumferences, as the condenser is conical. In calculating this average temperature we use the inner diameter, 6 cm. at the nose end, as shown in Fig. 1. For the diameter at the retort end of the condenser a correction is needed. The condenser is inserted into the retort 5 cm., so that the effective cooling length is 55 cm. with a total length of 60 cm. The corresponding diameter at 55 cm. from the nose end is 13.2 cm. Calculated with this diameter, the average temperature of the whole inner surface of the condenser is 725° .

As the liquid zinc is in the bottom of the condenser, which is cooler than the average of the top and bottom, the average temperature of the liquid zinc may be taken as 650° . At the temperature at which the

uncondensed gases escape from the nose of the condenser, 600° , the vapor pressure of zinc is 5.0 mm. by the Barus curve. At this pressure and temperature 0.7 per cent of the zinc which entered the condenser leaves the nose in the form of vapor. In addition to this vapor, zinc also leaves the nose of the condenser in the form of fume, but this fume has already given up its heat of vaporization to the condenser. There was delivered to the condenser 10.8 kg. of zinc. Of this about 0.1 kg. leaves uncondensed and the remaining 10.7 kg. is condensed by the inner surface of the condenser. With an effective length of 55 cm., a diameter at the nose of 6 cm. and an effective diameter of 13.2 cm. at the retort end, the condenser has an effective inner surface of 1,660 sq. cm. The space taken up on the sides by the plug in the nose is about made up by the area of the plug. We may take the distillation as active for 16 hours. There is, therefore, condensed by the condenser, 10.7 kg. of zinc in 16 hours, with an active cooling surface of 1,660 sq. cm., or an average rate of 0.40 grams of zinc per sq. cm. per hour. This is one of the factors we need for the engineering design of condensers. It is to be noted that it applies only to clay, with about the kind of surface used in retort condensers. The real cooling surface takes in the area of all the projections and indentations. To use this factor for other materials it will be necessary to introduce a co-efficient of roughness and determine it experimentally for each new material.

The 10.7 kg. of zinc which is condensed enters the condenser at $1,050^{\circ}$ and is cooled down to liquid zinc at 650° . In doing this it gives up 460 Calories per kg., or 4,930 Calories for the 10.7 kg. The carbon monoxide enters the condenser at $1,050^{\circ}$ and leaves it at 600° . In doing this it gives up 124 Calories per kg., or 604 Calories for the 4.87 kg. which accompanies the zinc. Together the gases give up 5,534 Calories in 16 hours, or 346 Calories per hour. This is absorbed by the 1,660 sq. cm. of the condenser surface at the rate of 0.21 Calories per sq. cm. per hour. This is the second factor we need for the engineering design of condensers.

As the mixture of zinc and carbon monoxide gas moves from the retort into the condenser, the outer layer next to the surface of the condenser begins to lose its zinc by condensation, and the zinc in the mixture away from the surface begins to move toward the surface by diffusion. As the mixture moves through the condenser, all the zinc which is condensed moves sideways an average distance of $\frac{1}{3}$ the radius of the condenser. As the diffusion goes on at a determined rate, the speed of the mixture through the condenser must be such that diffusion will have time to move the zinc up to the cooling surface, while the mixture moves through the condenser.

At the temperature of $1,050^{\circ}$, at which the mixture enters the condenser, both the 10.8 kg. of zinc and the 4.87 kg. of carbon monoxide gas occupy a

volume of 36.8 cu. meters. They both occupy the same volume, each being diffused in the other. This volume of 36.8 cu. m. passes into the condenser in 16 hours, which is at the rate of 2.29 cu. m. per hour, or 637 cu. cm. per second. The diameter of the condenser at the effective entrance at the retort end is 13.2 cm. and the cross sectional area 136.8 sq. cm., so that the average velocity of the gases entering the condenser is 4.67 cm. per second. At the temperature of 600° , at which the gases leave the nose of the condenser, the 4.87 kg. of carbon monoxide and the 0.075 kg. of zinc that leaves as vapor, each occupy a volume of 12.4 cu. meters. As this goes through in 16 hours, the rate is 0.78 cu. m. per hour, or 217 cu. cm per second. The diameter at the nose end is 6 cm. and the cross sectional area 28.3 sq. cm., so that the average velocity of the gases at the nose end of the condenser is 7.67 cm. per second. The mean velocity through the condenser, which is obtained by dividing the mean of the volumes passing each end by the cross section area at the center, is 5.17 cm. per second. It is interesting to note that this mean velocity corresponds to an average time of 10.6 seconds for the passage of each part of the gas through the condenser.

At the retort end of the condenser, where the gases enter with an average velocity of 4.67 cm. per second, the distance through which the zinc has to diffuse on the average ($\frac{1}{3}$ the radius) is 2.2 cm., and at the nose end it is 1.0 cm., where the gases leave with an average velocity of 7.67 cm. per second. As the total pressure on the mixture of gases is constant, the speed of diffusion varies with the square root of the absolute temperature, and it will be convenient to introduce this as a co-efficient affecting the mean distance of diffusion, adopting as unity the speed of diffusion at 864° , the condensing temperature of the normal mixture of equal volumes of zinc and carbon monoxide. At the retort end of the condenser we have taken the temperature at the center as $1,050^{\circ}$ and at the surface of the condenser as 850° . The mean of these is 950° , and the temperature diffusion co-efficient, calculated from the square root of the absolute temperatures and referred to the rate of diffusion at 864° , is 0.96. At the nose end of the condenser the temperature at the center we have taken as 600° and at the cooling surface as 450° . The mean of these is 525° and the corresponding temperature diffusion co-efficient is 1.19. If we multiply the mean distance of diffusion at each end by its co-efficient, we will get the equivalent distance through which the zinc would diffuse in the same time if the temperature were the standard, 864° . These are 2.11 cm. for the retort end and 1.19 cm. for the nose end.

The satisfactory operation of the retort condenser shows that with the velocity at which the gases travel through the condenser, the rate of diffusion is sufficiently rapid to move the zinc through the average diffusion distance in the time available. The relative

velocity of diffusion sideways is proportional to the equivalent diffusion distance, and the ratio of this sideways diffusion velocity to the longitudinal velocity through the condenser is the third or diffusion factor that we need in the engineering design of large condensers. This ratio equals the product of the equivalent diffusion distance and the longitudinal velocity. For the retort end of the condenser, where the equivalent diffusion distance is 2.11 cm. and the average longitudinal velocity 4.67 cm. per second, the factor is 9.9, and for the nose end, where the equivalent half-radius is 1.19 cm. and the average longitudinal velocity is 7.67 cm. per second, the factor is 9.1. The mean factor for the whole condenser is the arithmetic mean of the factors at the two ends, and is 9.5.

We have now the reliable data from the standard retort condenser in convenient form for use in the engineering design of large zinc condensers. Collecting the factors in one expression, the result is that if we provide 1 sq. cm. of clay cooling surface (or an equivalent amount of some other material) for each 0.40 gr. of zinc to be condensed per hour, and if we arrange the thickness of the material of this surface and the temperature conditions at the other or non-condensing side of the material so that 0.21 kg. Calories of heat will flow through each sq. cm. of the surface per hour when the average temperature of the condensing surface is 725° , and if we arrange the volume inside the condenser, so that the product of the average longitudinal velocity of the gases in cm. per second and the average diffusion distance (corrected to the standard temperature of 864°) in cm., through which the zinc has to diffuse, is 9.5, then, the design of the large condenser will be on the same basis as the proportions of the standard retort condenser. It is to be kept in mind that only the cooling surface in the condenser that is between 850° and 450° counts in the production of liquid zinc. As this upper limit of 850° depends on the atmospheric pressure, the reduction of the atmospheric pressure with high altitude is to be taken into account for a condenser so located. It is interesting to note in the standard retort condenser, the proportions of which have been determined by many years of "cut and try," that these proportions now in general use give a diffusion factor that is substantially constant throughout the length of the condenser. This means that these proportions represent the maximum commercial economy in the use of the condenser material.

So far in considering the matter we have taken the condensation as all occurring at the surface of the condenser. In practice a certain amount of zinc as fume escapes from the nose of the condenser along with the zinc that is in the form of uncondensed vapor. As the temperature, 600° , is well above the melting point of zinc, this fume is in the form of tiny drops of liquid zinc, which has condensed in the gas away from the surface of the condenser. Drops as small as these fall slowly. Clouds in the sky are

made up of very small drops, and they fall so slowly as to appear stationary to casual observation. But these small drops fall with regularity at a constant velocity which can be calculated. The important point for us to notice is that the velocity of fall is constant and not rapidly increasing with time, as it does when a large, heavy body falls through the air. If we consider a drop that starts to fall from close to the top of the condenser, it will move forward and downward on a diagonal line, the resultant of its constant velocity of fall and the velocity of the longitudinal movement of the gases through the condenser. To be saved, a drop which starts from a point near the top must do so at a distance sufficiently far back from the nose of the condenser to have time to fall to the bottom before the longitudinally moving gases reach the nose end of the condenser. All the drops that start from the top at points nearer the nose than this point will not have time to complete their fall and will be swept out of the nose with the uncondensed gases. There is, therefore, a wedge-shaped volume, lying with its base against the outlet end of the condenser, and tapering back along the top of the condenser, from which volume all the drops will be lost, by going out as fume with the uncondensed gases. The cubical contents of this wedge in different condensers will be proportional to the product of the longitudinal velocity and the vertical distance of fall. The vertical distances bear the same ratio to each other in different condensers as do the $\frac{1}{3}$ radii (or diffusion distances), so that between different condensers the volume of this wedge is proportional to the product of the longitudinal velocity of the gas travel and the diffusion distance. But this is the same factor that we have previously determined from consideration of the conditions of diffusion, so that if we properly design our large condenser to have the same proportions for diffusion that the standard retort condenser has, we will be designing it also to have the same proportions as to the amount of fume lost in the uncondensed gas.

An examination of the drawings of many zinc condensers that have been built or proposed for large capacity shows some means of applying heat to the condenser either internally or externally, by fuel fires, gas flames, or in most modern cases by electricity. As its name implies, a condenser is an apparatus for withdrawing heat, and it is evidently a contradiction of effort to also apply heat in any way. As the designers of these condensers understood this fact more or less, the probable purpose of this application of heat was to get the condenser up to the proper working temperature at the beginning of operations, or with a view of regulating it in operation to keep the temperature of the condensing surface within the right range. Consideration of the point shows that for both reasons it is unnecessary. If we design our condensing surface of such thickness of material and arrange the heat conditions at the non-condensing side so as to have the condensing side at 725° when the

proper flow of heat per sq. cm. is passing through, then the surface will automatically keep itself at 725° . For if in any way we could get it hotter than 725° , this would increase the heat gradient to the other side and increase the rate of flow through the surface above the rate at which heat was being received from the condensing zinc and as the surface would be losing heat through its material faster than it was receiving it, the surface would cool down to the 725° for which it was designed. The reverse action would take place if we could in any way cool the surface below the temperature for which it was designed. The change in the heat gradient would soon bring it up to the designed temperature. In starting up a new and cold condenser, it will only be a short time before the heat from the zinc brings the temperature of the condensing surface up to the temperature for which it is designed. There is no practical need for extra means of heating.

It will be interesting in the light of the data that we have secured to examine the design of one of the large condensers which have been built. Such a condenser is described in connection with the Lynen furnace. (Ingalls, Metallurgy of Zinc, p. 486.) This condenser is a rectangular chamber of brick, 1.2 meters high, 0.36 meters wide and 3.4 meters long. This chamber is divided into three compartments by vertical, longitudinal fire-clay partitions. The gases are discharged into the two outer compartments by the retorts, 36 retorts to each outer compartment. The gases move vertically upward in the outer compartments, and uniting at the top move downward in the center compartment to the carbon monoxide outlet. Running longitudinally through the compartments are a number of horizontal iron pipes 6 cm. in outside diameter. The condenser was cooled by blowing air through these pipes. The outer walls of the chamber and the bath of zinc forming the bottom of the chamber were so designed as to be able to absorb no practically important amount of heat, so that the entire condensing was done by the air pipes.

There were 19 of the condensing tubes, 6 cm. diameter and 340 cm. long, making a total condensing surface of 122,000 sq. cm. In normal operation the 72 retorts using this condenser would deliver to it 48.2 kg. of zinc per hour, which is at the rate of 0.40 gr. per sq. cm. per hour. This is the rate we found in use with the standard retort condenser. This Lynen condenser, therefore, had ample cooling surface. Divided over this cooling surface, the flow of heat provided by the zinc in cooling down and condensing amounts to 0.21 Calories per sq. cm. per hour. For the condenser to work successfully we have seen that the air cooling must be such that with this rate of heat flow the condensing surface will keep about 725° on the average and that the cold end will not be below 450° and the hot end not above 850° . This is on the condensing side of the pipes. It will take about 1° drop in temperature to get heat to flow through the iron of the pipe at this rate of 0.21 Calories per

sq. cm. per hour, and it will take a drop of about 179° to transfer this heat from the inner surface of the pipe to the air in the time available as the air rushes along. The temperature of the air as it enters the pipes at the cool end will, therefore, be 450° , less the sum of 1° and 179° , or 270° , and as it leaves the hot ends will be 850° less the sum of 1° and 179° , or 670° . To absorb all the heat given up by the zinc of the 72 retorts and do the absorbing between these temperatures of 270° and 670° will require the passing of 72 cu. meters of air per hour as measured at atmospheric pressure and temperature. This corresponds to a velocity of about 150 cm. per second through the pipes for the warm air. This velocity is good economic practice. The pressure drop between the cool and hot ends of the pipes would have to be kept at the point which would keep up this rate of flow in the air. From the drawing it looks as though the convection flow of air was expected to do this, but convection would produce only a very small fraction of the air current required. If the air had been forced in at atmospheric temperature, 20° , only about $\frac{2}{3}$ as much would have been required, but about $\frac{3}{8}$ of the length of each tube would have been below the freezing point of zinc and so we would have zinc dust in place of liquid zinc. This would soon have clogged up the cooler end of the condensing chamber, whereas with the proper amount of air at the proper temperature all the surface of the tubes would be available for producing liquid zinc.

In this Lynen condenser the width of the condensing space was about 6 cm. and the average velocity of flow of the gases after entering the condenser 7.76 cm. per second, so that the product of the diffusion distance and the velocity was 23.3 as compared with the average of 9.5 which we found as the diffusion factor of the standard retort condenser. This condenser was, therefore, deficient in the time it provided for diffusion. From the factors we should expect that about one-half of the zinc would have time to diffuse over the condensing pipes, and the other half would be swept out of the condenser with the undensified gases. The condenser as it was proportioned had diffusing capacity enough to handle the zinc from 36 retorts, but with 72 retorts it only had diffusing capacity to stop about one-half of the zinc delivered to it.

The following figures show the course of the principal stock exchange values in Germany for 1909 and 1910:

Shares.	Average dividend, 1909-10. %.	Average price, 1909.	Average price, 1910.
Coal and iron	7.87	187.96	195.72
Cement	9.22	179.34	168.55
Land companies	5.75	152.65	147.80
Textiles	7.85	159.99	166.12
Electrical	8.25	150.77	162.58
Shipping	5.25	121.63	129.08
Chemical	15.05	265.20	288.93

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NOTES AND COMMENTS.

The Chicago Section of the American Chemical Society Awards the First Willard Gibbs Medal to Prof. Svante Arrhenius.

The Chicago Section of the American Chemical Society, at its May meeting, held in the Congress Hotel at Chicago on the evening of May 12, awarded the first Willard Gibbs medal bestowed by this section to Prof. Svante Arrhenius, director of the Nobel Institute, Stockholm, Sweden, and of international fame for his work on the theory of electrolytic dissociation. Mr. S. T. Mather, chairman of the Chicago Section, presided, and in his speech indicated the hopes and aspirations of the Chicago Section in founding this medal; he also spoke briefly of the work of Dr. Gibbs as an illustration of the best work done by American research chemists and voiced the hope that in the future their progress in these lines might attain to the eminence reached by Gibbs. President Judson of the University of Chicago spoke on International Bonds of Science, the gist of his address being that as countries are bound more closely together by the achievements of their men of science and become more intimately acquainted by the exchange of visits between scientists of

various countries, it will be impossible to continue the militant attitude now possessed by many of our greatest nations. Mr. Harry Wheeler, president of the Chicago Association of Commerce, spoke on the subject, "Science and Commerce." His definition of commerce was one of the best that could be given. Commerce, he said, today is capital plus science. He spoke to some extent on the achievements of the scientist in the advancement of commerce and paid a high tribute to the contributions which science has already made to the development of our commerce. Prof. Alexander Smith, president of the American Chemical Society, presented the medal to Prof. Arrhenius. In his presentation address, he gave a general summary of the work done by Prof. Arrhenius from the time when he first announced his theory of electrolytic dissociation down to the present time. Prof. Smith pointed out very clearly the value which this theory has been to the modern chemist and explained many properties and reactions of the elements which had been up to that time unexplained. He also showed the distinct general value which this theory had been to industry as well as to science. The relation between the work of Gibbs, in whose honor the medal was named, and Prof. Arrhenius, its first recipient, was also pointed out. Prof. Arrhenius, in his address following the presentation of the medal, gave a brief history of his work on the theory of electrolytic dissociation, from the time when as a student the idea first came into his mind up to the time when the theory became generally accepted by the chemists of the world. His account of his early labors was very interesting. He met the same discouragements and the same failures that most others meet and it was no easier for him to overcome the prejudices and objections of his contemporaries than it is for the young scientist at present to obtain recognition for any new idea. Prof. Arrhenius was looked upon by the faculties of the various universities of that time as something of a heretic. They thought that he was advancing ideas which were very hair brained, and without taking the time to repeat his experiments to verify his results, they universally condemned both him and his work. Prof. Ostwald was the first scientist of rank to take up the ideas of Arrhenius, and by repeating some of his experiments and confirming his results, gave Arrhenius a standing which would have taken him years to attain in any other way. From this time until the time when the ideas contained in the theory were generally accepted, Ostwald and Arrhenius seem to have been rather closely associated. One illustration given by Prof. Arrhenius of the difficulties he encountered was interesting and deserves repeating. He said one day that Ostwald and his (Arrhenius') instructor, Dr. Edlund, were discussing in the laboratory the theories advanced by Arrhenius, not knowing that Arrhenius was anywhere around. Dr. Edlund was remarking on the concept of Arrhenius that the solution of sodium chloride consisted of ions, whereas they called them

at that time radicals of sodium and of chlorine. Edlund said to Ostwald, "How can you believe any such idea as that when we all know what metallic sodium is? It cannot exist as such in water. We know its properties perfectly well and know what it looks like. We also know what chlorine is, a vile smelling gas, imparting a green color to water and giving it poisonous properties. Why, it is absurd to think that these substances are present in a solution of sodium chloride as we see it." It took Arrhenius some time to convince them that there is a difference between ionic sodium ionic chlorine and the elements sodium and chlorine. Few chemists, however, at the present time have doubted the truth of his theory and there are probably few chemists at the present time who have not been greatly benefited in their conception of the properties and reactions of elements and compounds by this theory.

It is the hope of the Chicago Section that it may be able to bestow the Willard Gibbs medal to others of the same high class as the recipient of the first medal.

The idea of the donor of the medal, Mr. William Converse, that he wished to do something for the benefit of the Chicago Section, has in the hands of the members of this section been brought to consummation. None of those present at the presentation of the first medal can doubt the benefit of this both to the Chicago Section and to the membership of the American Chemical Society as a whole. Believing that our readers will be interested both in the man to whom this medal is bestowed and the man in whose honor it is named we are publishing short biographies of both gentlemen. We are indebted to the Scientific American for the use of the portrait and biographical material on Prof. Arrhenius and to the Popular Science Monthly for what relates to Dr. Gibbs.

Svante August Arrhenius.

BY PROF. WILHELM OSTWALD.

Svante August Arrhenius was born February 19, 1859, near Upsala. His ancestors were farmers, and the name Arrhenius is a Latinized derivative of *arena*, meaning a river bank, the name of the family estate in the south of Sweden. His father was superintendent

of grounds of the University of Upsala. At school Svante exhibited remarkable precocity and distinguished himself particularly in mathematics, physics and biology. He entered the University of Upsala in 1876, but in 1881 went to Prof. Edlund in Stockholm. He attained the degree of doctor of physics in 1884. His graduation thesis, "A Study of the Conductivity of Electrolytes," published in the same year, made a profound impression upon me.

I have devoted part of my life to the study of the chemical relation between acids and bases, and when that paper appeared, I had obtained for the specific affinities of various acids values closely agreeing with the results which Arrhenius had reached by a very different road. The method by which

Arrhenius had attacked and partly solved the problem was far more comprehensive and fertile than my own. I was then connected with the polytechnical school of Riga, Russia, and there I repeated and extended the experiments by which Arrhenius had proved the approximate proportionality of the chemical affinities of acids and bases to their electrolytical conductivities, and published the results. Then I went to Upsala to support Arrhenius, whose views were regarded as heterodox, in obtaining a position in the university. A year later we worked together for several months at Riga.

In 1887 Arrhenius published his theory of electro-



Svante August Arrhenius.

Drawn especially for The Chemical Engineer from a photographic reproduction in the Scientific American.

lytic dissociation, probably the most important and prolific of his many achievements, which explained the observed anomalies in the freezing point of electrolytes and interpreted as a dissociation factor the irrational coefficient ($\sqrt{-1}$) in Van't Hoff's then new and much discussed theory of osmotic pressure. The new theory of electrolysis rapidly developed, explaining many well known but little understood phenomena of chemical equilibrium, analytical reactions, the concentration of electrolytes, the galvanic battery, the solubility of gases, etc.

This rapid success spared Arrhenius the waste of energy in polemics which the iconoclast usually suffers, and also assured him means of support and of further scientific activity. In 1891 he was called to the University of Giessen, but he preferred to accept a position in the new University of Stockholm, where his subsequent promotion to a professorship was due mainly to the influence of foreign scientists. In 1897, however, his colleagues chose him as their rector, or president, and he was twice re-elected to this position, which he finally declined in the interest of his scientific work. In 1905 he became director of the Nobel Physical Institute.

The theory of electrolytic dissociation soon gained so many enthusiastic young advocates that it continued to develop without direct aid from the founder, except where such aid was required to refute criticism and enlighten skeptics. In 1890 Arrhenius, Van't Hoff and I were invited to discuss the theory of solution with a committee of the British Association, the members of which were at least partially converted to the new views by their foreign guests, aided by William Ramsay. The new theory made rapid progress against strong opposition in Great Britain. French chemists refused to discuss or consider the theory at that time and they are still far behind their German and British colleagues in this line of research.

Arrhenius published a great many papers in the ten years that followed the announcement of the dissociation theory. In the succeeding decade his duties as rector of the new university, to which he devoted himself with great diligence and success, diminished his scientific output.

Arrhenius has recently taken up the study of serum-therapy and has explained Ehrlich's important discoveries by a theory of unsaturated compounds, analogous to that which exists between weak acids and bases.—Adapted from an article published in "Die Forderung des Tages," Leipzig, 1910. Akademische Verlagsgesellschaft, m. b. H.

Josiah Willard Gibbs.

Gibbs was not, like Edison, Langley, Rowland, the inventor, experimenter or expert in delicate measurements, nor was he the great all-round physicist, like Maxwell, Helmholtz or Lord Kelvin. He was essentially and almost exclusively the mathematician, whose special function was not the discovery of isolated facts

or new methods of experimental procedure, but the introduction of new currents of ideas; and it was the severe and rigorous form in which his ideas were cast that for a long period of time retarded their general adoption by the scientific world.

Oswald, in his interesting "Biologie des Naturforschers," has divided men of science into two classes: The classicists (Klassiker), men like Newton, Lagrange, Gauss, Harvey, who, dealing with a limited number of ideas in their work, seek formal perfection and attain it, leaving no school of followers behind them, but only the effect of the work itself; and the romanticists (Romantiker), who, like Liebig, Faraday, Darwin, Maxwell, are bold explorers in unknown fields, men fertile in ideas leaving many followers and many loose ends of unfinished work which others complete. In the logical perfection of his work and in his unusual talent for developing a theme in the most comprehensive and exhaustive manner, Gibbs was emphatically the Klassiker. But in the scientific achievement of his early manhood he showed something of the spirit of the Romantiker also. His mathematical theory of chemical equilibrium was far in advance of any experimental procedure known or contemplated at the time of its publication, and, although some of his predecessors, like James Thomson, Massieu, Horstmann, had come within sight of the new land and even skirted its shores, Gibbs, with the adventurous spirit of the true pioneer, not only conquered and explored it, but systematically surveyed it, living to see part of his territory occupied by a thriving band of workers, the physical chemists. Cayley, in his report on theoretical dynamics in 1857 expressed his conviction that the science of statics "does not admit of much ulterior development." The work of Gibbs has added to it the immense field of chemical equilibrium and wherever "phases," "heterogeneous systems," "chemical and thermodynamic potentials" or "critical states" are mentioned he has left his impress upon modern scientific thought. It is not without reason, then, that Ostwald has called this mathematician "the founder of chemical energetics," asserting that "he has given new form and substance to chemistry for another century at least."

Josiah Willard Gibbs was born in New Haven, Conn., on Feb. 11th, 1839. His father, who has descended from Sir Henry Gibbs, of Honington, Warwickshire, was professor of sacred literature in Yale College during the years 1821-1861 and was esteemed for unusual scholarship in his day. The son, like many other mathematicians, showed early aptitude for linguistic as well as for mathematical studies, and, entering Yale in 1854, graduated in 1858 after winning many prizes and distinctions in Latin and mathematics. He began to teach mathematics and physics at Yale in 1863, having received his doctor's degree in that year. During 1866-69 he traveled in Europe, studying his chosen subjects at Paris, Berlin and Heidelberg, and hearing the lectures of Magnus, Kirchhoff and Helmholtz. In July, 1871, two years after his return, he was

appointed professor of mathematical physics in Yale College, about the same time that Clerk Maxwell assumed similar duties in the Cavendish Laboratory, at Cambridge. This position Prof. Gibbs held until his death, April 23, 1903. Professor Gibbs devoted his whole life to his work, the interests of his university and his pupils, and apart from the earlier years of travel and some excursions into the field of controversy his was the quiet and uneventful career of the typical man of science.

Gibbs began his work in thermodynamics in 1873, with two important papers on diagrams and surfaces. In the first of these he made a careful and thorough-going study of all the diagrams that might be of use or value in thermodynamics, the best known being that upon which volume and pressure are erected to scale as co-ordinates, derived from the familiar Watts indicator diagram found upon every steam engine.

Of the new diagrams which Gibbs introduced, he attached most importance to the volume-entropy diagram, because it tells more about the physical properties of a working substance than about the heat employed or the work done. But the most important of Gibbs's innovations for practical engineering purposes is the temperature-entropy diagram, which represents the efficiency of a Carnot cycle as a simple rectangular figure and is, he points out, "nothing more nor less than a geometrical representation of the second law of thermodynamics."

During the years 1875-8, Gibbs published the work which is his chief title to fame, his memoir "On the Equilibrium of Heterogeneous Substances." This treatise deals, as the title implies, with the statics of chemical substances which, as gases, vapors, liquids, or solids, are in actual physical contact with each other, whether influenced or modified by gravity, osmosis, catalysis, capillarity or electromotive force or existing under such varied aspects as gaseous mixtures, liquid films, "solid solutions" or crystals. For the first time chemical substances are treated as continuous or continuous "phases" of "matter in mass" acted upon, like mechanical systems, by forces having "potentials"—a new way of looking at things which has since become the definite viewpoint of physical chemistry.

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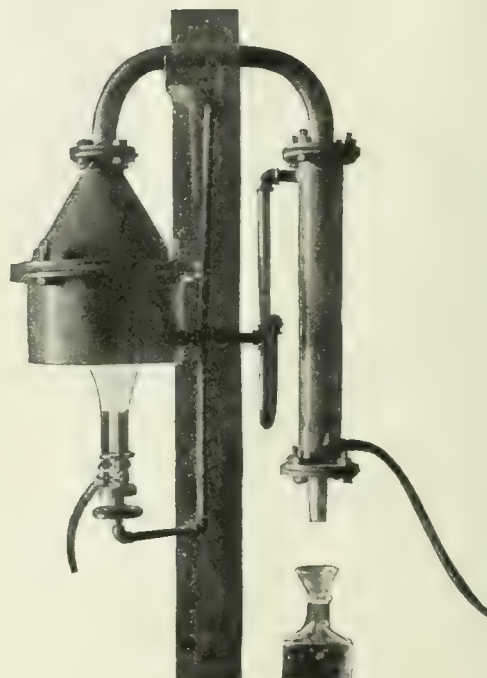
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THE CHEMIST IN THE SERVICE OF THE RAILROAD.*

By H. E. SMITH.†

Among the industries of the country, the railroads are probably the largest, whether they be judged from the standpoint of investment required, territory covered, or men employed. Considered as manufacturing establishments, the railroads use as raw materials, iron and steel of all kinds, brass, bronze and babbitt, wood and timber of all kinds, stone, brick, cement, oils and paints, and a great number of materials of lesser importance. The manufacturing process covers various departments of field engineering, power production, shop work and metallurgy. The finished product is the transportation of passengers and freight.

As in numerous other manufacturing industries, the equipment required is constantly becoming heavier, more elaborate and expensive. The methods of operation are constantly becoming more complicated, more specialized and more costly, yet the piece price of the product—transportation—is constantly diminishing.

This condition evidently necessitates increasingly scientific management and operation. The need of the civil, the mechanical and the electrical engineer is obvious. With the increasing development has come the need of the chemist and, beginning nearly 30 years ago, the Massachusetts Institute of Technology has at various times furnished chemists for this service.

As has already been partially indicated, the field for the chemist in railroad service is very wide. Early in the construction of a first-class road the chemist is in demand for the testing of cement, to ensure sound and durable concrete bridges and other structures; in the selection of ballast stone of such composition and physical properties that it will withstand the weather and the impact and wear of service. Timber for crossties now commands such a price that economy necessitates preservative treatment by carefully tested and regulated materials and processes. Rails must be of such composition that they will resist wear in the greatest possible degree, yet be free from brittleness.

The problem of boiler water supply, especially that for locomotives, is of very great importance. An average modern locomotive is a complete power plant

of 2,000 hp., mounted on wheels, and contained within a space only a fraction of that required by a stationary power plant of the same capacity. About 100 gals. of water are evaporated per mile traveled. In many parts of the country the only water available is of such quality that economy in operation requires its chemical purification. This brings to the chemist the problem of carrying out chemical reactions on a large scale, and in extremely dilute solutions, yet with very close adjustment and at a low cost.

Iron castings are used in such quantities that railroad companies frequently make their own. The pig irons and the coke must be tested to ensure proper quality. The mixtures must be adjusted to secure the necessary product. Car wheels must be tough and strong yet very hard on the circumference, to resist wear. Machinery castings must be strong yet machine easily. Packing rings must be very resilient. Yet in all mixtures economy must be practiced by using up scrap.

The proper selection of steel for efficient and economical service is a constant problem. Special alloy steels either with or without special heat treatment or other manipulation are used in increasing quantities. It is necessary to make very frequent and systematic chemical and physical tests to ensure uniform and satisfactory quality.

Large quantities of brass or bronze, mostly in the form of bearing metal, also babbitt for the same purpose, are required. The prices of the constituent metals vary from 5 to 35 cts. per pound, which constitutes a strong temptation to substitute the cheaper for the more expensive, so far as possible. The chemist must be called upon to detect such substitution and to determine whether the constituents have been properly proportioned. Scrap must be utilized and the chemist is needed to test the remelted metal and adjust its composition to standard figures.

Paints are used in large quantities on cars, buildings, bridges, etc. The pigments are frequently adulterated with inferior, inert or injurious minerals, the linseed oil with petroleum and inferior vegetable oils and the turpentine with benzine. The proportions of the specified ingredients must also be checked. Lubricating and burning oils constitute another important class of material for study, to secure the proper selection of grades and maintenance of satisfactory standards.

*Paper presented before the Congress of Technology at the 50th Anniversary of the Granting of the Charter of the Massachusetts Institute of Technology.

†Chemist and Engineer of Tests, The Lake Shore & Michigan Southern Ry. Co., Collinwood, O.

The list of articles which come up for occasional attention includes soaps, greases, roofing materials, fire-proofing materials, various chemicals, dyestuffs, ink, grinding and polishing materials, disinfectants, rope, cotton and woolen waste, fuel, etc., etc.

Not only must all these materials be examined after purchase but many of them must be bought on definite specifications in order to secure the desired quality under competitive bids. The writing of the specifications falls chiefly upon the chemist. To this end he must study carefully the needs of the service, the quality of material best adapted to meet those needs, as well as the quality available on the market, and finally the methods of test best adapted to determine the quality. This exhaustive study also may be the means of ultimately developing or improving various industries so that it is beneficial not only to the consumer but to the producer.

It is natural that the chemist should also be required to study various methods and processes of operation in different parts of the railway service, with a view to the economy of material or labor or to increasing the efficiency of the service.

In all the work above described the investigator must be more than a chemist. He must be something of a geologist, a physicist, a metallurgist, and electrician, or a sanitarian, as the case requires, and withal must have the ability to predict the effect from the cause, or to trace back from the effect to the cause.

It is for these reasons that the broad and comprehensive training offered by the Institute of Technology is especially adapted to fit men to take up scientific work for the modern railway.

According to a U. S. Consular Report a German chemist, Dr. A. Eichengruen, is reported to have discovered an incombustible substitute for celluloid. Numerous attempts have been made to find an acetylcellulose product that would replace celluloid, which is made of nitrocellulose. Incombustible compounds have been obtained, but none which could take the place of celluloid. The acetylcellulose products could not be used in the manufacture of all sorts of objects as can celluloid. For instance, it is impossible to make objects more than one-fourth of a millimeter in thickness out of cellit, of which incombustible cinematograph films are made. Dr. Eichengruen's product, called "cellon" (also an acetylcellulose product), is reported to furnish a satisfactory commercial substitute for celluloid which can be made into all sorts of objects. According to reports it can be colored in any way that celluloid can and also be made to imitate tortoise shell. When brought into contact with a flame "cellon" melts, but does not take fire. By a similar process Dr. Eichengruen secures a "cellon" varnish. According to newspaper reports, preparations have been made to manufacture "cellon" and "cellon" objects on a large scale.

THE ECONOMIC UTILIZATION OF LIGNITE COAL.*

By EARLE J. BABCOCK.†

North Dakota is fortunate in possessing an enormous supply of fuel in the form of lignite coal, and this is going to prove one of the greatest inducements the state has to offer for the establishing of industries. And so it is well that our attention be called to the benefits to be derived from the proper utilization of our great lignite deposits. I believe that these coal and clay resources are to form the basis for a development which will gradually convert North Dakota into a rich mining and manufacturing as well as an agricultural state.

The extent and economic importance of the lignite coal deposits of North Dakota are seldom recognized. There is a very large area in the western part of the state which is underlain with quite thick deposits of lignite. Indeed North Dakota is one of the largest coal states of the Union. It is estimated that she has a coal area of 32,000 square miles, capable of producing 500,000,000,000 tons.

To the people of the state the economic utilization of our lignite, in whatever form it may be, is a matter of very great importance, and the new fields which promise to open up for the use of lignite in the production of briquets and gas are well worth careful consideration.

For this reason there has begun at the College of Mining Engineering of the University of North Dakota and Mining Sub-Station some extended work on the practical tests of a variety of methods of manufacturing briquets and producing gas from lignite, with a view to showing the utility and economy of these products for heating, for power and for lighting purposes. In addition to this considerable work has already been done with reference to other improved methods of burning and utilizing lignite coal.

The brief statements which are here given will serve only as a summary of the work which has been done, the results thus far secured, and the promises which these results hold out for future development of our lignite fields. But what has been accomplished in our experimental work at the School of Mines and Sub-Station leads me to believe that great improvements can be made in methods of burning and utilizing lignite coal and that the manufacture of gas can soon be put on a successful commercial basis.

This will not only make our lignite coal much more serviceable and more generally used and of high enough value to be shipped for considerable distances in competition with other coal, but it will also save an immense amount of slack coal which would otherwise

*From The Quarterly Journal of the University of North Dakota.

†Dean of the College of Mining Engineering, University of North Dakota.

be wasted. In order to understand better why our lignite deposits have not been developed more rapidly and to see what changes are needed in lignite to increase its value and usefulness, we may well consider briefly the character of lignite.

From the many hundred of analyses of lignite coal made in our laboratories, the average composition of the dried coal would be about as follows:

Dried coal	Approximate average.
Fixed carbon.....	52 per cent.
Volatile matter.....	40 per cent
Ash.....	8 per cent.
Moisture from coal fresh from the mine runs from 25 to 35 per cent.	

It is evident from the investigations which have been carried on at the School of Mines that the heat value or calorific power of the average lignite of North Dakota, when entirely dry, is about 65 per cent that of the Hocking Valley coal, and about 60 per cent of that of the Pocahontas coal. The analyses which have been made show that the fixed carbon in the samples of North Dakota coal (dried) usually runs from 48 to 52 per cent; that of West Virginia bituminous coals about 67 per cent; that of the better grades of semi-bituminous dry coals of Maryland 75 per cent; and that of the fine, dry, close burning bituminous coals of Pennsylvania 70 per cent. In a general way it may be said that the heating power of one ton of North Dakota lignite, if it does not contain too much moisture, will equal about 65 per cent of a ton (1,300 lbs.) of bituminous coal.

In general appearance this coal is between a cannel coal and the brown lignite of Germany. It breaks up very easily when exposed to the air. This is no doubt due in large part to the rapid evaporation of water, there being usually from 25 to 35 per cent of moisture in the lignite as taken from the mines. The larger portion of this moisture escapes rapidly and causes, thereby, checking and splitting of the lumps into lignite slack. This is one of the greatest difficulties to be overcome in the commercial handling and utilizing of lignite.

In shipping coal containing so much moisture there is a heavy loss in freight and a corresponding reduction in the efficiency of the fuel. It is therefore important that the moisture be removed before shipment. But when this is done the fact that the coal still tends to become slack presents another difficult problem. As a result the utilization of lignite has been confined within comparatively narrow regions near the deposits. It would be an enormous stimulus to the utilization of lignite could it be satisfactorily and economically converted or concentrated into fuel free from moisture and of a size adapted to general commercial uses. These facts have directed our attention to the importance of some economical means of briquetting lignite.

At the present time lignite coal is chiefly used in lumps in heating and cooking stoves and in engines for

power. It is generally used in the most simple manner and very few special methods have been adopted for burning or utilizing this coal or for preparing it for the market. But there can be little doubt that for general stove and furnace use briquetted lignite would prove a most economic and profitable fuel. Lignite, however, has presented more serious difficulties to the process of briquetting than has bituminous coal. For this and other reasons the commercial briquetting of lignite has not, up to this time, been made a success in this country. There is little doubt, however, that a successful and sufficiently inexpensive method for the commercial briquetting of lignite is now being developed at the School of Mines and the Mining Sub-Station. The process which we are now working out, and which will be briefly explained in the following pages is made commercially possible and profitable by operating the briquetting in conjunction with a gas-producing plant in which a variety of by-products are saved and utilized, and by which process the residue after the gas has been driven off can be successfully briquetted into a concentrated and valuable fuel.

A large amount of work has been done by the writer on the problems of briquetting lignite, its use in the production of gas and other means of its utilization. During the progress of this work it has become clearly evident, by repeated experiments, that lignite coal contains two very widely different fuel constituents, namely, first, a large proportion of light gases, which pass off at a comparatively low temperature leaving, second, a residue, in character between coke and charcoal, which requires a very much higher heat for combustion and burns more slowly. This residue does not coke but remains as powder or slack which checks the draft and is slow in oxidation. The bases, however, are very light and are easily driven off from the coal at a comparatively low temperature. As a result they are distilled off quickly, in the ordinary methods of burning, and are either burned with a flashy flame or frequently lost, in a large degree unconsumed with the flue gases. Thus there is a very uneven combustion of lignite coal and it is impossible with the ordinary type of furnace and firebox to secure a high degree of efficiency in its consumption. Following out the investigation suggested by these observations the writer has been led to inaugurate a systematic research on the production of gas from lignite coal and the utilization of the by-products. The first work along this line was of a preliminary nature and was carried on in a small way with laboratory apparatus in testing out a large number of both lignite and ordinary gas coals for the purpose of obtaining a general idea of the comparative quality and character of the gas, tar and residue, of the temperature required for most efficient work with the various coals and an estimate of the amount of ordinary by-products obtained from each coal under the conditions employed.

One of the first problems to be studied was as to the uniformity of lignite relative to the gas contents,

the character of the coke residue, the amount of moisture present, the tar, ammonia and other by-products. To cover these points a large number of lignite coals were collected from different areas and these were subjected to a great variety of preliminary tests. This work required much time and labor, but gave most valuable data which were utilized in planning the later and more systematic attacks on lignite gas problems. The results proved conclusively that the gas from the lignites of different deposits is very similar, that it may be removed at nearly the same temperature and that it leaves a residue and by-products of a general similarity although the quantity of each varies considerably. The main products and by-products thus obtained were compared and those obtained from ordinary gas coals tested by the same processes. All of this preliminary work was very suggestive as to the possibility of the profitable production of lignite gas.

The small plant just described located at the School of Mines was the forerunner of the large specially designed briquetting and gas plant located at the Sub-Station at Hebron where both the briquetting and gas problems are being worked out on a scale so large as to ensure commercial results. The briquetting press is capable of turning out two tons of briquets per hour while the gas plant is a full size unit complete and of the standard type of construction.

The equipment of the large experimental gas plant at the Mining Sub-Station consists of a bench of one retort, one condenser, one exhaustor, two scrubbers, one purifier, one meter, all of the full size used in commercial plants, one gas-holder and miscellaneous apparatus necessary for the proper operation of the plant.

LIGNITE GAS.

The principles of construction permitted of its being operated in a general way very similar to that employed in many commercial types of gas plants. Every step of the operation was carefully watched and made to approach as nearly as possible for so small a plant the methods which might ordinarily be expected to be required for economic production. The quality of gas produced by this plant is probably inferior to that which could be secured in a large commercial plant, owing to the fact that in stopping and starting and carrying on the experimental work there is frequently introduced into the retorts, apparatus, and pipes varying amounts of air, a condition which it is impossible to eliminate in an experimental plant. This is of course detrimental to the gas since it gives a larger proportion of carbon dioxide and of residual air. Nevertheless this experimental plant has given excellent results and indicates much wider possibilities as to the amount and quality of gas from lignite coal than was expected. There was generally used for each charge of the retort 300 to 400 lbs. of air-dried coal, the moisture content of which was definitely known. The coal was crushed to an average size which would about pass through a 2-in. ring. The lid of the retort

was well clamped and sealed and the temperature increased gradually to the required degree. A series of experiments were conducted to determine the temperature giving the largest quantity and the best quality of gas. It was found that the gas distilled freely at 900° to 1,000° F., while the proper temperatures to give the maximum quantity and quality of gas seemed to be between 1,200° and 1,400° F. High temperature readings were made by the pyrometer along with simultaneous records of the rate of gas production. At frequent intervals gas samples were obtained for analyses. The desired heat was carefully maintained until the evolution of gas ceased, after which the

LIGNITE GAS ANALYSIS.

(Considered average standard results of unpurified gas.)

Sample Series	Illuminants	Carbon Monoxide	Hydrogen	Ethane	Methane	Nitrogen	Oxygen	Carbon Dioxide
A.....	3.11	17.24	45.16	0.14	15.59	5.49	1.76	11.51
B.....	2.35	18.70	45.45	1.02	14.42	5.34	1.21	11.51
C.....	2.10	19.24	41.26	0.39	16.31	4.30	1.00	15.40

GAS YIELDS AND HEATING VALUES.

Sample Series.	Unpurified.		Purified from Carbon Dioxide and Air	
	Yield Cu. Ft.	Calorific Value in B.T.U.	Yield Cu. Ft.	Calorific Val. B.T.U.
A.....	10,663	400.07	8,523	500.
B.....	11,306	406.60	9,347	491.8
C.....	11,801	399.8	9,416	501.1
Average.....	11,257	402.2	9,095	497.6

Retort Residue Yields.

Sample series.	Yield per ton dried lignite.
A	1208
B	1233
C	1266

Average 1236

Comparative Calorific Values.

Lignite as mined.....	7,500- 7,800 B. T. U.
Lignite retort residue.....	10,500-11,500 B. T. U.
Lignite briquets.....	12,000-12,500 B. T. U.
Anthracite coal.....	12,500-13,500 B. T. U.

TAR AND AMMONIA.

Tar per ton dried coal.....	.20 to 30 lbs.
Ammonia sulphate per ton dried coal.....	14.5 to 15 lbs.

COMPARATIVE COAL ANALYSES.

	Moisture.		Vol. Matter.		Fixed Carbon.		Ash.
	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	
Lignite as mined.....	35.01	25.11	34.67	5.21			
Lignite briquets.....	0 to 6	2 to 8	75 to 85	10 to 14			
Anthracite coal (a market sample)....	3.68	5.26	80.51	10.55			

valves were closed, the non-coking residue drawn, cooled down and carefully weighed. The gas was passed successively through the tar extractor, the condenser, the scrubber, the purifier and finally to the gas-holder, where the quantity was accurately read off from the holder after having been carefully corrected to normal temperature and barometric pressure.

After the gas was measured, analyzed and tested by the calorimeter to determine its heat value, it was used with different types of burners and in various practical ways to demonstrate its efficiency as a fuel gas. In the case of the by-products the tar was separated from the water and weighed and the ammonia

separated and determined. The resulting proportion of non-coking retort residue remaining was also determined and an analysis made indicating the fixed carbon, volatile matter and ash. From this data the composition of the residue could be compared with that of the original lignite and also with that of bituminous and anthracite coal.

The calorimeter determinations were made with the Sargent gas calorimeter properly standardized.

The gas made from the lignite has not only been subjected to a great number of chemical and calorific tests, but has also been put to a variety of practical tests. It has been burned in the ordinary manner for lighting purposes, and gives a most satisfactory and brilliant light with the ordinary mantle burner. It produces a clear smokeless flame and is easily regulated, with proper adjustment, so that the light is equal to that produced from any city gas. It is almost free from sulphur without purification. In fact it seems entirely unnecessary to purify the lignite gas. This saves materially in the cost of manufacture. All that seems necessary is the usual scrubbing with water which is of course a very simple and inexpensive method.

We have found from a large number of tests that the expensive lime or iron purification required in most city coal gas is unnecessary with lignite gas. This, however, could be done by the same methods usually employed and if the gas were purified in this manner to correspond with the treatment given the ordinary coal gas the composition and heating value would be considerably modified. The heat units developed would probably be increased to between 450 and 500 B. T. U. as compared with standard gas of 600 B. T. U.

By this purification and complete removal of air a value of 500 B. T. U. could be obtained, but as already stated the character of the gas and the freedom from objectionable impurities make unnecessary the additional expense of purification usually required of coal gas. The gas is dry and free from tar and other matter which would cause condensation or clogging. This is a material aid in the practical distribution and use of gas especially in cold climates. The most severe and continued tests have been given to determine the condensation produced in carrying the gas in slightly protected or unprotected pipes during the cold weather. In this case no trouble was experienced from condensation.

The gas was used successfully not only for lighting but it gave perfect satisfaction and exceptionally high efficiency. The power tests have not been completed and have only been carried sufficiently far to prove that the gas can be used with entire satisfaction for power in gas engines. Several tests have been run with modified gasoline and gas engines at the Sub-Station and at the University. At the latter place the tests were made in comparison with standard city gas. It was found that the lignite gas is satisfactory and

will develop a relatively high efficiency. It is almost certain that this gas will produce a very cheap power for general purposes and for the production of electricity.

It is expected that much more extended tests along these lines for the utilization of this gas for power and the production of electricity will be carried on during the coming year. There is little doubt that the use of lignite gas for power purposes can be successfully employed and power produced at so low a figure as to exert a marked influence in the development of a variety of industries dependent upon cheap and abundant power thus available in the western part of the state.

By reference to the tables it will be seen that air-dried lignite produces in the experimental plant 11,000 cu. ft. of gas per ton of lignite. This is an excellent gas production from lignite coal. Moreover it will be seen from the analyses that the gas is of fairly good quality, yielding about 400 B. T. U. unpurified and nearly 475 B. T. U. when purified and free from carbon dioxide. The retorting is done at comparatively low temperatures thereby reducing considerably the cost of production. The gas is rather high in carbon dioxide and hydrogen, as will be seen by the analyses, but as has already been stated this fact is undoubtedly due in a large degree to the moisture and air which are absorbed on account of the porous character of the lignite. Part of this air, however, is derived from the retort and pipes, a condition unavoidable in stopping and starting an experimental plant, but which would be much less proportionally if the gas were manufactured continuously in a large plant. In retorting, the oxygen from the air, present from the sources just mentioned, combines naturally with the carbon forming a larger proportion of carbon dioxide and carbon monoxide, thus leaving a relatively large amount of nitrogen. These conditions might be improved in a large plant in continuous operation.

The yield of ammonia is seen to be quite good, a condition to be expected from lignite coal. This ammonium sulphate is of very good commercial value and is extensively used as a fertilizer.

By reference to the tables the yield of tar is seen to be approximately 1 to 2 per cent. The exact determination however of the quantity and character of this tar has not been worked out yet. It seems to be rather high in paraffin ingredients of value. By special methods of treatment it may be quite possible to utilize a large proportion of the tar, in connection with other material, as a partial binder for briquetting the non-coking residue and thus reduce to a considerable extent the cost of lignite briquets.

This lignite tar is not adapted in itself, and without modification, to briquetting purposes. We have succeeded in using it very satisfactorily in small quantities by special methods of treatment which the investigations thus far carried on indicate as a possible method of utilization on a commercial basis.

Considering the ease with which the gas is produced, the low cost of the original lignite and the value of the residue, this gas should have a large commercial value for heating, lighting and power purposes, because of the low price for which it could be sold if manufactured in a plant used also to produce briquets from the residue.

It has been found that this concentrated residue has a high heating value when briquetted, producing a most excellent fuel, for all practical purposes equal to anthracite coal, as seen by calorimeter and stove tests. One ton of dry lignite will produce between one-half and two-thirds of a ton of these briquets in addition to 10,000 to 12,000 cu. ft. of good gas and other valuable by-products. Briquets of such excellent quality and high fuel value (at least 12/13 the actual heating value of hard coal), can be sold at a good profit at the price of soft coal in the western part of North Dakota and materially lower than hard coal in the eastern portion of the state. Having such high heating value and being so strong these briquets can be shipped for considerable distances and prove an excellent and economic fuel.

BRIQUETS.

The briquets present many advantages over the original lignite or even over the other varieties of coal. They have nearly double the heating value of the original lignite, as usually placed upon the market, they do not disintegrate on standing or burning, can be stored without being affected by atmospheric conditions, and are uniform in size and convenient to handle.

As has already been seen the process which we have developed for the successful manufacture of briquets from lignite coal is intimately associated with the production of gas; and according to this system a complete briquetting plant would also include a simple gas plant or at least the carbonizing portion. For these reasons the discussion of the manufacture of gas from lignite and an explanation of the gas plant have preceded that of briquetting.

The principles underlying the process of lignite briquetting which we have worked out at the School of Mines and the Mining Sub-Station depend upon the concentration of the raw lignite into a fuel of relatively high heating value, of permanent character and of sufficient value to be shipped in competition with anthracite coal.

This concentration depends upon, first, the removal of the moisture, second the expulsion and saving of the volatile gas, and third, the binding together of the concentrated residue into a strong, durable and satisfactory briquetted product. As the coal is removed from the mine it contains a very high per cent of moisture, usually more than 30 per cent, i. e., about one-third of every ton of raw lignite as mined is water. This fact is largely responsible for the difficulty in the successful introduction of lignite except in the immediate vicinity of the mines.

The cost of shipping fuel containing so much water is prohibitive for long distances. In addition to this the heating value of the coal is reduced by the amount of water present. On standing a portion of the water evaporates from the coal but in so doing tends to reduce the lumps to slack, causing thereby a large amount of waste. In converting the raw lignite into the condensed briquets, the first step is therefore the removal of this water. The lignite is run through a rotary drier which removes from 75 to 90 per cent of the moisture.

After the coal has been sufficiently dried it is conveyed to the retort and the gas driven off as already explained. This process is usually completed in about four hours. At the end of this time the retort is discharged and the remaining coal residue quenched and crushed ready for the mixing with the binder. The coal and the binder are then warmed to the proper temperature and conveyed to the mixer where a very thorough blending of coal residue and binder takes place.

At this stage lack of uniformity and proper mixing produces a briquet which is uneven and of inferior quality. The temperature and the proportions of material must also be carefully watched. This is one of the most particular phases of the briquetting process. When the mixing has been carried on for a sufficient length of time the material is discharged and conveyed directly to the briquet machine.

Besides the conditions which have already been mentioned there are numerous details of mixing, tempering, etc., which can be properly understood and controlled only by experience. Conditions vary greatly with the kind and character of the binding material used.

The press used at the experimental plant is of the rotary type and is capable of producing two tons of the briquets per hour. It was built to order, especially for the Mining Sub-Station.

This machine is extremely simple in its design and operation. The material to be briquetted is fed between two rolls, the surfaces of which are covered with pillow shaped pockets, half of a pocket being on the under and half on the upper roll. The rolls are so geared that the upper and lower portions of the pockets exactly correspond as the cylinder rotates. The coal mixture is fed between the two rolls and pressed into the pockets, by which process firm, dense briquets are made and as the rolls revolve the briquets are discharged to a belt conveyor by which they are carried to the storage bins and the briquetting process thus completed. The briquets are of convenient shape and size weighing about $2\frac{1}{4}$ ounces each.

Briquetted fuels have been extensively used in Germany and other parts of Europe where a variety of methods of manufacturing have been employed with varying degrees of success. With certain European coals briquet blocks have been produced without a

binder, while in other cases lime, sugar wastes, starch, pitch and other binding materials have been employed. Numerous attempts have been made in this country to briquet coal without a binder by merely excessive pressure but this has proved unsatisfactory. By such a process the expense for power and operation nearly equals the additional cost of the binder. For these and other reasons no satisfactory briquets of American lignites have been produced on a commercial scale without a binder.

In most of the briquetting work of the School of Mines and the Sub-Station, we have used some form or combination of pitch binders, but we have also used many other binders such as flour, starch, gypsum and various combination binders. About 200 tests have been run with different proportions and other varying conditions. Up to the present time our best results have been secured, however, in the use of a small amount of pitch as a binder base mixed with a small per cent of other materials mentioned.

In the process of gas manufacture we have found that between 20 and 30 lbs. of pitch is obtained from carbonizing one ton of dry lignite. By a simple process which the writer is now at work on, this pitch can probably be utilized for a considerable portion of the binder material thus reducing to a minimum the use of other pitch, requiring probably not more than 4 per cent of pitch in addition to that derived from the lignite. Or if this lignite tar is more valuable than ordinary pitch, as now seems probable on account of its peculiar composition, it may be sold at sufficiently high price to go a considerable way in reducing the cost of the other binders required in briquetting.

Attention has already been called to the use of the gas in connection with the briquetting and to the revenue which may be derived for the plant by a combination of gas and briquetting plants, but in addition to this, as already has been pointed out, there is a considerable amount of ammonium sulphate formed (nearly 15 lbs. per ton of dry lignite) and this by-product has a ready market. Thus we see that by saving the various by-products and applying the revenue derived from them to the manufacture of the briquets the cost may be very materially reduced. In a large commercial plant the saving of a few cents per ton will be sufficient to net a good profit.

As has already been stated a great variety of tests have been made at our Sub-Station but still other tests will be run at fast as opportunity and funds permit. Some further improvement and reduction of costs can do doubt be made, but even now the results have proved very satisfactory.

The briquets which we are now able to run out regularly and in considerable quantities are first class in every respect. They are strong, hard, durable and will stand weathering excellently and burn with entire satisfaction in most any type of stove or furnace. They have been tested in a very great variety of ways and have always made the best showing.

The chief advantages to be gained in the briquetting of lignite are the large gain in heat value, the prevention of slack so characteristic of this coal, the holding together of the mass during combustion, the prevention of matting in the firebox, free burning, uniformity in size, cleanliness in handling, and decrease in the loss, caused by the dusting. In addition to these points briquets can be used to better advantage in a variety of stoves, and may be kept under more perfect control than the ordinary form of lignite. But the greatest advantages are the increased heating power and the fact that the lignite is put into a satisfactory form which will stand weathering, handling and burning without disintegration.

There is comparatively little loss through the grate bars, the briquets burn to a perfect ash in which practically all of the fuel has been consumed. Anthracite coal leaves a considerable amount of clinkers containing a greater or less amount of black imperfectly burned fuel or carbonaceous matter. The complete combustion secured in lignite briquets is greatly in their favor from the point of economy.

The calorific determinations were made with the latest improved standard bomb calorimeter. These determinations show the following:

Lignite as mined.....	7,500 B. T. U.
Lignite briquets.....	11,500 to 12,000
Anthracite coal.....	12,000 to 13,000

It will be seen that lignite briquets have raised the heating power of the lignite about 70 per cent and give a calorimeter value of approximately twelve-thirteenths the heating value of anthracite coal. The chemical analyses of the briquets as compared with a raw coal are shown by the following table:

	Moisture.	Vol. Matter.	Fixed Carbon.	Ash.
	Per cent.	Per cent.	Per cent.	Per cent.
Lignite as mined.....	35.01	25.11	34.67	5.21
Lignite briquets.....	0 to 6	2 to 8	75 to 85	10 to 14
Anthracite coal (a market sample)....	3.68	5.26	80.51	10.55

Many practical burning tests have been made using the lignite in comparison with anthracite coal and the best of Youghiogeny soft coal. They have been used under boilers and have proved a very excellent and economical steaming fuel. They have been used in hot air and hot water heating plants with excellent results, in every way equal to anthracite in the heat produced and the length of time they lasted. They make a very pleasing grate fire, much easier to control than either hard or soft coal. In the kitchen they have proved particularly satisfactory; they are easily kindled, produce a high heat with a slight flame which is free from soot, leaving the stove and lids clean and producing an excellent fire for cooking and baking. They hold the heat well, last easily over night and can be controlled readily by operating the grates and drafts. They have burned with excellent results in the direct draft heating stove and self-feeder base burner hard coal stove. In order to determine the practical value of lignite briquets as compared with anthracite coal and as compared with raw lignite a

number of very carefully conducted stove tests were made.

After running a number of tests with a special direct draft stove giving evaporative as well as heating tests and in the hard coal self-feed and base burner, an average of the tests was made with the following relative results:

EVAPORATIVE AND HEATING TESTS.

	Per Cent.
Per cent of evaporation of anthracite coal considered.....	100
Per cent of evaporation of lignite briquets.....	121
Per cent of heating value of anthracite coal considered....	100
Per cent of heating value of lignite briquets.....	126

The manner in which the briquets and the anthracite burn in each case and the total heat produced and the duration of the fire may be readily seen by the curves shown in Fig. 1. From this it will be seen that burned in a regular hard coal stove the briquets made of lignite residue gave about $\frac{1}{4}$ more available heat than did anthracite coal.

In the regular process used for the manufacture of illuminating gas from coal there is left a residue, generally known as coke, which has considerable market value. Heretofore the difficulty in the use of lignite for the production of gas has been that the carbon remaining behind did not form into lumps like coke but broke up into small particles which were of no commercial value. Thus while the amount of gas produced from lignite was large, the waste of non-coking residue was so great as to render the manufacture of gas from this coal unprofitable in most parts of the state. The method of combining briquetting and gas production just described and which has proved so satisfactory in our investigations has overcome this difficulty.

But this method of gas production must not be confused with a new system known as the producer-gas method for the production of gas from anthracite and other coal waste and well adapted for generating power in specially designed gas engines and for certain other purposes.

PRODUCER-GAS.

Producer-gas is derived from incomplete combustion of the fuel held in a specially devised combustion and gas producing chamber. Unlike the production of ordinary coal gas, there is no residue of coke left in the producer but the coal is in part converted into gas by the combustion of the remaining portion. To bring about this transformation to the best advantage and to supply a large amount of hydrogen and carbon monoxide, steam is introduced into the chamber during the burning of the coal.

The development of producer-gas for power has been quite rapid although the details of the problem with reference to lignite coal have not been settled by any means, still the results which have been achieved are such as to promise much in the use of lignite by this method. Many fuels of comparatively little value, as used under boilers in the old direct burning method, have been made to yield very excellent results when utilized in the producer gas plant.

The more perfect utilization and saving of our coal and the cheaper production of power thus brought about are items of great interest to every citizen. Whatever method will bring a cheapening of power will result in great saving and benefit to all classes.

Until four or five years ago only very slight attempts had been made to utilize lignite as a gas producer. During the time of the World's Fair at St. Louis a model producer-gas plant was put in operation by the United States Government. Surprisingly successful results were obtained in the use of different coals for the production of gas for power. The laboratory work which had already been carried on at the School of Mines indicated at once the possibility of successfully using lignite coal in this producer-gas plant.

The writer therefore arranged for the testing of two or three carloads of North Dakota lignite at the government testing plant at St. Louis. The results secured were very satisfactory, for it was found that North Dakota lignite used in the producer-gas plant gave more power than many of the eastern bituminous

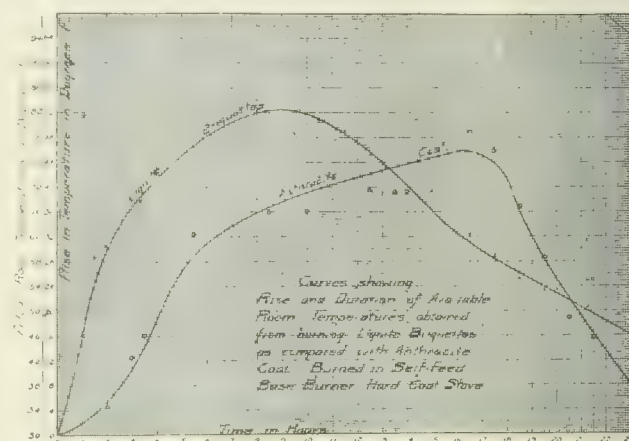


Fig. 1—Comparative Burning Qualities of Lignite Briquettes and Anthracite.

coals burned by the old method. The gas can be used also for kiln and special types of furnaces.

The composition of producer-gas varies greatly with the different fuels from which it is derived. The following table based upon the U. S. Government reports gives the analysis and relative heating and power producing quality and the comparative consumption of coal per unit of power produced by the steam engine and the producer-gas from these different grades of coal, including North Dakota lignite:

	Virginia. Anthracite.	Ohio. Bituminous.	North Dakota. Lignite.
Carbon dioxide.....	10.2	9.0	8.69
Carbon monoxide.....	19.1	20.2	20.90
Hydrogen.....	20.5	15.3	14.33
Methane (CH ₄).....	1.9	2.7	4.85
Nitrogen.....	48.2	52.3	51.02
Ethylene (C ₂ H ₄).....	0.1	0.5	...
B. t. u. per cu. ft. gas.....	160.7	165.2	164.1
Dry coal gas consumed by producer plant per h. p. switchboard per hour, lbs. . .	1.29	1.44	2.29
Dry coal consumed in steam engine per h. p. switchboard per hour, lbs.....	4.46	3.97	...

Nitrogen and carbon dioxide gases are not considered directly serviceable in the production of power, therefore the aim is to secure as small quantities of these gases as possible but to increase the proportion of hydrogen, carbon monoxide, and methane.

While this is true that, per cubic foot, the heat value of producer-gas is much below natural or coal gas, yet this fact is more than compensated for by the large quantity of gas produced per pound of coal. The heat units of producer-gas amount to about 1.5 to 1.6 that of the ordinary city gas, but the volume of producer-gas gotten from a ton of coal is so large as to more than compensate for the difference in value. Because of the relatively large amount of gas produced, and the ease and cheapness of the process, the final cost per thousand feet of producer-gas is very low.

The power developed and the results secured from North Dakota lignites are remarkable in comparison with other fuels, especially when compared with the usual method of using the lignite as a steam producer. There seems to be excellent possibilities in the utilization of this method in connection with our lignite coal and wastes.

CONCLUSIONS.

From the tests so far made in the briquetting and gas plant, it would seem that at least in the western part of the state of North Dakota, a heating and lighting gas can be produced economically having a calorific power of about 400 B. T. U. per cu. ft.

For lighting purposes the gas produced from lignite by several methods used here in the University laboratories and at the Sub-Station has proved very satisfactory when burned with a mantle of the Welsbach type. The gas burns freely with a strong light and a clean flame.

Considering the ease with which the gas is produced, the low price of the original lignite and the value of the residue, this gas should have a large commercial value for heating, lighting, and power purposes, because of the low price for which it could be sold if manufactured in a plant used to produce briquets from the residue. It has been found that briquets made from this concentrated residue produce a most excellent fuel, for all practical purposes equal to anthracite. One ton of the lignite will produce about half a ton of these briquets in addition to 10,000 or 12,000 cu. ft. of gas. These briquets have about 12-13 the actual heating value of average anthracite and can be sold with a good profit at about the price of soft coal in the western part of North Dakota and lower than hard coal even in the eastern portion of the state.

These briquets give a product of such fuel value that they can readily be shipped for considerable distances and still prove profitable. The briquets present many advantages especially over the original lignite as usually placed upon the market. The heating value is nearly doubled, the briquets do not disintegrate on standing or burning, they can be stored without being affected by atmospheric conditions, they are uniform in size and convenient to handle.

No detailed statements of the cost of operating a large commercial plant are given here, for the reason that the cost per ton of briquets and per thousand feet of gas and other by-products will depend upon a large number of factors any one of which may materially affect the cost.

All the data which we have secured from our investigations and the operation of our experimental plant indicate that in a plant of fair capacity so constructed as to economize in the original cost, the cost of operation and the efficiency, if under careful management should turn out excellent commercial products at a selling price which would admit of a good profit when the products are sold far under ordinary market price. As already stated, gas can be sold at a very moderate price and the briquets could be furnished with profit to the manufacturer much below the price of soft coal in the western part of the state or hard coal in the eastern portion.

From the results obtained by the methods being worked out at the School of Mines and the Sub-Station of the University there seems little doubt but that the briquetting and the production of gas from lignite will in the future be put on a commercially satisfactory basis in this state. While this will prove of great value to all parts of our state, it will be especially important to those communities nearest the great lignite deposits in the western portion of the state, for in some of these the wastes can be converted into electricity which in turn can be sent to surrounding towns and villages, thus distributing power and light from numerous central power plants.

All of this will, of course, mean in the aggregate the saving of large sums of money for the citizens of the state and the introduction and development of a variety of manufacturing industries in an otherwise purely agricultural region.

According to a preliminary bulletin of the U. S. Bureau of Census the total value of the crude products of the cotton seed crushing industries of the United States increased 154 per cent during the last decade and aggregated \$107,538,000 in 1909 in contrast with \$42,412,000 in 1899. Cotton seed to the extent of 3,827,300 tons, costing \$78,112,000 in 1909, and 2,479,400 tons, costing \$28,633,000 in 1899, was needed for the manufacture of these crude products. The output of cottonseed oil amounted to 158,328,500 gallons, worth \$55,328,000 in 1909, and to 93,325,700 gallons, worth \$21,391,000 in 1899. The meal and cake totaled 1,674,500 tons, valued at \$40,493,000 in 1909, and 884,400 tons, valued at \$16,031,000 in 1899. Hulls weighing 1,269,200 tons, and worth \$7,711,000 in 1909, compared with 1,169,200 tons, worth \$3,189,000 in 1899, were also included among the products of the crushing mills; likewise 175,512,100 pounds of linters in 1909 and 57,272,000 pounds in 1899, having a value of \$4,006,000 and \$1,801,000 in these years, respectively.

SUGAR REFINING.*

By W. D. HORNE.

Just as sugar made from the cane appears first to have been used in the orient and to have followed a westerly path, so we find the art of refining had its origin in the east and progressively traveled westward. Although the sugar cane was in use in India as early as the fifth century B. C., we learn of refining having been practiced first about 500 A. D. in Mesopotamia. A large trade in refined sugar gradually developed between the orient and Europe and in 1470 a certain Venetian was awarded a government payment of 100,000 crowns for discovering a process of refining sugar, which was from that time extensively carried on in Venice. In the sixteenth century Antwerp led in the sugar trade in Europe and in the refining industry. England began refining sugar in 1544 and gradually gained the supremacy over the other European countries in the industry. With the development of refining in Europe, the price fell from two shillings per pound in the thirteenth century to four pence per pound in the sixteenth century. This undoubtedly was partly due to the general introduction of the sugar cane into the West Indies from 1510 to 1650, and the consequent enormous production of sugar and its ever growing commerce.

As early as 1689 there was a refinery in New York City, and by 1795, New York, Philadelphia and Boston refined about 1,200,000 lbs. of sugar yearly out of the 60,000,000 lbs. which was annually consumed, being about 2 per cent of the entire amount. Refined sugar was heavily protected by duty and sold at 20 cents per lb. By 1860 there were 41 refineries in the United States, producing \$42,000,000 worth of sugar annually. Now there are about half as many refineries, turning out about 3,000,000 tons of sugar annually, worth about \$300,000,000.

In 1747 Margraf succeeded in obtaining 6.2 per cent of sugar from beets and Achard established the first beet sugar factory in Austria in 1769. Napoleon, to encourage the industry on account of the difficulty in obtaining sugar because of the blockade of the French ports during the wars, heavily subsidized the industry in France about the beginning of the last century. It gradually spread over Europe. The first experiments in beet sugar manufacture in the United States were made in 1830 and again some beet sugar was made in California, Illinois and Wisconsin during the period from 1863 to 1876. Claus Spreckles built a factory in 1870 at Alvarado, Cal., which was the first to meet with pronounced success. The industry has developed considerably since then, until there are now 64 factories, located principally in the western states and Michigan, with a total output last year of 457,000 tons, being 13.02 per cent of the consumption of the country. By 1900 the production of beet sugar had so

increased that it had grown to be 60 per cent of the world's sugar supply.

In 1910-11 the world's sugar supply will be 17,000,000 tons—50 per cent cane and 50 per cent beet. Sugar is produced also in small quantities from the sap of the date palm in some eastern countries. Prolonged attempts were made in this country some 15 or 20 years ago to produce sugar from sorghum, but without success, as the dextrinous bodies in the sorghum juice prevent large quantities of the sucrose from crystallizing, and this plant is now used exclusively for the manufacture of syrup. The annual production of maple sugar in this country amounts to 6,000 tons, but as maple sugar owes its principal value to its pleasant flavor, rather than to its sweetness, it is always sold in its raw state, as refining it would rob it of half its present value and render it impossible of competition with cane or beet sugar. Milk sugar is produced to some extent and used, always refined, for medical purposes principally.

Sugar refining consists essentially in the purification of the crude crystalline material through the well-known process of re-crystallization. To this we can add the second process of absorption of impurities by means of bone-black, first introduced by Derosne in 1812. By this means is effected a direct separation of impurities from the sugar solution and a large removal of the coloring matters, allowing of the more easy production of perfectly white crystals. While the process is relatively a simple one, and perhaps more mechanical than chemical in its nature, the enormous consumption of sugar in the country and keen competition in manufacture have led to the establishment of very large and elaborately equipped plants for the conduct of operations where every economy of procedure is carefully observed. The great amount of capital involved in erection and maintenance of such plants, together with the highly developed technique required for their operation has tended to keep the business in relatively few hands and to render it a dangerous enterprise for any but the most thoroughly equipped to venture upon. The sugar refineries of the country are in a few spots—New York, Philadelphia, Boston, New Orleans, San Francisco and Baltimore containing practically all of them. In 1910 the country consumed 3,350,355 long tons of sugar, practically all being refined. Of this amount 333,006 tons were cane sugar raised in Louisiana and neighboring states, 457,000 tons were beet sugar raised and refined during the process of manufacture in our western factories, 2,472,756 tons came from our insular possessions and Cuba, and 72,393 tons came from other foreign countries. We imported practically no beet sugar and we exported practically no sugar at all. With the exception of Great Britain, the United States leads the world in the consumption per capita of sugar, this being 81.6 lbs. per person. Most of this goes into direct use for household purposes and yet enormous quantities are consumed by manufacturers of confec-

tions, preserves, condensed milk, as well as by canners of fruit, ice cream makers, soda water fountains and many other users.

As will be seen, the natural location for a sugar refinery is on the water front where it can more economically receive and unload the great cargoes of sugar and of coal which come to it almost daily, for a modern refinery melts from a million to three or four million pounds of sugar per day. The economical handling of this vast amount of raw material and the successful guidance of so much organic matter in solution at high temperatures through long and complicated processes, without fermentation, decomposition, discoloring or waste is no mean accomplishment; but as in every other instance of manufacture the whole matter has been worked into a routine, which once properly installed, almost takes care of itself. As the sugar, whether in solution or later in granular form, has to pass through many operations in a continuous stream as it were, it is thus found advantageous to have the units of a sugar refinery many stories high, so as to take advantage of gravity to pass these solutions from place to place or in delivering the dryer material from one department to another. This holds true equally of the bone-black, and we find the char house is usually a tall building in close proximity to the sugar house itself. In many instances the boiler house also has its coal bunkers in the upper story, allowing the fuel to feed by gravitation into the fire room or even upon the grates themselves. The high cost of real estate in cities also makes it advisable to extend upward rather than laterally. Each sugar refining plant will be found to have a large dock department for the receiving, weighing and storage of raw sugar, a boiler house for the generation of steam for power and evaporation, a wash house for the so-called washing of raw sugars, the pan house for boiling, granulating and other processes of manufacture of dry sugars, a char house for the accommodation of the bone-black filters, kilns for the re-burning of the char, etc., and usually a warehouse for the storage and shipment of refined products. These departments are not always built separately and frequently more than one of them will be found in the same building.

Somewhat different processes are used for the refining of beet and of cane sugar, but the difference consists principally in the method of defecation or clarifying the original raw sugar solution of its suspended impurities and part of its coloring matter. In either case the first operation is the unloading, weighing and storage of the raw sugar, whether in bags, baskets, mats, hogsheads or in other packages, in proximity to the melting department. A portion of the sugar is usually taken direct from the scales to the wash house, where the sugar is raised by a bucket elevator to an upper story and mixed with a low grade syrup while passing through a conveyor to the raw sugar mixer. The magma of raw sugar and syrup is fed from the mixing tank containing revolving arms

into centrifugal machines, which purge the syrup from the sugar in a few minutes. Water sprinkled on affects the final washing, giving a sugar of high purity and a low syrup. This sugar is dissolved in a round melting tank with revolving arms in hot water to a density of about 30 Beaumé, pumped into blow-ups, treated with a very small amount of acid calcium phosphate and made slightly alkaline with milk of lime. Such a solution would boil at about 217.5° F. and its temperature is next raised to something under 200° F. whereby tricalcic phosphate is precipitated, entangling the suspended impurities and also removing, possibly by precipitation, some of the coloring matter. The lime has the effect of precipitating some of the gums and the heat at the relatively slight alkalinity precipitates most of the albuminoids. This process is applicable to cane or beet sugar or to a mixture, but in defecating beet sugar alone it is sometimes customary to add a few tenths of a per cent of caustic lime, precipitating this with carbon dioxide which precipitates suspended matters and largely removes the color. In either case the solution must be mechanically filtered. This is commonly done through cotton twill bags enclosed in woven sheaths. These are called Taylor filters. In the case of beet solutions the filter press is sometimes used, but this is not applicable to cane sugars which are gummy and sticky and clot the filter press. Beet sugar solutions are sometimes treated with sulphur dioxide after this preliminary alkaline defecation and again filtered in presses in a very faintly alkaline or neutral condition. The filter bags after becoming nearly exhausted are allowed to drain during several hours, are washed with hot water several times allowing to drain between times, and are removed from the closed iron tanks in which they hang, for washing. They are turned inside out and rinsed in several waters, passing through wringers between them. The muddy water thus obtained still contains some sugar and is pumped through filter presses, the sugar of the clear filtrate being recovered. The press cake, containing very little sugar, is discarded.

The clarified solutions are next filtered through bone-black, contained in cylindrical filters or cisterns. The liquor gives up the greater part of its color and a less per cent of its ash and organic impurities to the bone-black and is collected in storage tanks according to color and purity. Lower grade solutions of greater color follow the washed sugar solution on the char, so that the last filtrate from the char is pretty dark in color and much lower in purity.

The char filtered liquors pass to the vacuum pan, holding about 1,000 to 2,000 cu. ft., where it is boiled to grain and concentrated to a low water content.

This magma is dropped into the mixers or crystallizers, from which it passes to the centrifugal machines, where it is purged and washed with a spray of clear water, sometimes followed by a spray of blue water, formerly colored by ultramarine, which of late

years has been replaced by harmless aniline colors prescribed by the government, since the pure food law put its ban on ultramarine as a mineral substance, not to be allowed in sugar products. The syrups can again be boiled in the vacuum pan to produce granulated sugar, and when the impurities and color accumulate enough the syrups are used for yellow sugars.

The moist sugar from the centrifugal machines, containing a small per cent of water, is passed through nearly horizontal revolving drums containing longitudinal shelves projecting inward which have the effect of picking up the sugar, and sprinkling it through the current of warm air which is drawn through in the opposite direction.

The sugar thus dried, next passes through revolving screens which separate it into different grades according to the size of the crystals, giving rise to granulated sugar, to be bagged or barreled.

In making cube sugar, some of this moist granulated sugar from the centrifugal is pressed into cubical blocks by an ingenious machine, and gently dried in ovens during a few hours. Cut cube sugar was originally made by draining the magma or boiled mass of sugar in conical moulds for about two weeks, with occasional washing by means of pure sugar solution, sawing the dried cones into discs and cutting these across into cubes. A modified process of this kind is in use in Europe and to some slight extent in this country, in which the conical moulds are supplanted by rectangular frames, which are purged in centrifugal machines in minutes instead of weeks.

Yellow sugars are made from low testing syrups, which are boiled in a vacuum pan and contain smaller softer crystals than higher grades of sugar, with considerable amounts of adherent mother-liquor.

To return to the bone-black, this is washed down by hot water after the last sugar solution sinks below its surface and is thus freed from the sugar. Certain mixing of the water with the sugar solution is inevitable, giving rise to a zone of sweet water. As the water begins immediately to dissolve the impurities which the char had just absorbed from the impure sugar solution passed through it, this sweet water contains impurities along with the sugar and has to be separated from the main filtrate and treated by itself. The sugar washes out faster than the impurities, however, and when most of the sugar has been removed the stream of outgoing water is turned to the sewer, and continued until the char is pretty nearly free of what water can remove. The bone-black is then drained, emptied from the filter by gravity upon a dryer on the floor beneath, through which it passes to the kilns. These are furnaces provided with internal iron pipes, through which the char passes. By proper regulation of the flow of the char through these retorts its temperature is kept at a point which subjects the impurities still remaining in it to destructive distillation. Moisture, ammonia, carbon dioxide and other gases are given off,

some of which are highly inflammable, leaving in the char a small residual amount of carbon. The char after cooling is then ready for use again and this entire cycle of operations is repeated.

When the carbon accumulates to such an extent as to choke the pores of the bone-black, the decolorizing power of the char diminishes, so that finally it must be discarded. Of late years this trouble has been obviated by the introduction of the Weinrich system of de-carbonization in which the char is passed through a roaster very much like the granulator before described and heated on the outside by direct fire to a suitable temperature. The air admitted suffices to burn off the impurities without attacking the carbon of the char itself in any marked degree. By this means char can be kept at a high point of efficiency for a considerably longer time.

The large amount of heat passing away from the char filters in the hot wash water is recovered by passing this water through tubulated heat economizers in which the incoming city water passes in the opposite direction, absorbing the heat of the outgoing water.

Water for sugar refining should be as soft as possible and as free from sulphates, iron, color and suspended matter as can be.

The sweet-waters from the char are evaporated in multiple effect evaporators and usually mixed with other low grade solutions.

Fuel is an important matter in sugar refining, large quantities being used to generate steam for power purposes, evaporation and heating. This in large part explains why sugar is not refined where the raw product is made, the fuel cost being prohibitive.

The laboratory organization in a sugar refinery is a matter of great importance. It is the laboratory's function to keep track of the quality, not only of the raw and finished material but also of the material in process of refining, not only to show the efficiency of each process, but to enable the most economical and expedient combination of the various liquors, syrups and other solutions throughout the process of refining so as to prevent unnecessary work and to indicate the most advisable method of handling. All raw sugars have to be analyzed in considerable detail, a close watch is kept on fuel, variations in the water supply must be noted, finished products must be kept up to the mark and waste products must be carefully checked up to avoid excessive loss. New processes have to be devised to meet varying condition and experimental work must always be pursued to keep up with the latest improvements and discoveries in the art.

The routine must go on day and night without cessation and all chemicals, apparatus and other appliances must be properly standardized and those standards maintained. Analytical processes have to be investigated and tests devised for new needs. Many materials entering into use in a large factory have also to be subjected to occasional tests or analysis and com-

petent advice has to be given as to the desirability of new methods or processes, suggested from whatever source.

All this calls for an elaborate, competent and smooth running chemical organization. The routine tests, principally of purities, are conducted by young men carefully trained in this work and provided with every possible facility for its accurate and speedy accomplishment. In the laboratory many tests are made on char samples and wash water daily, all raw materials and waste products, all miscellaneous sugar samples, as those involved in stock taking, and so on. The laboratory also keeps careful statistical records of all its various work and compiles elaborate monthly and yearly statements covering all these tests. The laboratory should be provided further with a good general chemical library and ample files of sugar journals as well as those of more general nature. A good laboratory should lead the work of the house, rather than follow it, by showing the way over difficulties that are from time to time bound to spring up; by inventing and elaborating new processes of more economical work; by keeping in touch with the outside world, and in general by assisting a smooth running of the entire enterprise.

FACTORS IN THE FORMATION OF PRODUCER GAS.

"Essential Factors in the Formation of Producer Gas" is the title of Bulletin No. 7, recently issued by the U. S. Bureau of Mines. The authors, J. K. Clement, L. H. Adams and C. N. Haskins detail an investigation of the chemical and physical processes that take place in the gas-producer in which they kept in view not only the possibility of increasing the efficiency of the producer as a source of energy and the ensuing benefits to the public of cheaper power and greater utilization of low grade fuels, but also the application of the results to the problems of boiler-furnace operations.

In experiments made by one of the authors at the Bureau's Experiment Station it has found that the temperature in the fuel bed of the gas-producer varied greatly in different parts of the bed. In order to ascertain the conditions of temperature most favorable to the efficient operation of the producer, it became necessary to determine the temperature required for the formation of carbon monoxide and hydrogen.

Among other reasons for investigating the conditions for the reduction of carbon dioxide by carbon was that a small quantity of carbon monoxide is invariably contained in the flue gases of boiler furnaces and it was hoped that means might be suggested of preventing its formation and the resulting loss in furnace efficiency.

These investigations demonstrated that a very high temperature is necessary for the production of carbon monoxide from carbon dioxide and carbon. These conditions, however, which argue against operating the

fuel bed and the gas-producer at extremely high temperatures—above $1,300^{\circ}$ C.

A very hot fuel bed means that the gases will leave the producer at a high temperature, and thus lower the efficiency of the producer. The gain in capacity would, therefore, be accompanied by a loss in efficiency, unless the heat of the gases could be used for generating steam or preheating the air blast. A high temperature also favors clinkering. In the application of the results of these experiments to commercial producers and furnaces it will be necessary, of course, to consider the other questions which are involved.

Various explanations have been suggested to account for the presence of small amounts of carbon monoxide in the flue gases of boiler furnaces. Perhaps the one generally accepted by engineers is that the oxygen of the air first unites with carbon to form carbon dioxide, and that as this gas passes up through the bed it combines with carbon in accordance with the equation:



Assuming this to be the correct explanation, then the questions to be solved are what conditions favor this reaction and what conditions will tend to retard it. It has been shown by the authors that the higher the velocity of the gas and the thinner the fuel bed the less will be the percentage of carbon monoxide formed. A heavy fuel bed in the boiler furnace would therefore favor the formation of carbon monoxide. Also, the greater the supply of air to a given depth of bed the less would be the percentage of this gas formed; therefore with a hot fuel bed the formation of a small amount of carbon monoxide is inevitable. In order that this carbon monoxide may be burned to carbon dioxide in some way sufficient air must be added to the hot gases as they leave the top of the fuel bed.

The bulletin also contains a chapter by J. K. Clement and L. H. Adams on "Effective Temperatures for Water-Gas Generation."

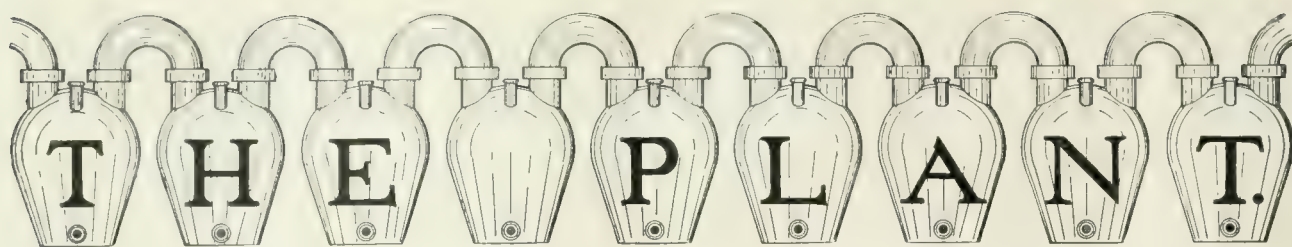
The results presented show that a high rate of gasification combined with a high percentage of carbon monoxide and a low percentage of carbon dioxide and water requires a high temperature in the fuel bed.

The higher the temperature the better will be the quality of the gas and the greater the capacity of the producer.

The use of large amounts of steam is inconsistent with the realization of high temperature, and is, therefore, to be avoided.

Although these investigations were undertaken primarily to determine the conditions governing the formation of producer-gas, the results have an important bearing on the water-gas process.

They show that although with very low rates of steam supply the decomposition of the steam may be complete at $1,100^{\circ}$ C., with higher rates of steam supply, such as are desirable in practice, a much higher temperature, $1,300^{\circ}$ or $1,400^{\circ}$ C., is required to obtain complete decomposition.



TURBINE DRIVEN AIR COMPRESSOR FOR BLAST FURNACE BLOWING.*

By **RICHARD H. RICE.†**

The General Electric Co. recently put in operation at the Oxford Furnace, N. J., plant of the Empire Iron & Steel Company, a turbine driven air compressor for blowing the blast furnace, which is the

a direct connected four-stage Curtis steam turbine. The design is such that this normal speed produces a blast pressure of 15 lbs. per sq. in. The unit is, however, designed to regulate the volume of air delivered per minute so as to keep the rate of discharge constant at any value, determined by the furnace superintendent, within its capacity. The manner in which this is accomplished will be fully described in the

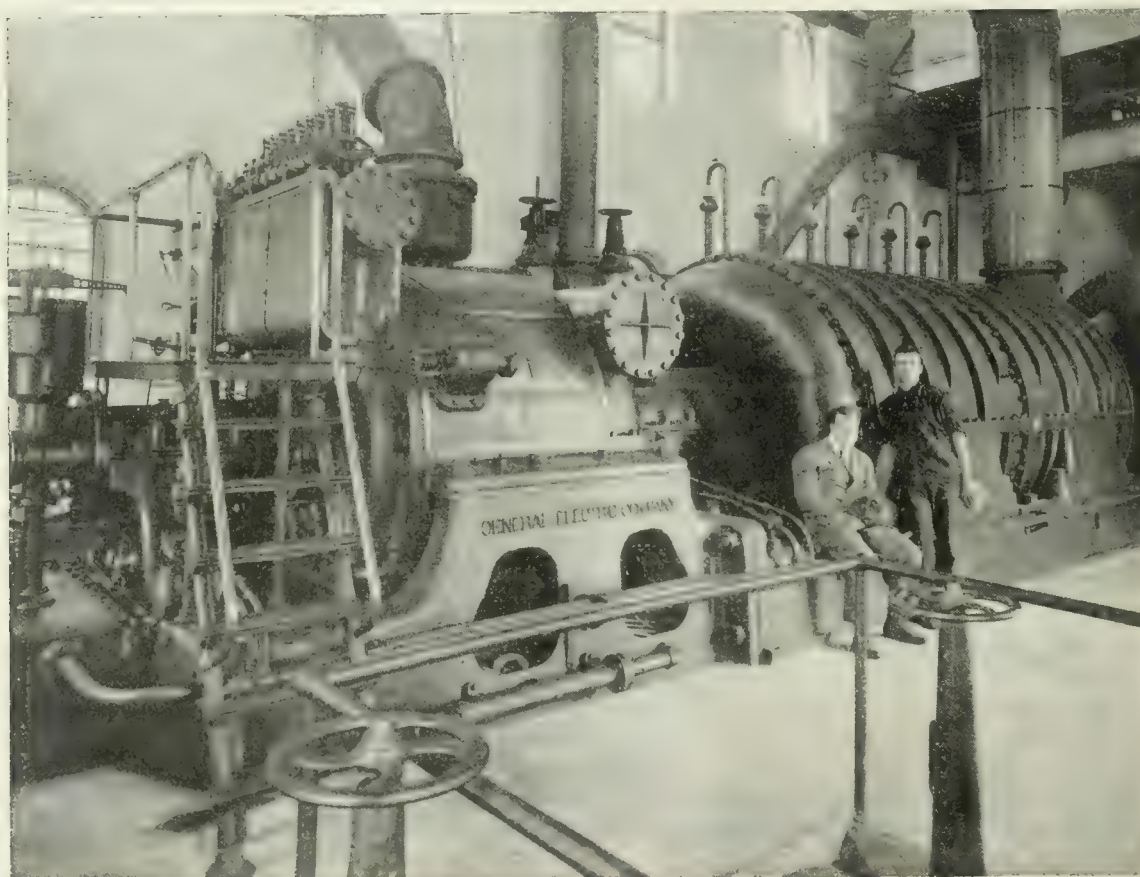


Fig. 1—Six-Stage Compressor Driven by Direct Connected Four-Stage Curtis Steam Turbine.

first installation of this type of apparatus to be made in this country. Some particulars concerning the features of the installation will no doubt be of interest.

The unit consists of a six-stage compressor operating at a normal speed of 1,650 r. p. m., and driven by

*Paper prepared for the American Society of Mechanical Engineers.

†Consulting Engineer, and head of the turbine shops of the General Electric Co.'s works at Lynn, Mass.

sequel—but it may be said here, that the regulation is by means of speed variations, so that the machine is a constant volume, variable speed apparatus, and not constant speed as in other classes of blowing units.

The six stages of which the compressor is composed are arranged in series so that the air enters at the end of the compressor nearest the steam turbine driver and passes successively from stage to stage until it reaches the other end of the compressor casing, where

it enters the discharge pipe. The impeller wheels are so designed that there is no unbalanced end thrust, so that the ordinary means used in the Curtis turbine for locating the rotating elements and preserving proper clearances are sufficient for the entire apparatus.

fore, the efficiency of compression must remain constant.

Fortunately, both turbine and compressor attain their best efficiency under similar conditions as regards rotating speed. Therefore the combination is a logical

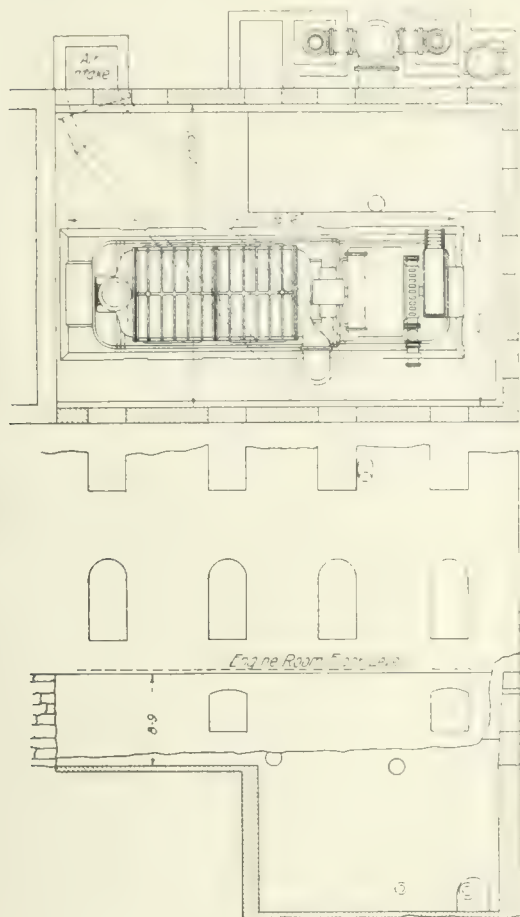


Fig. 2—Plan and Elevation of Installation at Oxford Plant of Empire Iron & Steel Co.

The air is cooled during compression in each stage, and also in passing between stages by suitable water chambers in the diaphragms, and this cooling is suffi-

cient to maintain the compression approximately along the adiabatic line.

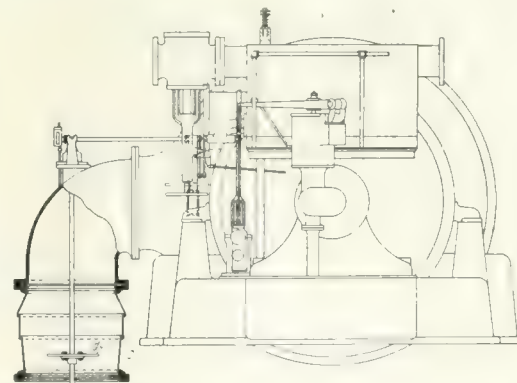
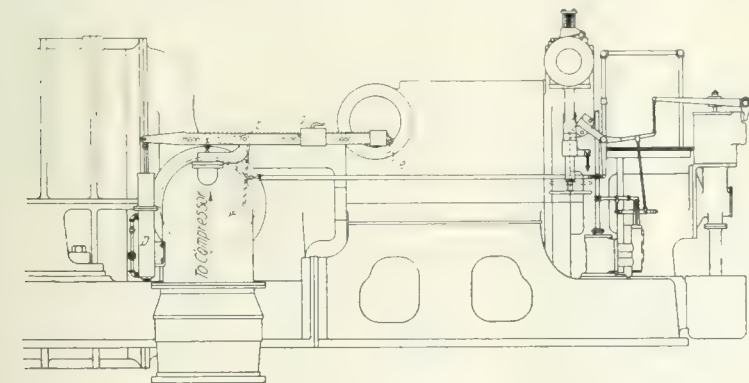


Fig. 3—Details of Constant Volume Governor for Turbo-Condenser.

cient to maintain the compression approximately along the adiabatic line.

No valves or rubbing surfaces are used in the compressor construction, and, as in the turbine, the rotating elements revolve freely with ample clearance so that no wear or deterioration can take place; there-

fore, the efficiency remains sensibly the same. A curve of efficiency at varying volumes is shown in Fig. 4, and above this has been drawn a curve of speeds and pressures which, taken in connection with the first named, shows the variations of efficiency with pressure at rated volume.

This latter curve shows graphically the variation of pressure with change of speed; which follows the law of squares; that is, double speed gives four times pressure, etc.; from which it will be seen that only moderate changes in speed are necessary to give considerable changes in pressure. It is these changes in

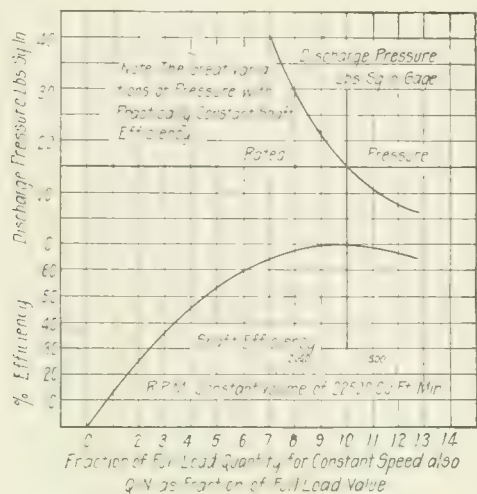


Fig. 4—Efficiency and Pressure Curves.

speed, increasing or decreasing the blast pressure, which are utilized to maintain a constant rate of flow of air into the furnace, against the varying resistances set up in the tuyeres and furnace by varying furnace conditions; as, for instance, clogging of tuyeres and changes in the size and composition of the charge, temperatures, etc.

The means by which these changes of speed are produced in the manner necessary to keep up a constant rate of influx of air per minute is shown in Fig. 3. This shows a steel disc sustained on the inflowing air current within a conical enlargement of the inlet pipe. By sliding weight, *a*, the resistance of this float and displacement by the air current is adjusted in accordance with a scale in the scale beam, *b*, which scale is graduated accurately to read, in cubic feet per minute, volumes of free or atmospheric air. By setting this weight at the graduation corresponding to the rate of discharge of air desired, the disc is caused to assume a position in the conical enlargement, *c*, which results in supplying steam to the turbine in quantity sufficient to establish the proper speed of the compressor and pressure of blast to cause the required flow of air through the furnace. In case the rate of air flow tends to decrease, the disc, *d*, sinks to a lower point in the enlargement, *c*, since the supporting air current decreases its sustaining power; more steam is by this admitted to the turbine and the speed is increased, resulting in increase of pressure, and this increased pressure re-establishes the desired flow of air. In case too much air tends to flow into the furnace, the reverse of all these effects takes place. In practice, the operation of this device is most regular and satisfactory.

This method of governing by the indications of a properly calibrated scale beam gives an entirely new instrument, which, in the hands of a skilled furnace manager, will undoubtedly enable improved results in furnace operation to be obtained, since an accurate knowledge of the amount of air supply is always at hand by this means.

Such accurate knowledge of the amount of air supply cannot be obtained from reciprocating blowing engines; since the expansion of air in clearance spaces causes an error increasing in amount as discharge pressure increases; also since any leakage increases with increase of discharge pressure; also because the slip is a variable and uncertain amount. On the contrary the air governor is unvarying in its action, and will not change its indications with time, since wear and leakage are absent.

It has been before intimated that this is a variable speed apparatus. In normal blast furnace operation, pressures may vary from 10 to 20 lbs. per sq. in. These pressures require speeds in the particular apparatus under description of about 1,500 for 10 lbs. pressure to 1,800 for 20 lbs., as appears on the curve, Fig. 4. The blast furnace operator therefore instructs the engineer operating the compressor not to maintain a certain number of revolutions, as is customary with reciprocating engines, but to set the scale beam weight for the required volume in cubic feet per minute.

The graduation and calibration of the scale beam in cubic feet per minute is determined during the shop test of the apparatus before shipment by accurate tests with standard orifices and pitot tubes and these graduations are accurate within about 2 per cent.

A simple oil dashport, *D*, Fig. 3, attached to the scale beam prevents any racing or undue fluctuations of speed.

In operating the blowing unit, it is only necessary to manipulate the hand throttle valve in the main steam

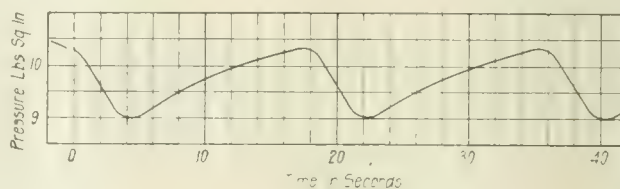


Fig. 5—Pressure Curves During Pulsations.

pipe when it is desired to bring the compressor to rest. At all other times, control is effected through the scale beam, with wide open throttle. At times of checking the furnace, or casting, the weight, *a*, Fig. 3, is moved to the extreme end of the scale beam at the position indicating the minimum volume for which the scale beam, *b*, is graduated, and still further decrease of speed and pressure is produced by adding an auxiliary weight at this end of the beam or by depressing it by hand. On removal of the auxiliary weight and replacement of the sliding weight, *a*, at the running volume graduation, the compressor speeds up until the

volume required is obtained. This manipulation is in practice of the simplest character.

The air governor acts upon the pilot valve of the hydraulic valve gear commonly used on the larger sizes of the Curtis turbine, through a system of floating levers; in such wise that when the turbo-compressor nears the maximum speed for which it is designed, in this case 1,950 r. p. m., a centrifugal governor of the usual type comes into action and keeps the

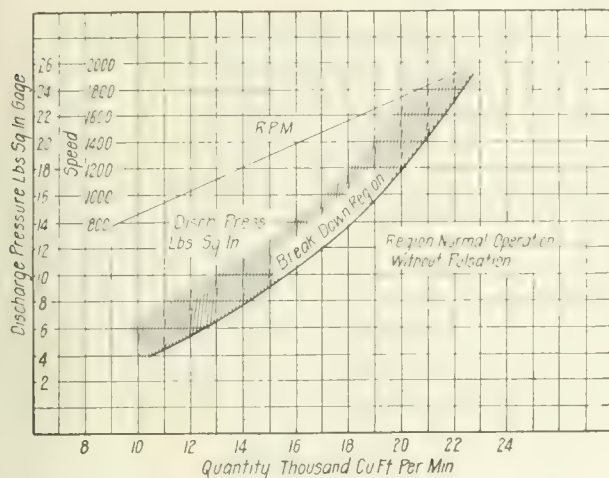


Fig. 6—Critical Pressure Curve.

speed at this maximum so long as the resistance to air flow in furnace or tuyeres remains so high that the volume of air, for which the air governor is set, cannot be forced through at the maximum pressure to which this speed corresponds, in this case 25 lbs. per sq. in. During this period the air governor is out of control of speed, but it comes into action immediately the furnace resistance decreases.

In case of breakage or sticking of the governor mechanism which permits the speed to exceed 1,950 r. p. m., an emergency governor mechanism, entirely independent of the mechanism previously described, comes into play and closes the main throttle valve, bringing the compressor to rest.

In all high speed apparatus the certainty of the oil supply is an important feature, and it is particularly so in this service. In this unit there are three shaft bearings requiring automatic lubrication, and this is furnished by a valveless gear pump, worm driven from the main shaft, which circulates oil under 15 to 25 lbs. pressure. The same pump also supplies this necessary oil to the hydraulic cylinder which actuates the valve gear. The oil is returned from bearings and cylinder to a tank where it is settled and strained before re-use. In order to guard against any stoppage of oil circulation, an alarm is provided which causes a steam whistle to blow in case the oil pressure falls to 5 lbs. per sq. in.

The oil is cooled in the bearings at the point where the heat is generated by means of water cooled coils embedded in the bearing linings.

The apparatus described uses, of course, high pressure steam; obviously the compressor is equally adapt-

ed to the use of low pressure turbines as drivers, and so driven, affords a ready means of increasing the efficiency of existing plants containing reciprocating blowing engines, by the usual method of exhausting from the reciprocating steam cylinders into the low pressure turbine. The governing by volume of air discharged is equally applicable here, and all the advantages of this system can therefore be realized. Increased efficiency of the plant to the extent of 20 per cent to 50 per cent may be thus realized with a very moderate addition to the cost of the plant.

The installation at the Empire Steel & Iron Co. was put in operation on the furnace on March 8, 1910, and has been in steady continuous operation ever since. At the time this apparatus was put in operation it was not expected that the volume of air required by the furnace would be at such a low figure as turned out to be the case, the machine having been designed for a normal volume of 22,300 cu. ft. per minute. On putting the machine on the furnace, it was found the volume required was only about 15,000 cu. ft. per minute and the pressure corresponding to this volume under furnace conditions ranged from 10 to 12 lbs. Under these conditions, it was found that pulsations were met with in the pressure line, this pressure fluctuating about 2 lbs., and in order to overcome this pulsation, it was found necessary to throttle the inlet opening. Fig. 5 shows the character and magnitude of these pulsations. Since this time, a convenient butterfly valve throttling mechanism has been

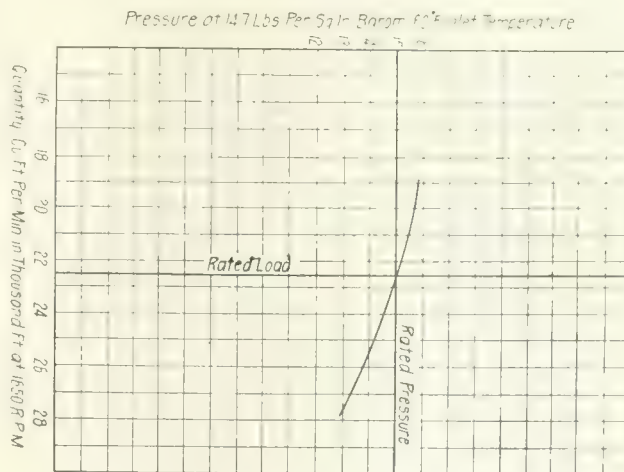


Fig. 7—Pressure Volume Curve at Constant Speed.

designed and applied, which is found to eliminate these pulsations without appreciable loss of efficiency.

The pulsations in pressure above noted are an inherent characteristic of all centrifugal blowing apparatus of similar construction, and they occur when the apparatus is operated at loads and pressures widely differing from those for which the apparatus is designed, that is from normal full volume and pressure. At any given volume they occur at a certain critical pressure and at all higher pressures, but do not occur at lower pressures than the critical. As volume is increased, critical pressure increases also. The crit-

ical pressure is slightly affected by the density of the air and the humidity. Fig. 6 gives the characteristic critical pressure volume curve of this compressor. The rate and extent of the pulsations are affected by the capacity of the discharge piping, stoves, etc., into which the air flows. The larger the capacity the longer the period or wave length and the greater the wave magnitude and vice versa. The diagram, Fig. 5, shows the pressure waves from the machine installed at Oxford Furnace. When tested in the shop with very short piping of small capacity, the wave length was only a second or so, and of very small height.

The pulsations only occur at such loads that the characteristic pressure curve of the apparatus is rising with increase of volume or remains horizontal and the effect of the throttling is to superpose a drooping pressure curve, falling with increasing volume, which alters the shape of the resultant pressure curve and makes it droop also. As the throttling required to entirely remove such pulsations is only a few inches of water, it has no appreciable effect on the efficiency of compression.

Fig. 7 is the curve of pressure and volumes for this compressor at constant speed.

At the time this was written the blast pressure at Oxford Furnace varied from 10 to 14 lbs. during the day with volume constant at 16,000 cu. ft. per min. The speed varied from 1,500 to 1,600 r. p. m. The steam pressure averaged 135 lbs.

The figures given in Table I are taken from a typical station log, showing the variation of pressure and volume during a 24-hour period of operation:

TABLE I—ENGINE ROOM REPORT, EMPIRE STEEL AND IRON CO., OXFORD FURNACE, N. J., MARCH 17, 1910.

Time.	Volume, Blast		Steam			Remarks.
	Cu. Ft.	Pressure.	R P. M.	Pressure.	Vacuum.	
1 a. m.	15,750	13 lbs.	1,540	140	24 ins.	
2 a. m.	15,750	12.5	1,490	135	25	
3 a. m.	15,750	13.5	1,580	135	25	
4 a. m.	15,750	12	1,510	140	21	
5 a. m.	15,750	13.5	1,530	155	24	
6 a. m.	15,750	13	1,550	150	25	
7 a. m.	15,750	12.5	1,550	150	25	
8 a. m.	15,750	12	1,490	120	hp.	
9 a. m.	15,750	11.5	1,500	130	hp.	
10 a. m.	15,750	13.5	1,580	160	26	
11 a. m.	15,750	12.5	1,520	155	26	
12 m.	15,750	12.5	1,500	150	26	
1 p. m.	15,750	13	1,530	140	26	
2 p. m.	15,750	12	1,490	150	26	
3 p. m.	15,750	13	1,560	130	26	
4 p. m.	15,750	13	1,560	150	26	
5 p. m.	15,750	11	1,445	145	26	
6 p. m.	15,750	11.5	1,450	130	26	
7 p. m.	15,750	11	1,490	135	26	
8 p. m.	15,750	13	1,510	140	26	
9 p. m.	15,750	12.25	1,500	155	25.5	
10 p. m.	15,750	11	1,383	140	25	
11 p. m.	15,750	11.5	1,410	150	25.5	
12 p. m.	15,750	11.5	1,440	145	25	

The apparatus used for blowing the furnace before putting this machine into operation consisted of two vertical reciprocating blowing engines built by the I. P. Morris Co., each of the following dimensions:

Steam cylinder diameter, 54 in.; blowing cylinder

diameter, 72 ins.; stroke, 72 ins. Blowing cylinder displacement, 339 cu. ft. per rev. each. Maximum speed rating, 30 r. p. m. each, giving 20,300 cu. ft. per min. total displacement. Actual maximum speed, 23 r. p. m. each giving 15,000 cu. ft. per min. total displacement. The average blast pressure was 8 lbs.

Judging from the revolutions of this engine, it was thought that the volume used was about 14,500 cu. ft. On putting the centrifugal compressor into action, an immediate increase in the amount of iron melted by the furnace, was experienced. The output went up from an average of 139 tons per 24 hours in February, 1910, to 176 tons in April, 1910, and the iron was found to be of a more uniform character, and the operation of the furnace was improved. A gradual increase in the amount of air has since taken place and the corresponding increase in pressure required to force the air through the furnace has been necessary as was to be expected. This increase of air has resulted in an increase in the production of the furnace from 176 tons, on starting to the present average of about 190 tons. The machine is now operating with 16,000 cu. ft. of air and the production of ore is 185 tons per 24 hours' average. It is proposed to continue this increase to 200 tons per 24 hours, the limit of the charging apparatus. The dimensions of the furnace are as follows: Diameter at bosh, 17 ft. 6 ins.; at hearth, 11 ft.; at top throat, 12 ft.; height from hearth to dumping ring, 80 ft.

The condensing apparatus is of the barometric type, and the injection water is supplied by a turbo driven centrifugal pump, placed in the sub-basement; and at the outset when the machine was first put in operation, difficulty was had with the condensing water supply, which made it necessary to operate the machine for a considerable period of time non-condensing. Owing to the unfamiliarity of the fireroom force with the new boilers which had been installed it was even necessary to operate with steam pressures as low as 60 lbs. per sq. in. gauge for various periods, under which conditions, the compressor set operated with entire satisfaction.

Owing to the fact that the condensing apparatus is of the barometric type, the further fact that the machine is operating far below its designed capacity and the difficulties involved in making an accurate boiler test to determine the amount of feed water under present conditions, no tests have been made to determine the actual efficiency of the machine. The machine is, however, furnishing considerably more air than the old machines, as is evidenced by the greatly increased product of the furnace, and at the same time it is operating with less boilers, and these boilers are more easily worked than when operating with the engine.

There is great difficulty in making comparisons of the performance of this type of blowing unit with reciprocating types either steam or gas driven, owing

to the absence of actual test figures, none of which have been published which permit of accurate and satisfactory comparison. From the results which have been obtained from all sources as to the actual performance of such machines and from actual experience with this machine and the sister machine, installed at the Northern Iron Co., in line with tests which have been made in the factory, it seems that the following conclusions are correct in reference to this apparatus as compared with reciprocating engines for blowing blast furnaces:

That the output of the furnace is increased on account of the greater steadiness of operation and more uniform conditions obtaining in the furnace.

That the quality of the product is improved.

That the steam consumption is equal to, or less than, that of the best compound engines blowing similar furnaces.

That the engine room space occupied is only a fraction of that needed by reciprocating engines, either steam or gas.

Considering all factors, including consumption of fuel; cost of operation, including oil and supplies, attendance, etc.; cost of buildings and foundations, interest on the investment; and cost of maintenance of plant, that of the centrifugal compressor is a blowing apparatus which can be operated for a lower net cost than any other means of blowing furnaces.

The market for amyl acetate and refined French oil is in a state of great unrest. The large manufacturers in this country are really not quoting on the materials. The real cause of the high price, which has steadily advanced during the past month from \$1.80 to \$2 per gallon, is not at once apparent. The producers of amyl acetate and fusel oil in England report a shortage of raw material, and consequently, with increased consumption of the refined products, a higher price can readily be obtained. Whether this is correct or not we do not know, but we do know that the present prices are higher than ever before, and that continued high prices for these two materials will not fail to have a disastrous effect on the lacquer manufacturing business. Lacquer, as is probably well known, is made by dissolving various gums and gun cotton in either fusel oil, amyl acetate or both, and these lacquers play a very important part in the brass business. The high price of lacquer, then, will cause considerable trouble in the manufacture of brass products, especially among the small manufacturers. Does it not seem as though the time had come for the industrial chemist to "get busy" and give us a cheap and abundant substitute for fusel oil and amyl acetate?—*The Metal Industry*.

According to the computation of the Minister of Finance for the Union of South Africa, the gold-mining industry of the Rand still has a life of 150 years before it.

THE ELIMINATION OF SOME SOURCES OF LOSS IN A LARGE PRODUCER-GAS ENGINE PLANT.*

By JOHN G. CALLAN.†

A short time since the writer, with some associates, had occasion to test a producer-gas engine plant which at the time was the largest in this country. The test was run to determine whether the plant met guarantees as to average and maximum output and fuel efficiency, and every effort was made to place the machinery in the best of condition and to obtain the best possible performance. It is not the purpose of this paper to report the entire test, which occupied the greater part of a month, but rather to point out some specific items wherein it was possible to improve performance and to briefly analyze the more important of these.

Since in spite of the best endeavors it was not possible to bring the apparatus up to guarantee, we omit out of courtesy to the manufacturers any specific references by which the plant would be immediately identified. None of the points of interest lose anything of value by this course.

The installation consisted of four single tandem, double acting 33x48 producer gas engines driving 810 kw. three phase 25 cycle 440 volt 28 pole generators at 107 revolutions per minute. The generators operated in parallel delivered power through a suitable switchboard to an industrial plant requiring substantially steady load, and to some other minor users. Power factor was about 80 per cent. Excitation was by motor generator, with a gas engine originally installed for this purpose held as a reserve.

Gas was furnished by three down-draft producers of a well-known type. From their individual wet scrubbers the gas passed through a common dry scrubber to a 30,000-cu. ft. holder and thence to the distributing main leading to the engines.

The fuel was lignite of about 7,600 B. T. U., carrying about 34 per cent of moisture and 8 per cent ash. The gas averaged about 105 B. T. U., using the high thermal value.

One of the engines had been wrecked shortly before the test and operation was upon the other three. The mill load was adjusted to suit engine output and there was always available as much load as the engines could carry. Indeed, the power plant was the limiting feature of the mill, and its failure to deliver rated power caused a direct money loss amounting to approximately 50 cents per day per kw. that the plant was in default.

The plant had been installed by its manufacturer, and shortly after its installation the designer of the engines had spent the greater part of his time for some three months in bringing it up to the best possible per-

*Electrical Engineer. Arthur D. Little, Inc., Boston, Mass.
†Paper presented before the Congress of Technology at the 50th Anniversary of the Granting of the Charter of the Massachusetts Institute of Technology.

formance. During this period the average output had been very materially below the rated load, whether figured on individual engines or on average station output. The fuel economy had not been very definitely determined but was also conspicuously deficient.

Subsequent to this the operating engineer of the station had continued running the plant along the lines which had been fixed upon by its designer and obtained slightly greater values for men and maximum output, and also somewhat improved producer performance. The results were still very far from satisfactory, however, and the efficiency test was determined upon. The plant was thoroughly overhauled in preparation for this.

We found that the engines had given much trouble with back firing, and some with premature ignition—the back firing having been so considerable that it had been deemed necessary to put throttles in the air lines and cut down the air supply to a point below that at which the rate of propagation of the flame in the mixture was a maximum. As is well known, this expedient, though wasteful of gas, is usually effective in stopping back firing, and it was so in this case.

We were told that the designer of the engines directed that they should be operated on "round-topped cards," and indicator cards which we took showed that these instructions had been observed. This shape of card is, of course, obtained by an ignition which is not early enough to bring about substantially complete combustion during the period of very slight motion near dead point. The exact timing of ignition to give this or other shape card naturally depends on many factors—principal among them being compression; composition and homogeneity of mixture; shape of combustion, space and placing of spark-plugs; temperature and character of spark. The sparking points on different cylinders varied somewhat and had been determined empirically as giving the desired card with the rich mixture deemed necessary to prevent back firing.

This practice led, as late ignition is likely to do, to a magnification of the unavoidable differences between successive cards, so that a diagram consisting of 20 successive cards on one unchanged governor and mixture setting showed a very large range of card contours.

We, therefore, found the engine running on round-topped cards, showing much variation of area from stroke to stroke, and with a mixture somewhat richer than that required for most rapid combustion, and we were assured by the attendants that if the mixture giving the best maximum card were employed the back firing would recommence and in time become prohibitive. Tentative experiments seemed to confirm this.

It, therefore, became necessary to determine the reason for this back firing, and we endeavored to analyze the situation and the possible trouble. Back firing is most likely to occur from ignition of the incoming

combustible mixture at the inlet valve. In a double acting engine it may also be caused by leakage of hot gases from the explosion in one end past the rings and into the other end where the suction stroke has just been completed. It may also occur from a lingering flame in an indicator fitting or from red-hot carbon or extremely hot metal parts so placed as to pocket gas between two heated surfaces.

If the back-firing is from escape of flame past the piston rings it can occur in only one of the ends of a given cylinder on a four-cycle double setting engine. For instance, if the cam setting is such that explosion is occurring in the crank end just as suction is finished in the corresponding head end, then we may have pre-ignition from this source in that head end, but since the cycle is never reversed, we can never have it in that crank end. Since one of the most obstinate cases of back-firing of which the writer has known was due to this cause the matter was carefully investigated. It immediately appeared that preignition occurred in crank ends and head ends indiscriminately and hence this was certainly not the only source even if it were an occasional one.

Indicator fittings, not already so made, were changed to a type which closed off practically flush with the inner surface of the cylinder and it was at first thought that this effected an improvement, but back-firing developed again, proving that it was merely chance which caused its diminution when the new fittings were put on.

This led to the inevitable conclusion that the trouble was either in ignition of the incoming gas by the outgoing exhaust, or in heated parts of the cylinder, or in both.

The design of the engine differed from that of most American tandem double-acting gas engines in that the valves occupied a bottle-shaped valve chamber connected by a relatively narrow neck with the main clearance of the cylinder. The exhaust valve was in the bottom, and the inlet in the top of this chamber, and they were separated from each other by a distance of only about ten inches. It will be remembered that this general type of cylinder was long regarded as standard but that most large gas engine manufacturers have worked away from it to a design with the inlet and exhaust valves separated by the entire diameter of the cylinder.

The valve setting was such that the inlet opened before the exhaust closed. The amount of this lap was different on different cylinders, due to slight wear of cams and rollers, and it could be controlled to some extent by adjusting the amount of clearance between the valve lifters and the stems, but with no normal adjustment which did not involve serious shock, was there a complete closure of the exhaust before the inlet opened. This suggested the idea that the slight back pressure presumably existing in the cylinders might force a little of the hot exhaust through the slightly opened inlet valve and thus ignite the mixture. This seemed highly improbable on account of the cool-

ing action on the gases which would result from intimate contact with the water cooled inlet valve-seat and the valve which has just risen from it.

By cards and other means we endeavored to determine whether the back-firing took place at the end of the stroke or at some intermediate time and concluded that it was at the beginning, and for the time we fell back on the hypothesis that the heated valve chamber caused ignition of the combustible mixture during the early part of the inlet stroke. Previous experiments on other engines made this seem improbable, however, even though the interior of this chamber might be assumed to be very hot, the improbability being due to the rapid motion of the gas and the fact that incipient ignition under such circumstances is usually promptly blown out and does not gain headway.

Finally, we took "toe cards" with a slight spring and a stop, and further cards with the ordinary spring, but with the back-lash in the indicator motion. These showed unforeseen conditions during exhaust and led to the true solution.

It was found that the pressure in the cylinder immediately after the opening of the exhaust valve was above atmospheric as might be expected, and that it remained there for a certain fraction of the exhaust stroke. For the remainder of the exhaust stroke, however, it was found that the back pressures became negative. In other words, during a portion varying with load and cylinder from the last two-thirds to the last half of the exhaust stroke there was a very slight vacuum in the cylinder instead of the positive back pressures which had been expected. This was found to be due to the wave set up in the long straight exhaust pipe by the first vigorous puff at release. From this it immediately became clear that on the opening of the inlet valve explosive mixture began at once to enter the segregated valve chamber even though the piston had not yet finished its exhaust stroke, and that this inflow became more and more vigorous as the valve opened farther, the incoming mixture mingling with the outgoing exhaust and in part passing out with it through exhaust valve.

The explanation of the back-firing became at once much easier. We did not have to assume that the exhaust gases were hot enough to pass through a slightly open, relatively cool valve and still retain the temperature required to ignite explosive mixture, nor did we have to assume walls so hot that a rapidly whirling and eddy blast of mixture impinging upon them was thereby ignited. On the contrary we need make only the very reasonable assumption that the exhaust gases aided by the walls were hot enough to ignite mixture which flowed in at a very low velocity, mingled thoroughly with the hot gases, and passed with them through a slightly separated valve and seat which had just been exposed to the hottest period of its cycle. It was no longer difficult to understand why back firing occurred; in fact it might fairly be assumed that a large number of incipient ignitions took place

which did not propagate back fast enough along the incoming gas stream to reach the mixing chamber, and were blown out as the velocity of induction increased. Assuming this to have been possible and also recognizing the fact that a theoretical mixture fires more easily, as well as propagates flame faster than others it was apparent why the rich slower burning mixture prevented most of the trouble.

It was all along apparent that the use of an unduly rich mixture besides being wasteful was objectionable through increasing the luminosity of the flame and hence the radiation, and through coating the interior of the cylinder with carbon deposits which tended to reduce the efficiency of the water jacket and to keep up to the skin temperature of the interior of the valve chamber. Thus the remedy applied, while effective, rendered it all the more necessary to continue its application in order to maintain running conditions. It was like using an opiate to induce sleep.

While this matter was being analytically investigated a series of cards had been taken running over various mixtures and ignition points, and also it had been concluded that on account of the extremely heavy reciprocating parts the maximum stresses on journals would not be greater with pointed than with round-topped cards, since apparently the largest component in maximum journal pressure was that due to inertia action rather than that due to the explosion. This last conclusion was borne out by the fact that the bearings ran much hotter on no load than on full load.

On account of the apprehensions of the station attendants these experimental settings were not at first maintained very long, but since they bore out the conclusions set forth above, we finally changed the operation of all the engines as to both mixture and ignition point. We first set the mixture with a slight excess of air over theoretic requirements, thus maintaining as complete and non-luminous combustion as possible, minimizing carbon deposit and keeping the valve chamber cooler and cleaner; and second, and even more important, we set the ignition materially earlier, thus running on a pointed rather than on a round-topped card, and thereby materially reduced the temperature of the gas at the moment of release when the explosive mixture mingled with it. After a short period of uncertainty marked improvement in operation became so apparent as to convert even the more skeptical of the station men, and while back-firing was not wholly eliminated occurring occasionally in all cylinders and particularly in certain ones, despite the change of practice, the degree of improvement was distinctively greater than we had hoped for.

This procedure alone enabled us to increase the engine output in a very notable degree and also, as would be expected, to materially cut down the amount of gas used per kw.-hour. This later reduction was so marked that it was not merely observable by the methods of the test but was noticed by the firemen. During a period when the engines were carrying con-

siderably higher load than had been their previous habit, the head fireman approached the writer and inquired why we were calling for so little gas. This, of course, was in large part due to the fact that a slightly "lean" rather than a slightly "fat" mixture was now employed.

In this connection some analyses of exhaust gas showed the interesting fact that if samples were taken as usual from a point a few feet below the top of the exhaust pipe the analysis indicated a rather large excess of air in the mixture at a time when it was practically certain that such an excess did not exist. This might easily have misled previous observers, and was, of course, due to the regurgitation of air into the exhaust pipe during the recession of the wave set up in the pipe by the "puff" occurring at release, as well as the loss of combustible mixture into the exhaust, as already noted.

A careful study of the producer conditions made by my associate showed the cause of certain wide variations in gas quality which had led to further very considerable losses of fuel, particularly as the mixture settings had not customarily been changed when gas quality varied. With suitable changes in firing practice it became possible not only to get a good quality of gas but what is more important to secure during most of the twenty-four hours a fairly uniform quality. The design of the producers was such that their operation was not continuous and during the periods of cleaning this uniformity could not be well maintained, but at such periods engine adjustments were permitted if they appeared desirable.

The station output records obtained while the designer of the engines was at the plant were available for comparison, while accurate records of previous lignite economy were not. In comparing output some difficulties were encountered due to the fact that in the earlier tests the engines were occasionally shut down, adjusted and otherwise interfered with in the hope of getting better operation. We believe, however, that it is a fair and conservative statement to say that reckoning either on peak or on average output during the 15 days of our tests subsequent to getting our system into operation, the performance of the plant was slightly better than 20 per cent in excess of that previously obtained. Since in both cases the same number of producers were run at maximum capacity it may be assumed that the fuel economy was bettered in about the same ratio, although, as stated, there were no thoroughly reliable early figures to compare our own careful records with.

It will be seen that the major part of this improvement turned upon the recognition of the fact that this engine would not of necessity give the same results upon a given type of card as would an engine of different clearance and valve design, and of the further fact that the arrangement of exhaust piping employed produced a period of slight vacuum in the cylinders during the latter half of the exhaust stroke.

The managers of the plant inform us that they were able after our departure to maintain and even to slightly increase the advantageous running conditions to which our work served to point the way. The conditions of their business were such that there was a market for all the product that could be produced and the power station was the limiting feature, hence this increase in power output paid in a very short time for a test that had been undertaken with a view to obtaining information bearing upon acceptance, and with no expectation of direct financial return.

MECHANICAL HANDLING OF MATERIALS.*

By RICHARD DEVENS.†

Within the last few years, some of our railroad, industrial and steamship companies have begun to realize the important part mechanical transference plays in the quick and economical handling of material.

The most efficient advances have been made in the handling of bulk material, such as ore, coal and grain, while package freight, comprising boxes, barrels, bags and other packages, which make up the load of a freight car, or the cargo of a steamship, has just begun to receive serious consideration.

It is no doubt a fact that the proficiency in handling bulk material was due to the difficulties to be overcome in the transportation and handling of iron ore to the center of the iron industry.

I have reference to the iron ore that was discovered in the Lake Superior country.

The first problem was the transportation, and this was overcome in 1855 when the Federal Government completed its first system of locks at the falls of the Sault Saint Marie River at Sault Saint Marie, Mich.

The second problem was the loading and unloading of the vessel. The loading was readily accomplished by the building of a long line of pockets on a dock, extending into the lake and the equipping of each pocket with chutes. The pockets were of such height that the ore would flow from them over the chutes and into the vessel by gravity. The railroad cars, of the bottom dump type, were brought over the top of the pockets and dumped into them.

It is interesting to note that the method used in the first loading dock is the one on which all docks have been constructed since.

The unloading has been the most difficult to accomplish in a quick and economical manner. The first vessels to carry iron ore were not constructed for the purpose, and while they carried some ore in the hold, most of it was carried on deck. When it was carried in the hold, it was hoisted to the deck by horsepower, and dumped into barrows, and then, like the deck cargo, wheeled ashore.

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The next step was the substitution of a small hoisting engine for the horsepower. This early method was in operation many years, and it was not until the dock managers were forced into it, by the great expense in carrying large storage on the dock, that any mechanical devices were attempted.

A cableway machine, built and erected at Cleveland, O., in 1880, under Mr. Alex. E. Brown's design and supervision, was the first mechanical plant. The next machines were of the bridge type. The method of handling the iron ore, over either the cableway or bridge, was to fill iron buckets by hand in the hold of the vessel and then hoist them by the machine and dump them automatically into railway cars or storage. In the hold there were from 12 to 15 shovellers to each machine, and there were two men on the machine, one an operator and the other a fireman.

Both of the above equipments were a great improvement over the early methods, and handled the iron ore in a satisfactory manner; yet they did not cut down the cost of the hand labor in filling the buckets in the hold. This was a very large part of the cost of unloading.

An automatic filling bucket had been worked successfully for a number of years in coal and similar soft material, but on account of the hard and lumpy nature of the early iron ores, it could not be operated in them.

With the use of soft Messaba ores, interest in the automatic filling or grab bucket was renewed, and about ten years ago the first successful grab bucket machines were erected and operated at the Illinois Steel Co.'s plant at Chicago by the Hoover & Mason Co. This plant was designed to unload from the vessel direct into railway cars. The success of this plant was the beginning of the present methods of unloading iron ore.

There have developed two types of grab bucket machines; one with the grab bucket suspended from wire ropes and the other with the grab bucket carried on a rigid arm.

The cost of filling the buckets by hand was about 13 to 15 cents per gross ton, and the cost of hoisting and dumping into railway cars or storage from 1½ to 2 cents per gross ton, making the total cost of unloading from 14½ to 17 cents. With the grab bucket machines, this total cost has been reduced to from 1 to 2 cents per gross ton, depending on the distance the ore is carried from the vessel.

The hand-filled buckets were of about 1-ton capacity, as this size had been found to be the most practical for filling and handling in the hold.

With the grab bucket the size is only limited by the dimensions of the hatch and the shape of the vessel.

The first grab buckets for iron ore were of 5-ton capacity, but since then machines have been built to handle 7½, 10 and 15 tons.

Besides reducing the cost of unloading, the ability to handle in larger units has reduced the time; whereas with the hand-filled buckets, to unload a 6,000-ton

vessel was a question of days, it is now only a question of hours.

The steamer "Morgan," of the Pittsburg Steamship Co., with a cargo of 11,319 tons of ore, was recently unloaded at Fairport in 5 hours and 58 minutes. The work was done with six Brown electric unloaders.

These improvements have also increased the earning capacity of the vessel by making possible a greater number of trips during the season. This is shown by the following comparative statement for the years 1906 and 1910, showing the average stay at upper and lower ports of the vessels of the Pittsburg Steamship Company:

	Year 1906.		Year 1910.	
	hr.	min.	hr.	min.
Average stay in lower lake ports.....	36	15	22	22
Average stay in upper lake ports.....	22	25	12	22
Average time spent in port receiving and discharging cargoes.....	58	38	34	44
	Gross Tons.		Gross Tons.	
Average cargo carried.....	5,951		6,631	
Largest cargo carried.....	13,333		13,296	
	In 70 min.		In 45 min.	
Fastest loading record.....	9,277		9,788	
	Tons per hr.		Tons per hr.	
Rate of fastest loading record.....	7,288		13,051	

In the foregoing I have outlined the development of handling bulk material, using iron ore as an example. The handling of package freight has not been brought to the same degree of perfection.

In many manufacturing concerns mechanical devices have been installed to reduce the cost of handling and to hasten the transportation of their products, but for quick and economical handling of freight at shipping docks and railway terminals little has been done in this country.

In Europe greater advances have been made, due largely to the encouragement given by the city or government, which frequently itself equips the docks.

At Hamburg, Antwerp, Bremerhaven, Glasgow, London, Manchester, Havre and many other ports are found mechanical appliances, each to meet the local requirements, but all aiming to reduce the number of handlings and the cost of same.

In England, at the freight stations and warehouses, the practice is to install jib cranes, so arranged that they can serve all the floor space from car or wagon.

In this country many of the railroads have put in hand cranes of the pillar or bridge type, for handling freight from cars to wagons, or vice versa, but they are mostly for heavy lifts, and are slow in operation, and cover only a limited area.

Some of the railroads have put in electric cranes in their freight yards and water terminals; as, for example, the Pennsylvania R. R. Co. on its Greenville docks, the New York Central & Hudson River R. R. Co. at Port Morris, the Philadelphia & Reading R. R. Co. at Port Richmond and the Central Railroad of New Jersey at Communipaw.

Many of the railroads are coaling their locomotives at greatly reduced cost and time by mechanical appliances, but the question of handling their package freight at terminals is still open.

Most managers have known that there is a great loss of time in transferring freight at terminal and intermediate points, but few seem to realize the high costs that this involves.

Perhaps the most complex movements in the handling of package freights are at the large steamship piers, due to the great carrying capacity of the large vessels, the many consignees, each having his allotted space, and the limited floor area that has to be cleared quickly to make room for the next vessel. The larger railroad terminals also have their many consignees, but the floor area is not so restricted.

The placing of the packages in the proper space is done by the hand-truck. A sling load from the vessel, or a railway car, may contain packages for several consignees. The truckman cannot wait to sort as he receives them, so must load his truck with them as they come. This means a long travel to get the packages to their allotted space. In order to tier them, several more handlings are necessary. All this leads to congestion and increasing cost per ton. This is further affected by the rise in the cost of labor, materials, rent and larger terminals.

Each terminal is a problem in itself, as is each manufacturing establishment, so that it is necessary to make a careful study of the conditions to be met before any mechanical method can be proposed.

In the last 30 years there has been a steady increase in the capital invested in manufactures, which means an increase of tonnage of all kinds of package freight carried by the steamship and railroad companies. To meet this, the railroads have increased their rolling stock and either enlarged their terminals or built more. In large cities this has been at great cost for land and buildings. The method of handling the freight has remained the same.

At a terminal there are two kinds of freight—outbound and inbound. The outbound is transferred from wagons into the outbound freight house and thence to the railway cars or directly from the wagons to the cars. The inbound is vice versa.

All the above movements, except between wagons and cars, involve the sorting and distributing of each package to its designated place. It is also necessary to transfer cars from one freight house to the other, as the use of the hand truck necessitates bringing the cars to the freight.

A mechanical equipment to be satisfactory must be able to distribute the outbound and inbound freight simultaneously. There should be no rehandling, and every square foot of floor space should be served with a single handling.

All motions of lifting and conveying should be done by power. The machinery should be designed to give the greatest lift required and to transfer to any reasonable distance, and then tier or lower into cars.

Continuous operation should be sought for to avoid delay. No part of the transference should be along the floor and the equipment should not take up any

floor space that can be used for other purposes. All movements of the mechanical equipment should allow of the assorting and distributing according to classification and allotted space readily and quickly. There must be reserve capacity to prevent congestion, in case of extra demands. The justification for the investment of the mechanical installation lies in the reduction of cost and the saving of time in handling. The expense should be in proportion to the size of the terminal.

There are many companies in this country engaged in the manufacture of hoisting and conveying machinery. While perhaps no one makes all the necessary appliances, yet a combination of their product could be used to fill the special requirements of each terminal point.

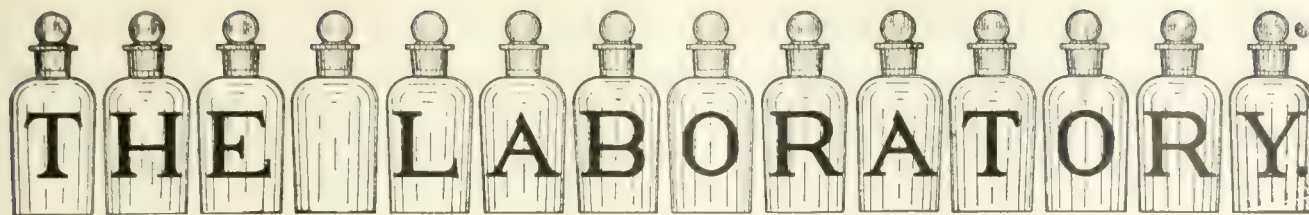
Fully to cover the floor space and obtain all the different requirements for the satisfactory handling of the package freight, three units or different types of conveying machinery are necessary. These are the single rail electric trolley, the bridge traveler and the cross traveler. The electric trolley is the actual load-carrying part of the equipment, the single rail, bridge traveler and cross traveler furnishing a combination of loop track system on which the trolley can reach any part of the area to be covered. All movements should be so regulated that there will be no interference, and many trolleys can be in operation following one another. Each trolley can draw a number of trailer trolleys, so that many packages can be hoisted and transported under the control of one man. This arrangement allows many loads to be transported in close sequence simultaneously, and with maximum hoisting and traversing speeds, gives the greatest range and capacity at a minimum of labor and maintenance.

At some freight terminals it may be necessary to have, in combination with the above mechanical conveyors, motor trucks on the surface; in others, belt conveyors.

There is no doubt that some such scheme as outlined above, when properly carried out to meet the special requirements at any terminal, would materially reduce the time and cost involved in the present method. This has already been exemplified in the handling of bulk material.

Considering the special attention now being given this question by several engineers and the interest shown by many steamship and railroad managers, it can be safely stated that within the next few years great changes and developments will be accomplished.

At Scheelite in Nova Scotia, on Moose river, continuous operations have been carried on since March 4, 1910, in connection with the tungsten-bearing veins discovered in the fall of 1907. An average of 40 men has been employed. The veins so far discovered are interbedded lodes and coincide with the bedding planes, and occur in thin layers of slate interstratified with beds of quartzite.



A COLORIMETRIC METHOD FOR VANADIUM IN IRON AND STEEL.

By CHAS. R. M'CABE.*

A colorimetric method for vanadium in iron and steel has lately been adopted in the laboratory of The Lima Locomotive & Machine Co., Lima, O. It requires from three-quarters of an hour to two hours according to the nature of the material, and has the advantage of permitting the determination of tungsten, nickel and chromium in the same portion used for the vanadium determination.

The color employed is that obtained by adding hydrogen peroxide to an acid solution of vanadium. Before proceeding to a description of the method, it is necessary to direct attention to certain properties of vanadium on which it is based.

The color of vanadium with hydrogen peroxide is subject to many disturbing influences. It is masked and otherwise influenced by iron; strong acids deepen it, and when present in large amounts impart a tinge of orange; an excess of hydrogen peroxide partially bleaches it, and finally any vanadium not in the most highly oxidized form does not yield any color at all. In view of these facts it need hardly be said that the conditions under which the color is produced must be rigorously controlled.

Vanadium, like other metals, may be separated from the bulk of iron by means of ether, but unlike nickel or copper cannot be further purified by the ammonia precipitation of iron. Ammonia does not precipitate vanadium existing alone in solution, but when any considerable quantity of iron is present, the precipitate produced by ammonia retains vanadium with great tenacity. This reaction may be a precipitation of vanadium as a vanadate of iron, although it apparently partakes of the nature of occlusion. These properties of vanadium make it possible to obtain all the vanadium from a vanadium steel with a limited quantity of iron and free of nickel and chromium when these metals are present. This is accomplished by making the ether separation of the bulk of the iron in the usual way, and following with an ammonia precipitation, having oxidized chromium when present after the ether separation is made.

It is evident that if we dissolve the ferric hydrate carrying vanadium in hydrochloric acid and decolor-

ize the ferric salt in some manner, that we have obtained the vanadium in form for a quantitative development of the color in hydrogen peroxide; that is, in such form that it may be compared with a standard. It is further evident that this standard must contain nearly the same amounts of iron, acid, and hydrogen peroxide as the sample being analyzed.

The essential features of the method are as follows: Two grams of vanadium steel are dissolved in hydrochloric acid, the iron oxidized with nitric acid, and an ether separation made. The amount of iron in the acid solution is purposely increased in order to have sufficient to occlude all vanadium when precipitated. In the absence of chromium, this acid solution containing vanadium is diluted and precipitated with ammonia. If nickel is present, a second ammonia precipitation is made. The ferric hydroxide carrying vanadium is dissolved in a limited amount of hydrochloric acid, the solution decolorized by addition of hydrofluoric acid, and the vanadium color produced by addition of hydrogen peroxide. The quantities of acids and hydrogen peroxide must very carefully be measured. The color thus produced is compared with a standard.

When chromium is present, it is oxidized by potassium chlorate before precipitation with ammonia, and so passes into solution along with nickel.

The standard is an artificial one. It is prepared by mixing ferric chloride and hydrochloric acid in such a way as to imitate the solution of the ferric hydroxide in hydrochloric acid following the ammonia precipitation, decolorizing with hydrofluoric acid, and adding a quantity of a standard solution of vanadium pentoxide. The color is then produced by hydrogen peroxide, and the standard diluted as required. Both standard and test must contain the same quantity of acids and hydrogen peroxide, which may be accurately and quickly measured by small pipettes.

The solution required for this method follows:

(1) A dilute solution of hydrogen peroxide. The amount required in a determination is 1 cc. of the 2.6 per cent solution furnished by the dealers. It is, therefore, convenient and also more accurate to employ a solution of such strength that an amount which may be measured by a small pipette will contain this desired quantity. For example, dilute 100 cc. of the 2.6 per cent solution to 500 cc. and use 5 cc. in a determination.

(2) Hydrofluoric acid. The amount used in a determination is $\frac{1}{2}$ cc. of the usual cp. product. It may be diluted for the purpose, or a small graduate may be used, first filling to a convenient point with water,

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then measuring the acid by adding drop by drop from the paraffin bottle.

(3) A hydrochloric acid solution of ferric chloride containing one-tenth of a gram of iron and an amount of acid equivalent to $1\frac{1}{2}$ cc. of the strong 1.2 sp. gr. product in each 5 cc. Dissolve 50 grams of the salt (crystallized with 6 molecules of water) in 300 cc. of dilute hydrochloric acid (1:1) and dilute to 500 cc. Or the solution may be prepared by dissolving a plain steel in nitric acid, precipitating with ammonia, filtering, washing, and dissolving in hydrochloric acid and diluting as required.

(4) A neutral standard solution of vanadic acid. Vanadic acid, or vanadium pentoxide, cannot at present be obtained on the market. Its preparation, however, is not difficult. Select a good grade of vanadate of iron containing about 60 per cent of V_2O_5 , 38 per cent of Fe_2O_3 , and 2 per cent of SiO_2 . This may be obtained of The Primos Chemical Co., Primos, Delaware County, Pennsylvania. Dissolve 5 grams in 100 cc. of strong hydrochloric acid in a beaker by aid of heat. Add 20 cc. of water and 1 cc. of hydrofluoric acid. Filter into a separatory funnel. Add 50 cc. ether, and shake under a hydrant in the usual manner. Draw off the lower layer, and discard the ethereal solution. Return the acid solution to the funnel, decanting from the silicious matter which settles in the bottom of the solution. Make a second ether separation of the iron and return the acid solution to the funnel, again decanting from silicious matter. Finally make a third separation of iron, and filter the acid solution. The vanadium solution, now free of iron and nearly free of silica, is purified of ether by heating on a steam or electrical heating appliance, or by blowing air through it. The quantity of contained ether is so large that it cannot be safely brought near a flame.

Dilute in a 600-cc. beaker to 250 cc. and heat to boiling. Add cautiously a small amount at a time, about 10 grams of potassium chlorate. Follow with a sufficient quantity (from 25 to 35 grams) of potassium carbonate to nearly neutralize the solution, but be careful not to over neutralize the solution. A copious precipitate of hydrated vanadic acid is thrown down when the right degree of acidity is reached. Dilute and boil. Filter through a 15 cm. filter paper, wash thoroughly with hot water, and apply suction to collect the precipitate in a cake. Separate from the filter paper, place on a watch glass, and dry in an oven at about 100° C. As the moisture is being expelled, break the mass into small particles from time to time with a glass rod. Finally when entirely dry pulverize and preserve in a vial for use.

If the vanadium pentoxide is dried at too high a temperature, or if it is not broken up in the partly dried state, it becomes very hard and difficult to pulverize.

Weigh accurately about two-tenths of a gram of the vanadium pentoxide into a small beaker. Add 36 cc. water and 4 cc. strong hydrochloric acid. Heat gently,

and when all is in solution add 4 cc. ammonia, sp. gr. .96. Dilute so that 1 cc. contains .0002 vanadium, or an amount corresponding to .01 per cent in a 2-gram sample of steel.

This standard solution of vanadium pentoxide assumes a yellow color upon addition of the ammonia. In time it becomes colorless, but this change has no influence on the strength of the solution, or results obtained by its use.

In preparing the standard vanadium solution, an important essential is to employ the minimum amount of hydrochloric acid in dissolving the vanadium pentoxide. There are several reasons for so doing. Much hydrochloric acid reduces vanadium pentoxide to dioxide upon application of heat, and this must be avoided by using hydrochloric acid as dilute as practicable and heating gently. Again, much acid in the standard must be avoided on account of its influence on the color. The ferric chloride solution contains the correct amount, and it is necessary to be able to use any required amount of the vanadium solution without appreciably affecting the acid content of the standard. Nor is it practicable to use a liberal quantity of acid in dissolving the pentoxide and neutralize the excess, since chlorine in any form beyond a certain amount retards the decolorizing action of hydrofluoric acid on ferric chloride.

The correctness of the vanadium standard is of course absolutely necessary to satisfactory results in this method. It must be remembered that the vanadic acid as precipitated contains a very large quantity of water, and time must not be called too soon on the drying process. The directions as given yield a strictly chemically pure vanadium pentoxide, but, nevertheless, it is advisable to test its purity by ignition or otherwise.

This method as practiced by the writer is applicable to nearly all steels and irons. Absence of chromium greatly simplifies the process. It is suggested that one first attempting to use this method select a standard vanadium steel free of chromium. The Bureau of Standards furnishes a plain vanadium steel containing .15 per cent vanadium which may be used to gain experience with the method. The details of the method follow:

ABSENCE OF CHROMIUM.

Weigh 2 grams of the vanadium steel into a dish or beaker, and add 40 cc. strong hydrochloric acid, and heat until the steel is dissolved. Add 5 cc. nitric acid (sp. gr. 1.2) and evaporate to about 10 cc. Pour into a separatory funnel, and wash the dish or beaker with hydrochloric acid (1:1) using a small quantity at a time until the volume of the solution amounts to 20 cc. as shown by a mark on the funnel. Add 50 cc. of ether free of alcohol, stopper tightly, and shake under a hydrant. Allow to stand in the rack until the two layers separate completely. Draw off the lower layer into a 25 cc. graduate, and take $1\frac{1}{2}$ cc. of the ethereal solution. This method of making the ether

separation yields approximately one-tenth of a gram of iron in the acid solution, an amount which occludes vanadium satisfactorily and yet permits of the separation of nickel when present.

Transfer the acid solution to a 400 cc. beaker, dilute to about 225 cc. and bring to a boil. Or, to save time, have some boiling hot water at hand and dilute with it. Add ammonia to the boiling solution cautiously, with brisk stirring; when iron is precipitated, add 10 cc. ammonia in excess. Continue to boil for one-half minute. The iron thus precipitated carries with it all the vanadium, while manganese is almost entirely left in solution. If the iron is precipitated in the cold solution and the solution then boiled, a certain amount of manganese is occluded. Upon attempting to dissolve the precipitate in the necessarily limited amount of acid, the manganic hydroxide remains undissolved and retains some vanadium.

Filter the ferric hydroxide with its occluded vanadium on an 11 cm. filter paper. Wash with hot water, and if nickel is present, dissolve in hot dilute hydrochloric acid back into the beaker in which precipitation was made, and make a second precipitation in the same manner, filtering and washing as before. In the absence of nickel, a single precipitation of iron by ammonia is sufficient. This is the case in the Bureau of Standards, sample No. 24.

Open the filter paper containing the ferric hydroxide and vanadium in a dish. Add 3 cc. hydrochloric acid (1:1) and about 7 cc. water. Rock the dish, heating moderately, until the precipitate has dissolved. Do not attempt to hasten solution by stirring or macerating the paper with a rod, as objectionably large quantities of fiber become suspended in the solution. The solution is clear if the occlusion of manganese is avoided by making the precipitation in the manner described.

Pour the solution into the tube in which comparison is to be made. Camp's manganese comparison tubes are recommended for the purpose. It is convenient to use two of the large, glass-stoppered tubes, since the standard can thereby be prepared without the necessity of pouring from one to another.

Wash the dish and paper with cold water and pour washings into the tube containing the solution. Dilute to about 40 cc. and add $\frac{1}{2}$ cc. of hydrofluoric acid in dilute form or by use of a small graduate as described. A complete decolorization of the iron salt occurs. If the solution is warm, this effect is not complete until it is cooled. Presence of chlorine beyond the stipulated amount retards the decolorization of iron, while an unusually large amount of iron may require more hydrofluoric acid. But if directions are carefully observed, no trouble need be experienced at this stage.

To the colorless solution add 5 cc. of the dilute hydrogen peroxide solution containing 1 cc. of the 2.6 per cent reagent. Upon mixing the brown vanadium color appears.

The standard is prepared as follows: Into the other tube place 5 cc. of the ferric chloride solution containing one-tenth gram of iron and 3 cc. dilute (1:1) hydrochloric acid. This solution is similar to that obtained when the ferric hydroxide is dissolved in the dish. Add a quantity of the neutral standard vanadium solution, accurately measured from a burette. The amount to be used depends upon the vanadium content of the test; 15 cc. is the best quantity to take at first. Dilute to 40 cc. and bleach the solution with hydrofluoric acid, as in the test, and add hydrogen peroxide, using exactly the same quantities of acid and hydrogen peroxide as were used in the sample being analyzed. Dilute to 5 times as many cc. as there were cc. of the standard vanadium solution taken, and mix. Compare colors in the usual way. One-fifth of the number of cc. to which the test is diluted is hundredths per cent vanadium. For example, if 15 cc. of the neutral vanadium solution are used in the standard, dilute same to 75 cc. If the volume of the test is 82 cc. when the colors match, the percentage of vanadium in the sample is .164.

As in color methods generally, the standard must contain approximately the same amount of the substance being determined as the test sample. The range of negligible departure is about .03 per cent. Hence, the number of cc. of standard vanadium solution is conveniently taken in multiples of five. In the example cited, if the test is diluted to 95 cc. before the colors match, another standard is prepared containing 20 cc. of the vanadium solution and diluted to 100 cc. Upon again comparing colors, the test is found to be slightly dark, requiring further dilution. It is best to use a minimum amount of vanadium solution at first, since any necessary change in standard is effected by merely adding more of the vanadium solution. In the example given, it is sufficient to add 5 cc. of the vanadium solution to the standard and dilute to 100 cc.

PRESENCE OF CHROMIUM.

When chromium is present in a vanadium steel, it is all found in the ammonia precipitate following the ether separation. Hence, some modification of the method is necessary, since upon solution of the precipitate in hydrochloric acid the chromic chloride imparts a green color to the solution. The plan followed is to oxidize chromic oxide to chromic acid following the ether separation, thus avoiding the precipitation of chromium by ammonia. The details are as follows:

The bulk of the iron is removed by the ether separation, which is made in the same manner as when chromium is absent, except that the volume of the acid solution is received into the 400 cc. beaker, and $1\frac{1}{2}$ cc. of the ethereal solution received into a small graduate and added to the acid solution in the beaker. This is merely to avoid increasing the bulk of the acid solution by the washing which is necessary when it is received into a graduate.

Evaporate the acid solution, which need not exceed 20 cc., almost to dryness in the beaker. This is accomplished rapidly, and with a good quality of glassware, without danger of breakage, by constant agitation of the uncovered beaker on a hot plate or over a bare flame. The solution is carried down to 1 cc. and 50 cc. of strong nitric acid added. The beaker is covered and the solution boiled until fumes are all expelled and the solution is a clear green. To the boiling nitric acid solution enough potassium chlorate (about 2 or 3 grams) is added to precipitate manganese and oxidize chromium. Continue boiling a moment or two, then dilute to about 225 cc. with boiling hot water. Add cautiously to avoid loss by violent ebullition. Then add cautiously, to the boiling solution, with vigorous stirring, enough ammonia to precipitate iron, and 10 cc. in excess. Boil for $\frac{1}{2}$ minute. The chromium being oxidized to chrome acid is not precipitated, while the iron occludes vanadium as before.

Filter the precipitated manganese, iron and occluded vanadium on an 11 cm. filter paper, and wash with hot water. In addition to manganese, this precipitate contains a notable quantity of nickel, when this metal is present, and also a certain amount of occluded chromic acid, which cannot be removed by washing. A second ammonia precipitation is therefore necessary. Dissolve the precipitate in hot dilute hydrochloric acid back into the beaker in which precipitation was made, dilute to about 225 cc. and make a second precipitation by ammonia, stirring the boiling solution vigorously and adding 10 cc. of ammonia in excess as before. Filter on an 11 cm. filter paper and wash with hot water.

The iron precipitate being now freed of nickel, manganese, and all but a trace of chromium, is treated as described when chromium is absent. It is to be observed at this point that upon addition of hydrofluoric acid the decolorization of the ferric chloride generally reveals a small quantity of chromic acid in solution. If the ammonia precipitations have been properly carried out, this retained chromium is not a source of error, the hydrogen peroxide converting the yellow color into blue, which rapidly fades. This change, however, is not apparent, being masked by the brown vanadium color. If through improper management of the ammonia precipitation the amount of chromium in the solution is increased, its color is not thus easily destroyed.

PRESENCE OF TUNGSTEN.

When tungsten is present in a vanadium steel the filtrate from the tungsten determination is used for the vanadium determination. An important fact to be remembered in this case is that the evaporation to dryness in the tungsten determination has reduced a portion of the vanadium. It is therefore necessary to reoxidize it during the evaporation of the filtrate preceding the ether separation of iron. To the filtrate

add 5 cc. of nitric acid (1.2 sp. gr.) and proceed according to the presence or absence of chromium. Avoid the use of sulphuric acid in the tungsten determination.

PRESENCE OF TITANIUM.

Although hydrogen peroxide gives a yellow color with titanium, nevertheless presence of titanium does not interfere with the vanadium determination. This is due to the fact that hydrofluoric acid in the amount prescribed destroys the color due to titanium. This is true for any amount of titanium apt to occur in iron or steel. The color of vanadium with hydrogen peroxide, however, is slightly modified in presence of titanium. Comparison by artificial light is advisable when titanium is present. The dissimilar character of the colors then vanishes, and the eye perceives only their relative depths. The error, if any, due to titanium is slight.

PRESENCE OF MOLYBDENUM.

Molybdenum, like titanium, gives a yellow color in acid solution with hydrogen peroxide. However, the properties of molybdenum are such that when present in a vanadium steel it does not appear in the solution in which the vanadium color is produced, the bulk being removed in the ethereal separation with iron, and the balance in the ammoniacal filtrates with nickel and chromium.

PIG IRON.

The application of this method to pig irons is obvious. It is only necessary to remove the insoluble in any convenient manner, avoiding the use of sulphuric acid. As in tungsten steels, the reducing action on vanadium of a complete evaporation of the acid solution must not be forgotten, and an oxidizer, preferably nitric acid, added to the hydrochloric acid solution previous to the ether separation of iron.

There are certain limitations to this method which must be understood in order to apply it successfully. A cardinal rule is to treat standard and test exactly alike in making the comparison. The vanadium color with hydrogen peroxide is sensitive, and neglect of this precaution will lead to erroneous results. Another feature is the limited capacity of ferric hydrate to occlude vanadium. This property of ferric hydrate is such that with a two-gram sample of vanadium steel, the amount of iron, one-tenth gram, used in the acid solution preceding the ammonia precipitation occludes all the vanadium without appreciable loss when the percentage is under .20. Above .20 per cent and under .30 per cent the loss may be neglected in ordinary routine analysis. But above .30 per cent the loss rapidly increases, so that it is necessary to take a smaller sample for analysis. A safe working rule is to take such a sample that the contained vanadium is not in excess of six milligrams. When less than two grams of sample is taken for analysis the amount of iron necessary in the acid solution following the ether separation, namely, one-tenth gram, may be ob-

tained by taking more of the ethereal solution. Or some of the ferric chloride solution may be added.

An important precaution is to avoid any step in the analysis which tends to reduce vanadium. A small amount of reduced vanadium is reoxidized by hydrogen peroxide itself after a lapse of about ten minutes with a simultaneous development of color, but large amounts are only imperfectly reoxidized by the limited amount of hydrogen peroxide.

The question of the reliability of this method is one which cannot be judged by a comparison of its results with those of older methods. Rather, we must pass judgment by a careful analysis of the method itself, the principles on which it depends, and its sources of error. It may be said, however, that the Bureau of Standards Sample No. 24, containing .15 per cent vanadium, shows an exact check. It also shows the correct result when nickel, chromium, titanium, and molybdenum are added to the hydrochloric acid solution preceding the ether separation. In view of the care exercised in the analysis of this sample, such a result appears not without value. With certain analyzed samples submitted by interested parties for confirmation the method showed somewhat high results, amounting to .05 per cent in one instance. Others showed a very satisfactory check. To interpret such evidence, we are obliged to examine our method and judge independently of its merits. It is not unfair to consider that every factor which influences the color of the vanadium with hydrogen peroxide is under control; that the property of ferric hydrate of occluding vanadium quantitatively is easily demonstrable; and that if there is any source of serious error it remains to be discovered. Note must be taken of the anomalous circumstance that the method often shows high results compared with older methods. Slightly low results might reasonably be expected, since the power of ferric hydrate to occlude vanadium, while satisfactory for our purpose, is not absolutely perfect. This consideration directs suspicion to the older method when the two show discordant results. But the evidence which appeals most strongly to reason is that obtained by the analysis of artificial mixtures of vanadium solutions with solutions of plain and special alloy steels. Such mixtures have always shown the true vanadium content, within very reasonable limits. The recovery of all the vanadium present in the hydrochloric acid solution thus being an established fact, the correctness of the vanadium standard may be confirmed and the chain of evidence is complete.

It is perhaps too much to claim that this method is universally applicable to all irons and steels. Reference has been made to the influence of elements occurring in vanadium steels on the color. It must be understood that these observations refer only to the quantities of these elements commonly found in vanadium steels. When present in abnormally large quantities, the question of the availability of the

method must rest on the judgment of the operator. But it may be affirmed without reserve, that material to which this method is inapplicable is of rare occurrence.

The writer acknowledges his obligation to the following parties for materials and samples furnished during the experimental work: Geo. L. Norris, of the American Vanadium Co., Pittsburg, Pa.; J. Lloyd Uhler, of the Union Steel Casting Co., Pittsburg, Pa.; C. M. Johnson, of the Crucible Steel Co., Pittsburg, Pa. and the Primos Chemical Co., Primos, Pa. Credit is also due Booth, Garret & Blair for correcting a misstatement relative to the non-interference of molybdenum.

A METHOD FOR THE RAPID DETERMINATION OF LIME IN LIMESTONES, MARLS, SLAGS, ETC., SOLUBLE IN DILUTE NITRIC ACID.

By H. B. KINNEAR.

Method: Weigh $\frac{1}{4}$ gram of the pulverized sample and transfer to a 150-cc. beaker; moisten with water, add 30 cc. HNO_3 (1.2 sp. gr.) and heat to boiling on a hot plate. While heating, measure 60 cc. of a standard oxalic acid solution (1 cc. = .0025 gns. CaO) from a self-filling pipette into a 400-cc. beaker and dilute to 200 cc. with boiling water. Now add the nitric acid solution of sample, washing the beaker thoroughly with hot water and carefully make strongly alkaline with ammonia; heat just to boiling and by using a bell-jar and strong suction, filter at once through a cylindrical separatory funnel fitted with a platinum disc and thin pad of asbestos pulp; wash precipitate three or four times with plenty of hot water catching filtrate and washings in a 500-cc. beaker.

Acidulate with hot sulphuric acid (1:3) and titrate with a standard KMnO_4 solution, each cc. of which is equal to 1 cc. of the oxalic acid solution added above. The percentage of CaO in the sample is of course equal to the difference between the amount of oxalic acid measured out and the reading of the KMnO_4 burette.

The above method has been used by the writer on samples of limestone containing from 3 per cent to 17 per cent magnesium carbonate, on marls containing as high as 7 per cent organic matter and on blast furnace slags ranging from 29 per cent to 35 per cent in silica with admirable success, checking gravimetric standards nicely.

The time consumed should not exceed 8 minutes including weighing, and determinations have been made in 6 minutes where the ferric oxide and alumina in the sample were very low, thereby favoring rapid filtration.

In all determinations strong suction has been used

and never has a precipitate "run through." The method used here, in precipitating calcium oxalate seems to leave it free from magnesium oxalate thereby making reprecipitation unnecessary in case of dolomites. Furthermore it produces a coarse granular precipitate easily retained.

The solution required follows:

Potassium permanganate, 2.881 gms. per liter. Standardize against a standard iron ore or pure iron wire. 1 cc. = .005 gms. Fe.

Oxalic acid, 5.75 gms. per liter. Standardize against the above KMnO_4 solution making the two solutions equivalent cc. for cc.

REPORT ON METHOD OF DETERMINATION OF GLYCEROL.*

By EUGENE PROBECK.

For some time chemists have failed to agree upon a standard method for the determination of glycerol. As the two methods most in use, the acetin and the bichromate, are fully described in many textbooks, it is unnecessary to repeat them.

In the bichromate method, I did not remove the chlorine and albuminous matter as recommended by many chemists but found it more advantageous to make a correction for these impurities by means of a blank test. This blank test depends upon the volatility of glycerol below 160°C. , leaving behind the salts, polyglycerols, albuminous matter and other organic impurities which do not volatilize.

Preliminary Test.—I weighed out about 2 grams of the sample, diluted it to 200 cc. and tested 20 cc. of it for acidity, using phenolphthalein as an indicator. I found it took 2 drops of $N/2$ sodium hydroxide to neutralize the 20 cc. I added 2 drops to each 20 cc. I used for analysis and also added 2 drops to each 0.2 gram of glycerine used for the blank test. The purpose of using an alkaline solution is to prevent any volatile acid from escaping by converting it into its sodium salt. If the glycerine is not neutralized in the blank test, the effect of the volatile acids on the bichromate solution cannot be determined.

The so-called blank test is made by evaporating about 0.2 gram of neutralized glycerine on a watch glass at 160°C. in a manner similar to the carbonaceous test given by Lewkowitsch in his "Technology on Fats, Oils and Waxes." The residue is oxidized with the bichromate solution at the same time that the glycerine is. To prevent overheating and be certain of complete oxidation, both the residue and glycerine are heated on the same water bath from $1\frac{1}{2}$ to 2 hours. The number of cc. of bichromate solution used to oxidize the residue is deducted from the number of cc. used for the sample and from the difference, the per cent of glycerol is calculated. The blank test makes a correction for all impurities likely

to be present except volatile monohydric and dihydric alcohols and volatile aldehydes, chiefly acrolein.

I tested the glycerine with an ammoniacal silver nitrate solution but no silver mirror appeared, proving absence of aldehyde. If aldehydes are present, they must be removed, or the acetin method must be used. Experiments were made to find out the conditions under which acrolein is formed. I heated some crude glycerine to 160°C. as quickly as possible and then to 200°C. and no odor of acrolein was noticeable.

Carbonaceous tests have also been made to find out first, the effect of temperature on the glycerine and second, the effect of sodium chloride on glycerine when heated. For convenience and clearness, it is well to divide the work into five parts:

I. Two carbonaceous tests were made of the crude glycerine with these results: first trial 5.45 per cent, second trial 5.50 per cent.

II. A 100 cc. Florence flask was nearly filled with crude glycerine and heated as quickly as possible in an oven to 160°C. The glycerine was kept at this temperature for one hour and then two carbonaceous tests were made with these results: first trial 5.50 per cent, second trial 5.55 per cent.

III. The Florence flask was refilled with fresh glycerine, heated as quickly as possible to 200°C. and was kept at this temperature two hours. The carbonaceous tests made with this glycerine gave nearly the same results as in the first two cases, indicating that the glycerine can be heated to 200°C. without polymerization or decomposition. The results were: first trial 5.54 per cent, second trial 5.60 per cent.

IV. A fresh portion of glycerine was treated in the same manner as above except that the temperature was raised as quickly as possible to 250°C. and kept at 250°C. for two hours. The carbonaceous tests gave much higher results, as shown. First trial 12.60 per cent, second trial 12.50 per cent. These results indicate that glycerine cannot be heated to 250°C. without decomposition, polymerization, or both.

V. Another Florence flask was nearly filled with crude glycerine and considerable sodium chloride was added. The glycerine was treated in the same way as in IV. The carbonaceous tests gave these results: first trial 8.80 per cent, second trial 9.01 per cent. The presence of salt in this case seems to retard the formation of polyglycerols. Sodium chloride was used because it occurs in soap lye glycerine and it has been claimed that salts in glycerine interfere with the bichromate method.

The object of making so many carbonaceous tests is to show that the blank test can be used to advantage in determining glycerol in both candle glycerine and soap-lye glycerine. In case volatile monohydric and dihydric alcohols are present, both the acetin and bichromate methods are unreliable, as these alcohols are capable of being converted into acetyl compounds

*From The Journal of Industrial and Engineering Chemistry.

in the first method and capable of being oxidized in the second.

To show how accurate each method is, I made duplicate analyses of a crude glycerine with the following results:

	Acetin method. Per cent.	Bichromate method. Per cent.
First trial.....	86.08	86.93
Second trial.....	86.56	87.27
Average	86.32	87.10
Blank test correction.....		0.30
Corrected result.....		86.80

From my experience with the acetin method, I conclude that no matter how carefully the method is carried out, the results are from 0.2 per cent to 0.5 per cent below the true value. The difficulty with the method is not on account of the impurities but on account of the manipulations introducing several sources of errors.

The bichromate method is preferable in the absence of acrolein and other volatile aldehydes. If aldehydes are present, they must be removed before using the bichromate solution. A blank test simplifies the method and makes the result more accurate than by purifying the glycerine.

METHOD FOR NICKEL-ZINC SEPARATION IN GERMAN SILVER AND OTHER ALLOYS.*

By LA VERNE W. SPRING.†

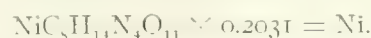
The nickel-zinc separation has always been a serious matter for the chemist who had to do it only occasionally and nearly as forbidding for him who was so unfortunate as to encounter it regularly. It was one of those things from which he was praying to be spared, but which he felt he could sometimes do little to avoid.

Some of the modifications of the sulphide separation can be made to work but it is no method for a busy laboratory or for inexperienced hands. The fixed alkali hydroxide method is open to objections. The titration of the nickel with potassium cyanide in the presence of sodium pyrophosphate works at least approximately under the proper restrictions, but does not allow of a direct determination of the zinc.

Since the announcement during the past three or four years of the German methods for direct precipitation of nickel as nickel glyoxime or as nickel di-cyandiamidine from solutions of almost any other metals, our own troubles have ceased quite completely. The analysis of German silver means now little more than analysis of a brass and the new methods could be used advantageously by many more chemists than appear to be using them. I do not claim originality in the method as used in our laboratory for the past

two years, for though I do not know that certain parts of the method have ever been published, they have, in a way, been forecast by the German chemists, Grossmann, Brunck, *et al.*, in articles in the German chemical magazines. The method is as follows:

Weigh out one-half gram of the drillings, or more, according to the percentage of the constituents, take out the tin with nitric acid, the lead as sulphate, copper by electrolysis and iron with ammonia as usual. Add five grams of ammonium chloride. Make the solution just neutral with hydrochloric acid, and add four-tenths of a gram of dimethylglyoxime dissolved in a little alcohol for every tenth of a gram of nickel supposed to be present. Add ammonia, drop by drop, until the solution smells slightly ammoniacal. Let stand just below the boiling point for one-half hour (more or less according as the nickel separates completely). Filter hot through counterpoised filters or on a weighed Gooch, wash with hot water and dry in the air bath at 105° C. to constant weight. Weigh and calculate the nickel.



Until a little experience has been acquired, the filtrate should be tested to make sure that no nickel has escaped precipitation. Add 0.05 gram more of the dimethylglyoxime in a little alcohol, see that the solution is just ammoniacal and let stand for a few minutes. There should be no reddening of the solution which would mean more nickel. Make the filtrate just acid with concentrated hydrochloric acid and add an excess of ten cubic centimeters. Boil for ten minutes to break up the dimethylglyoxime in the solution, add ten grams of microcosmic salt dissolved in a little water, neutralize with ammonia and hydrochloric or acetic acid until the solution neither turns blue litmus paper red nor red paper blue, and let stand just below the boiling point (do not let bump) until the precipitate has become granular. Filter under suction, wash with hot water, ignite in a weighed porcelain crucible, cool and weigh as zinc pyrophosphate. $\text{Zn}_2\text{P}_2\text{O}_7 \times 0.42913 = \text{Zn}.$

Dimethylglyoxime can be bought for \$2.50 per ounce.

In both the beet and cane industry the complete utilization of by-products will be the determining factor in the matter of profits. The utilization of cane fiber for paper making, the extraction of cane-wax from the sugar house residues, the manufacture of potash salts, ammonia, cyanide and other chemicals from spent beet molasses, improvements in the manufacture of cattle foods from waste molasses and exhausted chips, utilization of press cake for fertilizers, for burning cement and in other ways, improvements in desaccharifying molasses, manufacture of denatured alcohol from waste molasses; these are a few of the lines along which cane sugar and beet sugar chemists are now working to strengthen their respective industries and place them upon a better financial basis.

*From the Journal of Industrial and Engineering Chemistry.
†Crane Company Laboratory, Chicago, Ill.

METALLURGY.

THE ELECTRIC FURNACE FOR THE MANUFACTURE OF IRON AND STEEL.*

By JAMES LYMAN.

The recent development of the electric furnace for the manufacture of iron and steel registers a distinct advance in the art. We may expect its rapid introduction particularly as a refining furnace.

It is therefore well at this joint meeting of our two societies to briefly consider the features of this new process of making iron and steel that are superior to the old processes. More and more the manufacture of iron and steel is becoming highly scientific. The analysis of the ore or the iron treated and the flux and fuel ingredients is made at the start and from time to time until metal of the chemical characteristics desired is obtained. All losses of heat, material and labor are reduced to a minimum and the maximum output of furnace is sought. Where the product is to be rolled or forged the greatest care is used to produce the most dense and uniform grain of the metal.

Highly skilled chemists, mechanical and electrical engineers have wonderfully improved and increased the output of our iron and steel mills in recent years. They have probably developed the present process to very nearly their limit of perfection. There are, however, certain inherent characteristics of these processes that prevent obtaining the very highest qualities of iron and steel. For instance, these processes depend upon various fuels as the source of heat and during the treatment for the elimination of harmful ingredients, certain foreign impurities, such as nitrogen and oxidation products enter the bath injuring in a measure the desired qualities of the metal. Further, there is not always a perfect homogeneity and soundness of metal throughout the entire charge, and either the chemical analysis of different charges or the physical properties of these charges or both may differ perceptibly.

Many metals, including iron and steel, are refined by melting them in contact with a slag composed of ingredients which will absorb their impurities. For the most effective action the slag must be fluid only at the highest temperatures. The electric furnace is capable of creating the highest known temperatures, much higher than any temperatures of combustion and therefore producing the greatest fluidity of flux and metal and this without introducing air or fuel, but on the contrary maintaining a perfectly neutral atmosphere above the metal and slag. While the high fluidity of metal and slag conduce to the most rapid, uniform

and chemically perfect action on the part of the absorbing flux, no oxidizing products, nitrogen or products of combustion can enter the metal and the whole operation can be continued as long as the operator desires; introducing or removing any ingredients and in exact percentages.

Furthermore, irons and steels of specifications which have been obtained with difficulty or never have been obtained in any of the standard processes are readily obtained in the electric furnace. For instance, nickel steel which Mr. Schwab is quoted as stating will be the steel for rails in the future, can be made directly, without difficulty, in the electric furnace. The superior control of the electric furnace over all furnaces depending in their action on heat of combustion especially commends its use in the refining of high grade irons and steels.

The extent to which the electric furnace will be generally introduced depends on the cost of operating the electric furnaces and the superiority of the product as compared with that of the combustion furnaces. The cost of electric heat obviously depends upon the cost of power from water or other sources, as compared to the cost of fuel at the locality under discussion. Further, electric energy may be supplied from water power or from blast furnace gas at a cost which, when the efficiency of the electric furnace is considered, compares very favorably with the cost of high grade fuels such as charcoal and producer gas.

Prof. Richards has pointed out in his excellent paper on "Electric Furnace Reduction of Iron Ore" presented at Niagara Falls in 1909, that the amount of fuel used in a blast furnace is determined by the amount which must be burned at the tuyeres to produce the necessary smelting temperature and not by the amount necessary to perform the reduction of the metallic oxides. The amount necessary for performing the reduction taking place in the furnace is only $\frac{1}{3}$ to $\frac{1}{2}$ of the amount necessarily burned to provide smelting heat, and he is confident that any practical method of introducing electrical heat into the crucible of a blast furnace will result in large economies in the furnace working. Only $\frac{1}{4}$ of the heating power of the fuel is developed around the blast tuyeres and yet if half of this could be replaced by electrically generated heat, an economy of 50 per cent could in all probability be reached upon the fuel bill. It takes 1.2 tons of coke to make a ton of pig iron in the blast furnace and about $\frac{3}{4}$ of a ton is burnt by the blast, producing the melting zone; about 25 per cent of the calorific power of the coke. If one-half this smelting heat or $12\frac{1}{2}$ per cent of the actual heat of combustion could be sup-

*Paper presented at the joint meeting of the Chicago branches of the American Electrochemical Society and the American Chemical Society, January 20, 1911.

plied by electric energy the coke required could be reduced one-half or .6 tons per ton of pig iron. The quality of the pig iron would also undoubtedly be greatly improved and it is probable the output would be increased.

TYPES OF FURNACES.

There are two general types of furnaces: The first called arc furnaces, because the heat is developed in an arc established between two large carbon or graphite electrodes or between these electrodes through the slag and metal bath. The second called the induction furnace because the melted metal forms a single turn closed circuited secondary of a large transformer. The arc furnace is adapted to both the reduction of iron ore to pig iron and also to the refining of pig iron and steel. The induction furnace is only adapted to the refining of metals and cannot be used for the reduction of ore.

Among the many designs of these two general types of furnaces I will refer briefly to the following:



Fig. 1—Frick's Electric Ore Reduction Furnace.

The Frick Electric Reduction Arc Furnace for Smelting Iron Ore.—The capacity is approximately 25 tons of pig iron per 24 hours; current taken 2,000 kw. or approximately 1 kw. hour per pound of pig iron produced. Where the iron ore used is of not less than 57 per cent Fe the theoretical power required for the reduction of the iron in the presence of carbon and unslacked lime is from 1,200 to 1,350 kw. hours per ton exclusive of the heat absorbed by the slag. This heat is from 100 to 300 kw. hours per ton of the pig iron, making a total of from 1,300 to 1,650 kw. hours per ton provided burnt limestone is used.

The radiation losses occur partly through the walls, partly through the electrodes. In the case at hand the radiation losses are said to be from 320 to 370 kw. hours. The electrical efficiency of this furnace is therefore approximately 80 per cent. Smaller furnaces have larger radiation losses and lower efficiency.

The Frick Induction Type Furnace for Refining Iron and Steel.—This furnace consists of:

A ring shaped crucible of uniform cross section holds the melted metal forming the secondary winding of the transformer.

A magnetic core built of laminated iron forming a closed magnetic circuit around the coils.

Two primary coils of insulated copper ribbon mounted one above and one below the crucible on the mag-



Fig. 2—Sectional View Frick's Induction Furnace With Stationary Cover.

netic core. These coils may be wound for any desired voltage up to, say, 6,600 volts. Five thousand volts I believe is used in the furnace shown in the illustration. This furnace was built by Mr. Frick for the Messrs. Fried Krupp A. G. in Essen, Germany, last January, and has been in successful operation since, refining approximately 20 tons of steel per day from cold scrap in three charges in about 6½ tons per charge of 6½ hours duration on charge. The power required is approximately 1 kw. hour for three pounds of steel refined, or 665 kw. hours per ton. To obtain a reasonably high power factor a special low frequency alternating current of from 5 to 15 cycles is used. An engine driven generator designed to furnish this low frequency single phase current is employed. The electrical power supplied is controlled by regulating the generator voltage. The electric efficiency of this furnace is said to be about 65 per cent.

The power consumed in melting the material (cold iron and slag) to say 1,500° C. in a 10-ton furnace is approximately 600 kw. hours per ton or .3 kw. hours per pound. The power consumed for refining the steel after melting is from 1,800 to 2,000 kw. hours for a 10-ton charge, or approximately .1 kw. hours per pound. Therefore, only where cheap power is available can the electric furnace be commercially used to melt cold material. The melting down can be effected more economically in the open hearth or other combustion furnace.

It is estimated that in a well designed 15-ton, 2,000-kw. induction furnace the total thermal efficiency from coal or gas would be approximately as follows:

Gas engines, say 20 per cent.

Electric generator, say 94 per cent.

Electric furnace, say 80 per cent.

Total efficiency, 15 per cent.

If steam turbine generators are used at, say, one-half the cost for fixed charges over the gas engine installation and, say, 15 per cent efficiency from the coal or blast furnace gas; the total thermal efficiency of the electric furnace would be, say 11 per cent.

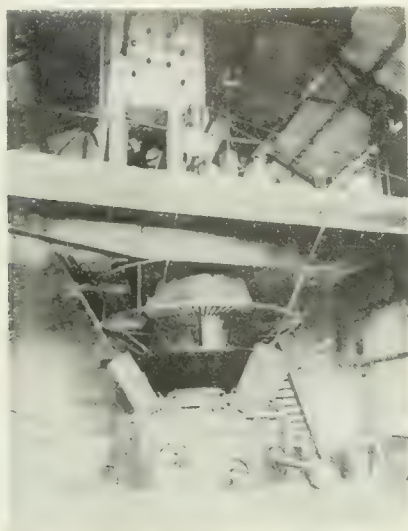


Fig. 3—Heroult Reduction Electric Furnace, Noble Electric Steel Co., Heroult, Cal.

The efficiency of an open-hearth furnace is said to be from 20 to 25 per cent. In this case the fixed charges and maintenance of the gas producers partly offset the turbines or gas engines. It is therefore, apparent that the cost of refining melted pig iron or steel in the elec-

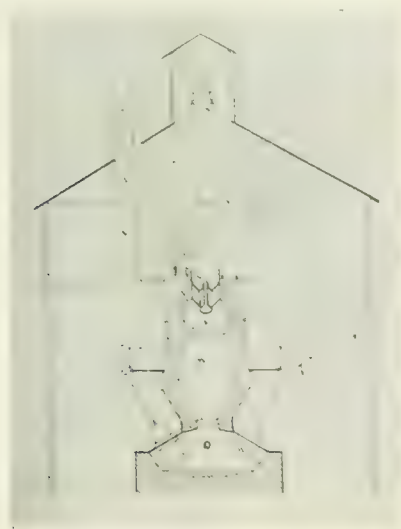


Fig. 4—Heroult Reduction Electric Furnace, Noble Electric Steel Co., Heroult, Cal.

tric furnace, while considerably greater than by the open hearth, is not excessive.

The Heroult Furnace.—The most notable installation of the Heroult arc furnace for the reduction of iron ore to pig iron is that of the Noble Electric Steel Co., at Heroult, Cal. In 1907 the first large furnace was built taking 1,500 kw. of power, but not proving entirely successful this furnace was replaced with a smaller experimental furnace of 160 kw. capacity. From the

experience obtained with this furnace a second large furnace was built early in 1909, which has been in almost continuous service, making 25 tons of pig iron per day of a quality comparable with the best Swedish kind. Four more furnaces of the same size are now being built, which will make this the largest plant of the kind in the world.

A very pure magnetite ore of 70 per cent Fe and a high-grade limestone are quarried near by. The company make their own charcoal and purchase hydro-electric power from the Northern California Power Co. Three 750-kw. oil insulated water cooled, 60-cycle transformers wound for 2,200-volt primary and a secondary range of 35 to 75 volts. The secondary current varies from 10,000 to 21,400 amperes. The range in voltage is controlled by a dial switch in the primary

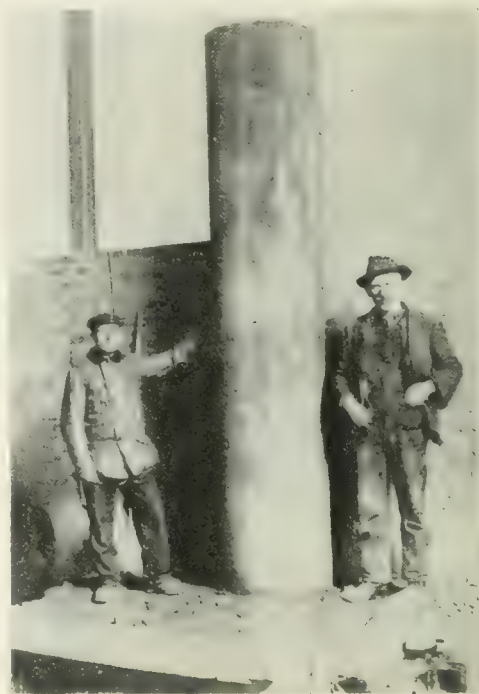


Fig. 5—Carbon Electrode.

circuit of each transformer, giving steps of about three volts in the secondary. The generation of energy in the furnace is therefore, controlled externally without movement of the electrodes whose position is changed only to accommodate their wear in the crucible of the furnace. There are six electrodes, two for each transformer. While the primaries of the transformer are connected to the three-phase power line, the secondaries are not interconnected.

There are some 30 Heroult arc furnaces for the refining of steel in service in Europe and America in sizes from 1 to 15 tons per charge. One of the largest of these is installed in the South Chicago works of the Illinois Steel Co. This furnace has been in regular service for more than a year refining Bessemer steel to a wide variety of specifications. Power is supplied from three-phase, 25-cycle, 2,200-volt circuit. The furnace has three electrodes connected delta to the secondaries of the transformers. They are each counter-

weighted and are raised or lowered by a motor driven mechanism which is either automatic or hand controlled.

Kjellen and Röchling-Rodenhauser Furnaces.—The Kjellen and Röchling-Rodenhauser furnaces are the most noted induction furnaces. The first one was built by Mr. Kjellen in February, 1900, in Cysinge, Sweden. Thirty of these furnaces are now in operation in differ-

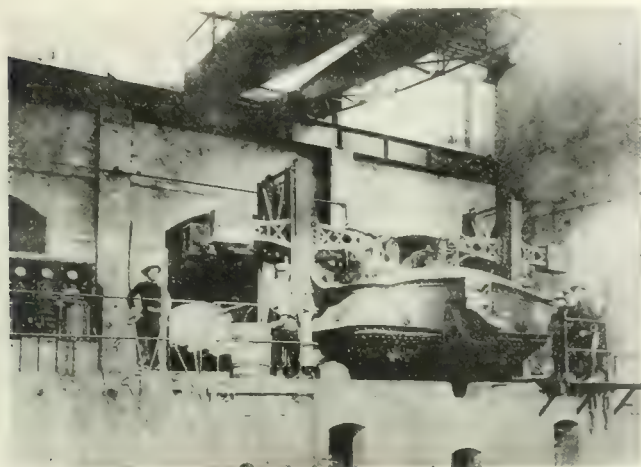


Fig. 6—Kjellin Arc Refining Furnace.

ent parts of Europe. Sectional and full views of one of these furnaces is shown herewith. Electric heat is not only supplied through the metal bath acting as a short-circuited secondary but also from a special secondary winding which supplies current to metallic plates imbedded in a conducting mixture of refractory material which forms a part of the lining of the furnace. The current from these plates passes through

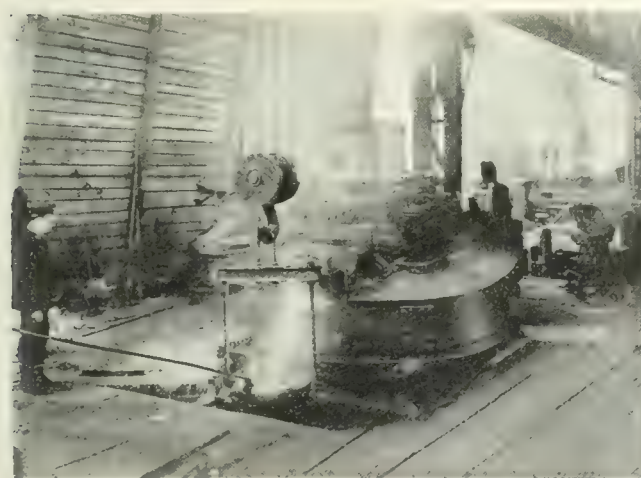


Fig. 7—Röchling-Rodenhauser 3-Phase Electric Induction Furnace.

the lining, then through the melted metal in the main hearth of the furnace. This design greatly improved the action and raised the power factor of the current taken by the furnace.

The demand for strictly high-grade steel, absolutely homogeneous and with great fineness of grain, is to-

day coming from the railroads for steel rails and for structural bridge steel, from the government for armament, from the automobile industry, from every manufacturer of tools and machinery, engines, steam tur-

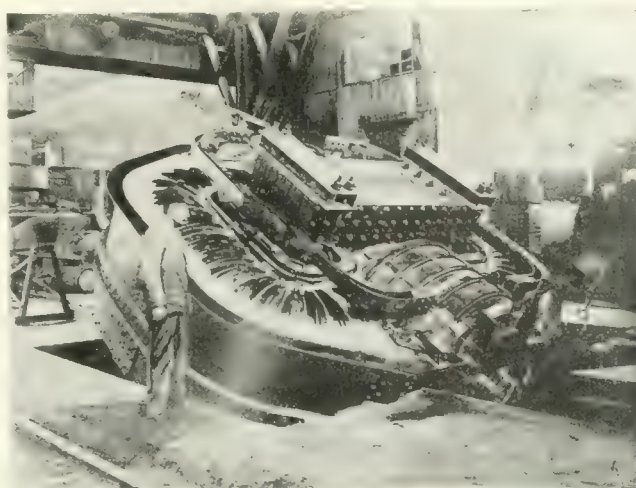


Fig. 8—Röchling-Rodenhauser Single Phase Electric Induction Furnace.

bines, electrical manufacturers, and indeed every manufacturer using iron or steel. The treatment of Bessemer and open-hearth steel in the electric furnace, at an increased cost entirely within the limits of the purchaser, will make this steel comparable in its fineness with crucible steel and have the physical characteristics best adapted to the particular application desired. Since the railroads have investigated the causes

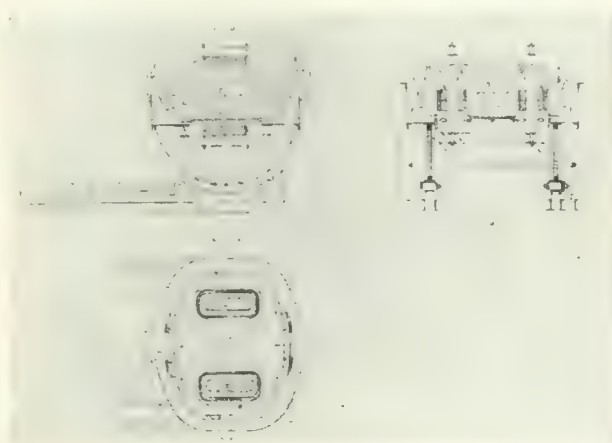


Fig. 9—Sectional View of Röchling-Rodenhauser Single Phase Electric Induction Furnace.

of rail breakages it has been proved that many of them were due to the presence of foreign injurious bodies, such as slag, manganese, sulphides, etc. The possible presence of these impurities as well as the products of oxidation and nitrogen is inherent in the Bessemer and open hearth furnaces but can be almost entirely eliminated by a treatment of an hour or two in the electric furnace.

Rails from electric furnace steel are now being tried out on curves, railway crossings and points where the

service is most severe by a number of the large railway systems. These rails combine unusual tensile strength, toughness and hardness. Their life alone, aside from their increased reliability, will probably justify the increased cost.

To bring the enormous output of steel rails, structural, merchant and plate steel up to the high grade of crucible will mean a new era for the steel business and rapid advances in all lines of manufacture employing iron and steel.

The manufacture of modern and light weight steel casting has always been a difficult and unsatisfactory problem. There is a large wastage particularly in castings of odd shapes. This is principally due to impurities and sluggishness in the flow of metal in castings. With less than one-third the electric power necessary for purification, the liquid metal can be held indefinitely in an electric furnace without any loss in its composition or danger of burning and a highly liquid metal can be cast uniformly free from impurities and gases. Such steel castings can in a large measure replace steel forgings at present used, at a much reduced price. While the large steel companies will no doubt introduce electric furnaces as a refining process for their principal output, the electric furnace can probably be used to advantage by manufacturers of all

chinery in the steel plants and the hoists in the mines, in such instances working a wonderful transformation, so in entering this new field we may anticipate equally

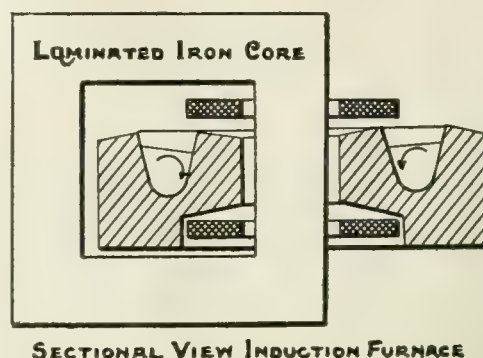


Fig. 11.

marked advances and a very great demand for electric power and electrical apparatus.

The Hongkong Chamber of Commerce has taken action with respect to the requirement of the Japanese Government that the recipes of all proprietary medicines entering Formosa shall be turned over to the Government. Practically the whole of the import drug trade of Formosa in the past has been handled by Hongkong concerns. Of late years Japanese houses and foreign houses in Japan have been securing more or less of the business. The enforcement of the rule referred to is regarded as a marked disadvantage to Hongkong merchants and American and other foreign houses dealing through them.

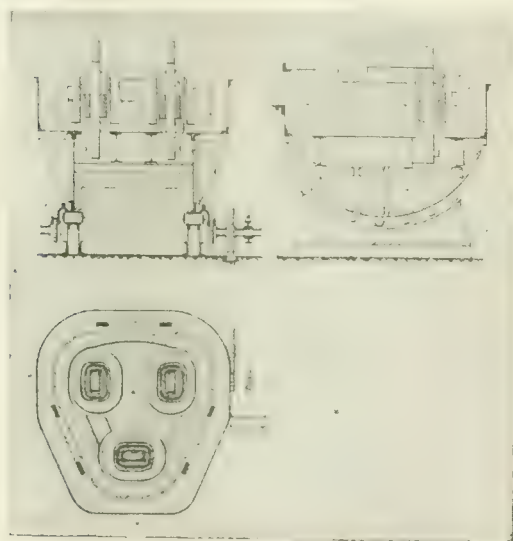


Fig. 10—Sectional View of Kjellin Röchling-Rodenhauser 3-Phase Induction Refining Furnace.

kind of iron and steel products in making special high grade steels from their waste scrap iron and steel, including borings and turnings, as they accumulate in process of manufacture. These furnaces will either be entirely electric or the metal may be brought to melting point by gas or coke fuel and then treated by electric heat.

As electricity has revolutionized our means of communication through the telegraph and telephone, displaced gas and other illuminants for lighting, supplanted the horse on the tram car and mechanical drive in industrial plants, even driving the rolling mill ma-

A recent consular report states that a British invention of interest to users of automobiles and motor cycles has recently been brought out for the purpose of removing carbon deposits from the cylinders of internal-combustion engines without removing the cylinders. The amount of carbon deposit which adheres to the cylinder walls, piston heads, and valves is considerable, particularly in air-cooled engines. The invention does away with the necessity of dismantling the engine or disturbing any of the connections, and it is claimed that the decarbonizing is done quickly at trifling expense. The apparatus consists of a cylinder of pure oxygen, to which is fitted a pressure-reducing valve with a flexible tube and blowpipe connected, and a small petrol hand lamp with a long nozzle and wick, which is used for starting. The process, after the valve caps have been removed, consists of inserting the oxygen blowpipe and the lamp nozzle through the openings into the cylinder head and allowing the flame to impinge on the carbon, which immediately becomes incandescent and comes away in the form of light sparks. It is considered advantageous to warm up the engine prior to commencing operations, this having a tendency to soften the carbon slightly, making removal easier.

AN ELECTRIC FURNACE FOR ZINC SMELTING.*

By FRANCIS A. J. FITZGERALD.†

There is no branch of metallurgy which is apparently more suited to electric furnace treatment than that of zinc smelting. The regular method of zinc smelting is extraordinary in its crudity, inefficiency and expense, hence the relatively high cost of heat generated electrically is not by any means so serious a consideration as in certain other metallurgical processes. Moreover the electric furnace possesses certain characteristics which make it specially applicable to the conditions of zinc smelting. In the following paper it is proposed to describe briefly a new form of electric furnace originally designed for zinc smelting, although it has useful applications in other kinds of work.

It is not intended to discuss here the metallurgy of zinc smelting, but to appreciate properly the electric furnace which will be described it will be necessary first briefly to consider the particular method of zinc

process in the ordinary zinc retort furnace would not be satisfactory, for the process should be carried out with a much larger unit than a zinc retort. When it comes to applying fuel heat to such a process numerous difficulties arise which are sufficiently plain without mentioning them in detail. This naturally led to the idea of using an electric furnace and many experiments with various kinds were made. Finally Mr.

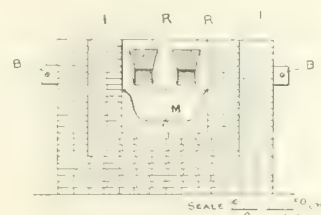


Fig. 2.

John Thomson and the author designed a furnace which was used on a large scale in the working of the Imbert process. One of these furnaces of 150-kw. capacity was constructed and worked under the author's supervision in Hohenlohehutte, Upper Silesia, with highly satisfactory results.

In order to design a satisfactory furnace it was necessary to keep certain points in view: The furnace must be gas tight; the temperature must admit of careful regulation; the construction must be rugged so as to stand severe usage; the heat losses must be reduced to a minimum since electrically generated heat is always expensive.

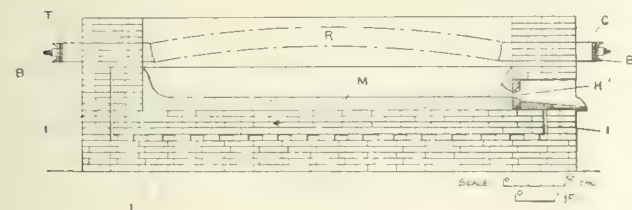


Fig. 1.

production for which the furnace was designed. It has long been known that when zinc sulphide and metallic iron are strongly heated the following reaction takes place:



but the reaction does not seem to be complete unless there is a relatively large excess of iron, or unless the temperature of the reaction is very high. Imbert, however, discovered¹ that by using suitable "dissolvents" this objection to the process is overcome. Imbert, for example, found that ferric oxide and iron sulphide mixed together in the proportion of one part and three parts respectively formed a very fluid bath at a temperature between 1,000° and 1,100° C., and that this bath would "dissolve" six parts of blende. Now when the blende is "dissolved" in a bath in this way the reaction with iron mentioned above takes place with the greatest ease, is complete, works at a comparatively low temperature and as a residue produces two distinct substances: A slag consisting of the gangue from the ore and a ferrous matte which may be used for the regeneration of iron, etc.

A great many experiments were made with this process and the results were highly satisfactory, except that it was very difficult to construct a suitable furnace for the purpose. Obviously working the

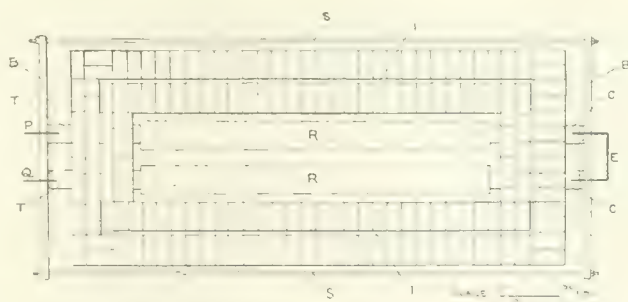


Fig. 3.

In Figs. 1, 2 and 3 are shown respectively a longitudinal section, transverse section and plan of the furnace with the cover removed. The walls of the furnace are double with air-spaces, I, which are designed to prevent the loss of heat by conduction through the walls. The furnace is provided with carbons, T, T, C and C. The two former serving as terminals which are connected to the source of current by means of cable indicated by P and Q, while the two latter are simply connector terminals which form the other terminals of the two sections of the resistor, R, R, and are connected by E. Bearing on the terminals, T, T, and the connector terminals, C, C, are channels, B, B, which are connected with each other by the tension rods, S, S. The channels are, of course, insulated from the terminals. The furnace is lined with a suit-

*Paper presented before the Congress of Technology at the 50th Anniversary of the Granting of the Charter of the Massachusetts Institute of Technology.

†Consulting Chemical Engineer, Niagara Falls, N. Y.

¹U. S. Patent 875,579; Dec. 31, 1907.

able refractory, M, and is provided with a tap-hole at H. The resistor of the furnace is built up of a series of corrugated plates, which are illustrated in Fig. 4; the lower sketch showing an end view of the plate, while that above shows the shape in which the plates are cut as viewed from the front. In Fig. 5 is shown a view of the plates set up so as to form a resistor. Considering one of the plates it is to be noted that the thickness is not the same from top to bottom, but increases from the bottom up so that when put in place they form an arch of very long radius, as

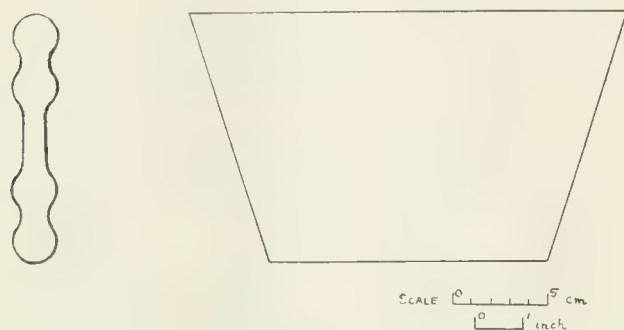


Fig. 4.

shown in Fig. 5. Because of the interlocking of the plates this arch form is not necessary, but seems to be desirable in the preliminary assemblage and is also utilized to produce a somewhat greater current density along the lower surface of the resistor. The cover of the furnace, which is not shown in the illustration, carries feeding tubes by means of which the ore mixture may be fed into the bath below the resistor.

The peculiar construction of the resistor plates has two purposes: To give a sufficiently high resistance to the resistor and at the same time to form an interlocking device so that even if no arch form is given to the resistor yet it will not fall down. A furnace was built with plates having these dimensions:

Length at top.....	405 mm. 16 ins.
Length at bottom.....	255 mm. 10 ins.
Width	165 mm. 6.5 ins.

The two sections of the resistor contained 71 plates each. This when cold had a resistance of 0.200 ohm and when running at the full capacity of 150 kw., and with a temperature in the furnace of 1,400° C., the resistance was 0.0375 ohm. This resistance is due almost altogether to the contact resistance between the plates, for by calculating the resistance of the carbon itself we find that it would not amount to more than 0.00064 ohm.

In order to regulate the rate of generation of energy in the resistor there must be some means of varying the voltage at the terminals of the furnace. At Hohenloheutte, as well as in the Fitzgerald and Bennie laboratories where these furnaces have been worked, this is done by means of a transformer with several taps brought out from the primary coils which allow the voltage on the secondary circuit to be varied from 50 to 100 volts in 2.5-volt steps, and from 100 to 200 volts in 5-volt steps.

It will be seen that the weakest part in this furnace is the carbon resistor, due to the fact that if working in an oxidizing atmosphere the resistor will be destroyed. In the particular work for which it was designed, however, there would be no danger of this because the furnace is filled with vapor of metallic zinc. During the process of heating the furnace, or at any time when zinc vapors were not generated, there would be danger of burning through air leaking in; but this is easily overcome by keeping a reducing atmosphere in the furnace slightly above external pressure. It has been found by actual experiment that a furnace of this type running continuously for two months showed no appreciable wear of the resistor.

The regulation of temperature in this furnace is most satisfactory. In the Hohenloheutte experiments thermo couples of pyrometers were placed in several parts of the furnace to study the temperature conditions carefully. It was found that the most accurate regulation of the temperature in the furnace was possible, the workman in charge adjusting the rate of generation of energy in the resistor so as to keep the needle of the pyrometer stationary.

The furnace is a highly efficient one. In one of the earlier models where the heat insulation was far from being satisfactory careful determinations of all heat losses were made. When working at temperatures between 1,250° C. and 1,260° C. the total heat losses were 33 kw., and when working at temperatures between 1,400° C. and 1,450° C. the heat losses were 42 kw. Consequently when the furnace is work-

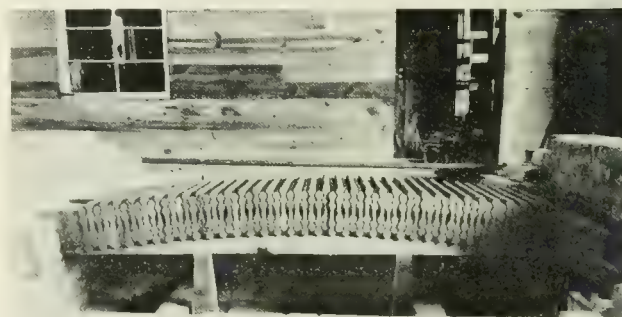


Fig. 5.

ing at full capacity, 150 kw., the thermal efficiency at 1,250° C. is 78 per cent and at 1,425° C. is 72 per cent. No exact determinations of the efficiency of later models have been made, but it is known to be much higher than those given above.

The metallurgical end of the problem has not been completely worked out, but the satisfactory working of the furnace has been clearly demonstrated, and furnaces built on similar principles have been used experimentally with great success in the melting of aluminum, copper, brass, etc. This is thought to be of some interest, as a development in the use of electric furnaces using the heat generated by the passage of an electric current through a resistor. There is a

tendency in electric furnace work to employ the arc which is often a mistake because of the difficulty in regulating the temperature. Finally, the furnace described above from its construction lends itself readily to adaptations which permit of using the combined heat effects of fuel and electricity, and it is thought that a great future is in store for furnaces of that type.

TITANIUM ALLOYS IN IRON AND STEEL.*

By CHARLES V. SLOCUM.

TITANIUM IN IRON.

When the manufacture of titanium in the shape of an alloy was undertaken on a commercial scale in 1907, the results in primary tests were considered for a time as the basis upon which it would be necessary to place this material on the market. When a number of trials had been made, however, it was found that different results were apparently secured and it then became necessary to modify these original ideas in accordance with these developments and in fact since three years is not a long time to learn everything in regard to a new element, this modification is still on.

When it is remembered that the output of titanium alloy exceeds that of all other alloys combined above the grade of manganese and silicon, and that this development has been made in such an unusually short period, it is safe to assume that as time goes on the results will be still more developed, percentages changed for certain purposes and the whole situation become more definitely systematized and made to conform more readily to the methods and ideas which time and experience can alone develop to the fullest extent.

It was an idea at first in using titanium in iron that since comparatively large percentages gave increased strength in tensile and transverse tests, that these larger percentages were essential to the process of securing a titanium reaction in the metal and therefore must be used. Gradually, however, it became evident that minute percentages of titanium were successful in producing many of the benefits of a larger quantity. As for instance the use of two-tenths of one per cent of alloy which contains only 0.02 of metallic titanium is frequently sufficient for the purpose desired, which consists principally of cleaning the iron of impurities remaining in the metal after the ordinary fluxes have accomplished all that they can in this direction. This cleansing is not one which is possible with limestone or with fluorspar, for these fluxes have no effect whatever upon the oxides or nitrides containing in the molten iron.

Titanium, on the other hand, removes both of these objectionable elements partially in any event and wholly or practically so when sufficient alloy is used

for the purpose. The effect then of the small fractions of a per cent in iron is, more practically speaking, to increase the fluidity of the iron and at the same time to make the metal more homogeneous, so that the resulting castings are closer grained, are free from pin holes and gas bubbles or blow holes, the iron has a denser structure and at the same time is easily machined.

The shrinkage is less in the treated iron than in the plain or untreated casting. This is particularly true in hard or chilled iron, where heavy castings, such as car wheels, will shrink an eighth of an inch less or one whole tape number in circumference.

Much finer work may therefore be done with iron treated with the alloy, since the castings will conform more nearly to the original pattern and will have a more uniform shrinkage than the plain iron.

In a letter dated April 5, 1911, the writer was advised by Mr. Asa W. Whitney, metallurgist of the Enterprise Foundry and Machine Company of Bristol, Va., that he has made a number of careful trials of titanium in hard or chilling iron. He finds that 0.1 to 0.2 per cent of the alloy is usually all that is necessary to make otherwise viscous, high chilling mixtures come from the cupola close grained and pour from the ladle as freely as iron carrying half as much chill and of more open grade. The iron pours well to the last and gives clean solid castings. "I find," Mr. Whitney says, "that I am able to compensate for the cost of the titanium with less manganese and a trifle less silicon."

To foundrymen the fact that the metal remains hot longer than untreated iron is a matter of much importance for certain classes of work, since the greater fluidity means that the iron will settle more slowly in the mold, and thus give time for the gases to escape and more time for the iron to fill every smallest outline of the mold without pulling away from the main body of the casting. These features of good foundry practice are some of the ones which are often overlooked, and a high percentage of bad work results. Now in relation to the benefits to be expected from using larger proportions of alloy, I maintain that a great deal of undue importance is often attributed to an increase in transverse and tensile strength. This may be necessary for certain government work and for some few castings which must resist unusual stresses in service, and for such it is necessary to use at least one per cent of titanium alloy. . . .

To summarize, it may be said that titanium alloy is of great benefit, in small percentages, for fluidity, for sharper castings; for remarkable uniformity and for durability while the larger percentages are for strength and density of metal in addition.

I realize of course that certain classes of work do not justify an increase of cost to any extent worth mentioning, and yet even in cheap work the use of a pound or two of alloy per 1,000 pounds of iron is a distinct benefit to the metal and costs but 25 to 50 cents

*From the Industrial World. Condensed report of two papers read before American Foundrymen's Association, Pittsburgh meeting, May 24, 1911.

per ton of product. Other classes which will perhaps bear a little more expense may be increasingly benefited with three more pounds per 1,000.

One important feature of titanium in iron should not be overlooked, viz.: the great durability of all metal into which it enters. In some tests made by Prof. M. Hokanson at the Carnegie Technical Schools some two years ago the following results were obtained:

Chilled test blocks machined down to approximately $\frac{1}{2}$ inch x $\frac{1}{2}$ inch x 1 inch were used.

First Test.—Under crushing strain, blocks from plain untreated iron developed 173,000 pounds per square inch at failure. Metal from same tap of cupola, but treated with one per cent of titanium alloy developed 298,000 pounds per square inch at failure.

Second Test.—Chilled test blocks from a different foundry and made from plain iron, machined as were the others previously mentioned, developed 200,000 pounds per square inch crushing strength at failure, while similar pieces from same tap of cupola but treated with one per cent titanium alloy, developed 392,000 lbs. per square inch at failure. It seems to be this quality of burden bearing imparted by titanium which has made such a demand by railroads for its use in steel rails.

The feature of greater fluidity cannot be emphasized too strongly. When we seek for increased fluidity through the addition of ordinary fluxes the result is often disastrous, for much fluxing makes the metal boil and blow holes, pin holes and defective castings generally result, together with increased expense in the repairs or even in actual relining of the cupola, sometimes necessary from this method of procedure.

In a recent trial of titanium alloy in the cupola with a mixture of burnt iron, stove plate, etc., in a large foundry in the Pittsburgh district the rather large percentage (for iron) of half of one per cent alloy was used together with one-half of one per cent of ferromanganese of the usual 80 per cent grade. This trial demonstrated the interesting fact that the poorest mix which can be imagined, perhaps, may be brought to good normal number one iron by this method. The average of all the test bars, of which there were ten, was 3,100 pounds breaking strain and an average deflection of .146 inch.

Under ordinary conditions, with the addition of the manganese only, the strength would scarcely have reached 2,000 pounds per square inch.

The name of the plant where these castings were made will be cheerfully given to anyone desiring to make the same kind of an experiment. It should be remembered that burnt iron is usually high in sulphur and the iron in these castings contained .151 and .147 sulphur respectively by two different analyses. Such a remarkable strength from such remarkably poor iron is unobtainable at anything like the small cost by any other process, since the expense of making the metal homogeneous, of excellent wearing quality, etc., was

about \$1.50 per ton of castings. The iron before treatment could scarcely have been given away for ordinary purposes and was possibly worth \$10 per ton. The complete analysis of the treated iron by two separate determinations was as follows:

Silicon.	C.C.	G.C.	Phos.	Sul.	Mang.
1.30	.70	2.50	.504	.151	.71
1.30	.74	2.42	.455	.147	.79
Average of 10 test bars, 3,100 pounds.					
Minimum, 2,900 pounds.					
Maximum, 3,265 pounds.					

Not to be misunderstood, I desire to repeat in brief that high sulphur iron may be made strong and available for practically all purposes by an expenditure as above of \$1.50 or less per ton of metal melted.

Silicon	1.60
Sulphur	.011
Phosphorus	.055
Manganese	.050

One-fourth of one per cent of titanium alloy was added in the cupola and the foundry foreman reported immediate benefit in the fluidity of the metal and in the distinct improvement to the castings. This rejuvenation of the iron, so to speak, involved an extra cost of 62½ cents per net ton of metal treated, and was more than repaid in the reduction in bad work without regard to the benefit to the iron. Castings from this iron, with fine grain and good metal, were made which were only 5/16 inch x ¼ inch section.

As a substitute for charcoal, titanium is making much progress. In a letter dated January 9, 1911, the secretary of the American Foundrymen's Association, Dr. R. Moldenke, states as follows:

"I am getting more and more confirmation that by the use of titanium as one of the most oxidized elements I know of, we will eventually make a common coke iron equally as good for our purposes as the best cold blast charcoal iron of the same composition."

In a letter dated December 3, 1910, Prof. John Porter, of the University of Cincinnati, gives expression to the following idea of the merits of titanium as a means of securing better castings without regard to the mere question of strength, which as a rule takes care of itself in a well-regulated foundry:

"I believe most thoroughly in the great value of titanium in cast iron. I know that it will remove many of the causes of bad castings and feel sure that many foundries can save money by using it, in addition to obtaining a better quality of castings. I have several times seen this demonstrated."

Now there are a few suggestions which occur to the writer and which may have become familiar to others, but it will do no harm to repeat them, viz.:

In using any alloy do not govern the quantity or the method entirely by the mere untested statements of others, but be governed largely by the particular features and practices of the foundry in which the trial is to be made. For instance, one of our best known manufacturers reports a failure of titanium alloy to improve iron when used in a cupola which was not slagged and which therefore kept the titanium in

almost constant contact with the very impurities which should have been kept away from it. Such a poor use of good gray iron should be placed in the dark corner and kept there for all time.

Another questionable procedure is to have a few of the castings looked after carefully by some good man like the foreman or even the superintendent, and then to have the other titanium additions for the same kind of iron and from which the same results are expected, looked after by some laborer or "hunkey" with as much knowledge of the necessities of the case as the city directory has of metallurgy.

The don'ts are almost more necessary in the use of this alloy than important rules of guidance. We put an alloy into an iron or into steel for a distinct purpose, but unless we can at the same time place a little common sense and good foundry practice along with it, the results will be doubtful in the extreme. We can undertake to put a great deal of pressure upon the foreman, but unless he has sufficient foresight to realize, for instance, that silicon although used as a softener, is also at times a great weakener of good iron and therefore should be used as sparingly as possible with titanium (which does all the softening necessary without doing any weakening at all), it would be useless to instruct him as to the merits of any alloy. A mere man is only one of the most obtuse of God's creatures unless he uses the good gray matter with which most men are endowed and which is so necessary in the manufacture of good gray iron.

In conclusion the uses of titanium in iron may be summarized as follows:

Very small percentages, as low sometimes as 1/10 of one per cent (0.1) of alloy (only .01 Ti), are sufficient to cleanse the iron of impurities not touched by limestone or fluorspar, while larger percentages increase the improvement in other directions, making the iron more fluid (usually hotter), making the castings sharper, finer grained, free from blow holes and pin holes and easily machined, while at the same time the expense runs from 25 cents per ton of castings up to say \$3 a maximum, all of which is often repaid in the reduction of bad work and in improving poor irons.

II. TITANIUM IN STEEL.

In steel castings the same principles may be said to hold good as outlined on titanium in iron. In order to secure increased fluidity and ductility of the steel, the small percentages of alloy such as one or two-tenths of a per cent have been used successfully by a number of manufacturers and for a further increase of ductility, density and greatly increased strength, the larger percentages from three-fourths up to one and a half or even two per cent are extensively used.

It must be remembered that the smaller proportions are not sufficient to give the full benefit of which titanium is capable, any more than a small dose of medicine is capable of doing what a more powerful one might be expected to accomplish. When we make

a decided effort to improve any material it is always necessary that some good judgment be used in connection with it. For instance, what good would it do to add a certain percentage of manganese to the slag? And yet the writer has several times seen the usual deoxidizers added so late that a further deoxidation by the use of titanium was wholly impracticable. . . . In a recent trial of the alloy in basic open hearth by one of the large steel makers, it was found that in order to remove the last vestige of brittleness and of blow holes, the use of one and one-half per cent of alloy was necessary. This in turn made the steel so dense that it was unusually strong and became the source of much comment and investigation by all concerned. It is not possible to remove all of the causes of poor castings by the addition of titanium nor with any alloy whatever. Good foundry practice is more necessary in steel than in iron because more harm is likely to result if the castings are not as good as they should be. When the mold is not properly figured and prepared; when the melt is not looked after to prevent overheating, etc., it all comes back to the original statement that other things are necessary as well as the use of alloys, for good steel making.

When we learn that in some mills a sinking head is cast which weighs considerably more than the casting itself and the latter in some instances weighs eight tons, then we see at once that foundry practice has given way to the sink head and it is a wonder the casting itself does not give way as well.

Compare the cost of a nine-ton sinking head to make an eight-ton casting, with the cost of titanium alloy in sufficient quantity to remove the blow holes which the big sinking head removes and it figures out about as follows with castings at three and one-half cents per pound and scrap at three-fourths cents per pound:

9 ton head	2,000x9x.035.....	= \$630.00	
Less scrap value	2,000x9x.0075.....	= 135.00	
			\$495.00
Maximum head of say 40 per cent by using			
Ti	2,000x8x.40x.035.....	= \$224.00	
Cost of 1.00 per cent titanium alloy for 8 ton casting and 40 per cent head	22,400x1.00		
at 12½.....		= 28.00	
			\$252.00
Difference in favor of titanium.....			\$243.00

When we take into consideration the fact that so small a proportion as one-tenth of one per cent of titanium alloy makes a noticeable improvement to steel as vouched for by some of the largest makers in this country and that larger proportions up to say two per cent add materially to the improvement, then it becomes clear that the small cost involved which runs from 25 cents as the minimum up to \$5.60 per gross ton of product as the probable maximum, is something which every steel mill may well take into consideration and begin to save money by reducing their percentages of bad work.

The following proportions have been used regularly in a well-known steel mill where only the best results

are accepted and where the product is said to be equal if not superior to the best stamping steels made in this country:

BASIC OPEN HEARTH STEEL.

Size of heat 120,000 lbs.

	C.	Mn.	P.	S.
Analysis	.10.	.33	.007	.019
Titanium alloy added in ladle, 250 lbs. = 0.21 per cent, or 0.021 Ti.				
Aluminum added, 10 lbs. = .01 per cent.				
Elastic limit, 37,060 lbs. per sq. in.				
Tensile strength, 44,920 lbs. per sq. in.				
(After dead soft anneal.)				
Elongation in 2 ins., 40 per cent.				
Elongation in 4 ins., 28 per cent.				
Reduction of area, 56.4 per cent.				
Fracture, silky.				

In the mills of one of the most scientific steel makers in the United States, steel castings are required of the following specifications:

Elastic limit	45,000 lbs. per sq. in.
Tensile strength	85,000 lbs. per sq. in.
Elongation after rupture	12 per cent.
Contraction of area	18 per cent.

"These specifications have been met and the necessity for numerous heat treatments have been avoided by the use of eight pounds of titanium alloy per ton of metal (— 0.4 of alloy or 0.04 Ti) added in ladle. No aluminum was used."

These parties write as follows:

"Comparing the results of the tests made upon specimens after the first anneal, of the last fifteen heats in which ferro-titanium was used, with those of the last fifteen previous heats in which it was not used, it appears that the mean tensile strength was increased from 81,633 lbs. per sq. in. to 91,533 lbs. per sq. in., that the mean elastic limit was increased from 47,233 lbs. per sq. in. to 50,000 lbs. per sq. in., that the mean elongation was increased from 15.1 per cent to 19.2 per cent and that the mean contraction of area was increased from 18.9 per cent to 24.3 per cent."

In a letter dated April 12, 1911, Mr. J. H. F. Dixon, general manager of the Keystone Steel Casting Co. of Chester, Pa., makes the following comment on the use of titanium alloy in their steel castings:

"The added cost of the production of our metal by the use of this alloy is so slight that we are prepared to furnish genuine crucible castings titanium treated, at the same schedule of prices which apply to our carbon steel. We strongly recommend," he states, "the use of steel so treated for automobile work, where the castings are subject to unusual shocks and where uniformity of the material is absolutely essential."

Of the 45,000 applications for patents made every year to the German Patent Office two-thirds are rejected after correspondence or personal interview with

the applicants. The Patent Office came into existence only 33 years ago, but its operations have now so grown that nearly 300 lawyers, with bureaus and clerks, make a living by representing patentees. In Germany an invention to be granted a patent must have a commercial value and must not have been already patented; more than that, the Patent Office must be satisfied that the invention is going to be of real practical value for some branch of industry which the patent law enumerates. Perhaps no patent office in the world is so strict and searching as the German, and it is claimed that as a result there are fewer fetters on trade, fewer sham patents, and less patent litigation in Germany than in any other country.

Owing to the increasing American interest in the chemical industry of the world, and the desire to secure information concerning its remarkable expansion in Europe, the Bureau of Manufactures of the Department of Commerce and Labor, Washington, D. C., is about to undertake a special investigation along these lines. Practical suggestions are invited from the trade as to the most desirable subjects to be taken up, and as to the most promising lines for commercial expansion. The imports of chemical goods and drugs into the United States are reaching enormous proportions, having been \$90,000,000 in value last year, against \$67,000,000 in 1908, while the exportations of chemical products from the United States have remained almost stationary, the aggregate reaching about \$20,000,000 per annum. The chemical trade is a very broad one, but has special phases well worth study from the commercial aspect. To this end it is thought that correspondence as indicated will prove beneficial.

Recent publications of the International Union for Statistics of the Sugar Industry estimate the production of sugar in the countries belonging to the union at 7,540,000 tons in 1910-11, as compared with 5,780,000 tons in 1909-10, and in the non-union countries, Italy, Denmark, Spain, Switzerland, Roumania, Bulgaria and Servia, 408,200 tons, against 312,000 tons in 1909-10, a total European production of 7,948,200 tons, against 6,092,000 tons in 1909-10, an increase of 1,856,200 tons. Germany retains its first place as a producer, with 2,500,000 tons; Russia comes second with 2,100,000 tons, while Austria-Hungary, which was second in 1909-10, takes third place in 1910-11. France, with a decrease of 90,000 tons in 1910-11, occupies fourth place, and is the only country in the union showing a decrease. Outside the union, Italy and Denmark, with a production of 184,000 and 109,000 tons, respectively, are the only countries of any producing importance. To produce the foregoing amount of sugar in 1910-11 the 1,290 factories of Europe consumed 51,900,000 tons of sugar beets, as compared with 40,600,000 tons in 1909-10.

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NOTES AND COMMENTS.

Chemical Fakes and Business Men.

The value of the expert practical chemist in protecting the otherwise shrewd business men against swindling schemes which have too often proved easily successful, is crisply set forth in a recent article by Arthur D. Little, Chemical Engineer, of Boston.

A curiously large number of gold bricks, says Mr. Little, are gilded by chemical methods invariably applied by amateurs who have nevertheless utilized most ingeniously some few scraps of chemical information. The surprising thing about the industry is the character and quality of its clientele and the psychology of

the selling arguments. Hard-headed business men enriched by dearly won success, who have learned to trust their judgment, and who have demonstrated their capacity for affairs, men who turn a box of strawberries upside down, and require a pastor's certificate of character from the office boy, these are the best prospects. They listen, they witness a demonstration, they calculate profits, and they are lost.

After all, the psychology of the transaction is not so obscure. It is because they have learned to trust their own judgment in the things they know about that they are led to regard it as equally trustworthy in case of something about which they only think they know. They have so long ignored expert assistance in their ordinary affairs that when the extraordinary occasion arises they feel quite competent to cope with it alone.

There is, too, unfortunately, still a certain atmosphere of mystery around the processes of chemistry as viewed by the average mind, which clouds deduction and seems to justify the otherwise unreasonable. "Chemistry accomplishes so many extraordinary things, why not this one which I have seen with my own eyes?"

Dangerous as the more grossly fraudulent schemes have proved to would-be investors in the past, there is often even greater danger in propositions put forward with the best intentions by half-informed inventors and promoters. While, therefore, it can be amply demonstrated that no class of investments can be counted on for larger and more regular returns than these in well considered enterprises based on chemistry, no one untrained in chemistry should consider such investments without expert advice.

The Escape of Gas From Coal.

THE ESCAPE OF GAS FROM COAL is the title of Technical Paper No. 2 just issued by the Bureau of Mines. The authors, H. C. Porter and F. K. Ovitz, in their general statement say: "It is a matter of common knowledge that inflammable gas made up chiefly of methane, which forms with air the so-called fire damp of the miner, escapes from the coal in many mines; yet little is known as to the condition of this gas in the coal, its quantity and rate of escape. In addition to the gas set free by the coal itself, there is in many mines a large amount of inflammable gas that comes from reservoirs in the rock strata above or below the coal bed and enters the mine through cracks in the roof or floor. Gas entering the mine in such manner from extraneous sources is not considered in the following report, which presents rather some results of a laboratory investigation of the rate of escape of gas from several coals while kept in bottles connected to gas-collected reservoirs. This investigation was begun by the United States Geological Survey and is being continued by the Bureau of Mines at the mining experiment station at Pittsburg, Pa.

The results of the investigation show that gas escapes from coal not only during the breaking down

of the coal in the mines, but also continuously for a long time after the coal is mined. The volume from some coals is so great as to merit serious consideration in connection with the ventilating of the mines and the choice of methods of breaking down the coal. For example, during the first two weeks after mining, there was set free by one coal, exclusive of the gas that escaped during mining, a volume of methane equal to three-fourths of the volume of the coal itself. Probably each cubic foot of this coal (in the seam) set free in all during its mining and during the first two weeks thereafter a cubic foot of methane. It was shown by further test that during the first five months after mining there was set free from this coal a volume of methane equal to one and three-fourths times the volume of the coal.

Many investigators, Chamberlin, among others, have shown that methane accumulates along lines of fracture, such as joint cracks, or in places of increased porosity in the coal bed and escapes from these as they are cut in mining. In any locality of the mine where for any reason adequate ventilation is not maintained, the methane which has thus escaped may become a source of danger from explosion.

The results of the experiments discussed in this paper show that certain American coals, whose mining is attended with danger from accumulations of inflammable gas, liberate this gas not only while they are being broken down in mining, but also during a long period thereafter. At first the gas escapes rapidly, but the rate diminishes and tends toward a final cessation in three to eighteen months. If the volume of the small lumps of coal used in the experiments be taken as the unit of measurement, about one-fourth volume of methane escapes during the crushing of the coal, as shown by Chamberlin, and one-half to one and one-half volumes on continued exposure to the air, as shown by the authors. The loss of fuel value by this loss of gas is small, but the danger of accumulation of explosive gas from this source in mines and in coal bunkers is sufficient to justify its being taken into account in the ventilation of mines and in the storage of coal.

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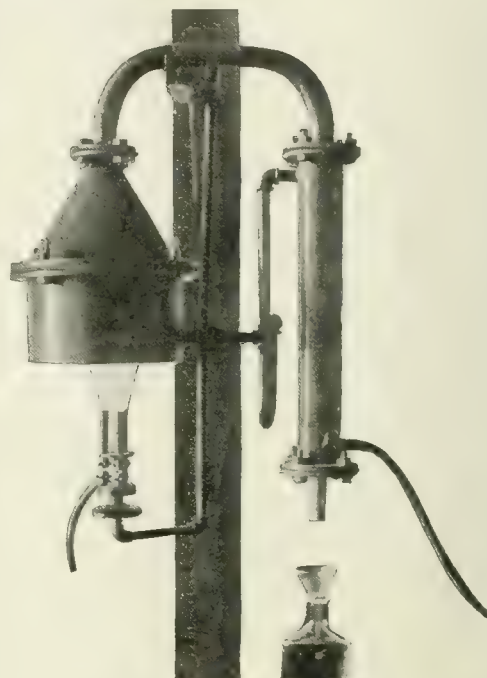
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No. 1.

THE CONTROL OF THERMAL OPERATIONS AND THE BUREAU OF STANDARDS.*

By GEORGE K. BURGESS.

The name that is generally recognized as standing foremost in the determination of the thermal properties of materials is that of Regnault, but it is perhaps not as generally known that the experimental investigations in heat carried out by him over an interval of some 30 years, beginning in 1840, were initiated and supported by the French Government. The problem that Regnault first attacked was that of the properties of steam, a particularly opportune subject at that epoch, and the title page of his first memoir on the subject reads as follows:

"Account of the Experiments undertaken by order of the Minister of Public Works, and on the proposition of the Central Committee on Steam Engines, to Determine the Principal Laws and the Numerical Data which enter into the Calculation of Steam Engines."

This was published in 1847 or seven years after the granting of the government subsidy. This delay was in part due to the necessity Regnault labored under of carrying on many researches, fundamental and subsidiary to his main problem, including, for example, the construction of both a temperature and pressure scale, as well as determinations of absolute densities and of specific and latent heats.

The example thus set by France in giving sustained governmental aid for the determination of thermal constants of fundamental importance has been appreciated and followed after many years, it is true, by other countries, several of which have established government laboratories for the carrying out of investigations, not only on the fundamental thermal properties of materials but also, in some instances, with activities covering the whole range of the physical sciences including the testing of instruments and materials submitted by the public. Of this type, are the German Imperial Physico-Technical Institution (P. T. Reichsanstalt) established in 1887, the National Physical Laboratory of Great Britain organized in 1899, and the Bureau of Standards founded only ten years ago.

It would take us too far afield to endeavor to cover

the whole subject of standardization and its relation to industry, and it will serve our purpose to limit ourselves to the consideration of a domain, that of heat, in which the development of standards has been less rapid and less exact than in many other lines. We shall sketch in some detail the part taken by the Bureau of Standards in the control of thermal operations, with the object of calling attention to an aspect of the question of economic efficiency that is of fundamental importance, namely, the role of common units and standards, their establishment, maintenance and availability.

It is a most painstaking matter to set up and maintain a temperature scale; in fact, the text books state that temperatures cannot be measured and that we can only recognize or locate a given temperature in terms of some other phenomenon that we can recognize or measure. For such reasons, in great contrast to the certainty and permanence of our standards of length, mass, time and more recently the electrical quantities, we have had and still have great discrepancies in our temperature scales and all the varied quantitative relations and thermal constants depending on them.

It is eminently fitting, therefore, and particularly necessary that there be a central authority for the establishment, maintenance and distribution of a single temperature scale and the derived heat units. The International Bureau of Weights and Measures, established in 1871, at Sevres, near Paris, revising the work of Regnault and others performs this function in the first instance for the fundamental thermometric interval 0-100°C, while the higher and lower temperature ranges and derived quantities have been left mainly to the various national laboratories since their establishment, as they were previously left entirely to individual initiative. Among the most brilliant examples of the latter are those of Barus in the early eighties when he was physicist of the Geological Survey, Rowland at the Johns Hopkins, and our own Holman.

Experience has shown, however, that it is quite impracticable for any individual working alone, and engaged in other pursuits as he must be, to maintain and disseminate a temperature scale. Indeed, progress in exact measurements in thermal constants and in all physical constants depending on temperature has often been greatly hampered in the past by the unavailability of any common temperature scale in terms of which results might be expressed. Each

*Paper presented before the Congress of Technology at the 50th Anniversary of the Granting of the Charter of the Massachusetts Institute of Technology.

†Associate Physicist, Bureau of Standards, Washington, D. C.

investigator was speaking a different temperature language which he could translate into no other. It required, for example, several elaborate, subsequent researches to express the results of Rowland's magnificent work on the mechanical equivalent of heat in terms of the Hydrogen Scale of the International Bureau. Today, it is neither necessary nor desirable for an independent investigator to try and set up his own temperature scale. Continuity and permanence of thermal standards can only be maintained to advantage by laboratories especially equipped for the purpose, the organization and working of which are not dependent on any individual, and the equipment and staff of which are adequate to meet the demands of the public.

Since its creation in 1901, the Bureau of Standards has been actively engaged, in so far as its resources would permit while developing many other lines of work, in serving the United States in the capacity of an advisor on best values and available methods in the many lines of thermal measurements. This activity has necessitated the undertaking of a considerable number of experimental investigations, the carrying on of a very extended correspondence, and the execution of many tests of instruments and materials. The bureau does not impose its authority in thermal standards on anyone, and any prestige in this field it may possess must rest on the reasonableness of its suggestions.

The temperature scale of the International Bureau has been reproduced here to 0.002°C in the interval 0 to 100°C or to as close as temperatures are known within this interval; and for the remainder of the temperature range, for which there is no international authority, a temperature scale has been constructed which, except perhaps for the very highest temperatures, is in substantial agreement with those maintained by the German and British laboratories. This temperature scale of the Bureau of Standards is reproducible to about 0.03° , at 500° , 1° at $1,000^{\circ}$, and 10° at $2,000^{\circ}\text{C}$., and has required for its establishment a great deal of experimental work, and advantage has of course been taken also of similar work elsewhere.

It may be of some interest to consider the relation of this Bureau to some of the various bodies with which it comes in contact, and mention in particular some of the thermal problems to the solution of which it has contributed.

The Bureau is by law constituted the custodian of the standards and is authorized by the organic act creating it to exercise such functions as "the comparison of the standards used in scientific investigations, engineering, manufacturing, commerce, and educational institutions with the standards adopted or recognized by the Government; the construction, when necessary, of standards, their multiples and subdivisions; the testing and calibration of standard measuring apparatus; the solution of problems

which arise in connection with standards; the determination of physical constants and the properties of materials, when such data are of great importance to scientific or manufacturing interests and are not to be obtained with sufficient accuracy elsewhere.

Questions arising in the departments of the Government requiring advice or decisions on the physical properties of materials are usually referred for solution to the Bureau, and this demand for its services and advice is growing at a greatly accelerated rate for tests of various kinds of temperature measuring instruments, for the formulation of specifications, for the purchase of thermal apparatus, and of materials on which thermal tests may be made to control their quality, and for the carrying out of specific experimental investigations on materials, processes, or methods involved in the solution of some problem in which one or another department of the Government is interested.

To mention a few examples among many of the control thus exercised, the purchase of clinical thermometers for two of the departments as well as of the various kinds of thermometers for several of the scientific Bureaus, is based on specifications drawn up by the Bureau and its testing of the instruments before their acceptance; and the knowledge on the part of manufacturers that a bid may be rejected due to the findings of a disinterested, competent authority is not detrimental to the class of instruments submitted in competition for purchase in this way.

The purchase of refractory brick by the Panama Canal Commission is also based on such tests, as well as that of many other materials.

The decision as to whether contested materials are inflammable and, therefore, to be barred from carriage on passenger steamers is left to the Bureau, and such decision sometimes entails an unexpected amount of experimentation.

Investigations of lubricating and illuminating oils and of the types of apparatus used in the testing of them are being carried out with several objects in view such as the drawing of better government specifications, the furnishing of data for better and more uniform testing methods both in this country and by international agreement. The tests of viscosity, for example, are now on a purely empirical basis, different countries use different instruments, and in the United States there are several incommensurable instruments in common use. It is hoped to reduce all such measurements to a common basis.

The Bureau has been asked to take part in the work of several International and National Societies or Committees, usually with the object of establishing by experimentation the necessary conditions or specifications for the carrying out of some method of testing or standardizing materials, methods, or instruments. We have not been able to meet all the demands of this kind in the field of thermal operations, but have limited ourselves to some of the prob-

lems that are in the most unsatisfactory condition. Besides the oil question, which is very troublesome, the subject of combustion and gas calorimetry has been attacked from the foundation, and although less than two years have been spent on this, we not infrequently receive praiseworthy letters of a slightly ironic cast asking for our final results on this or that phase of the problem. Accepting Regnault's standard, we still have five years to make good on this problem however. An outsider does not always realize that no one is more impatient to finish his problem than the worker himself, but where fundamental standards are concerned the worker is bound to be more conservative than his severest critics.

In this calorimetric work it has been necessary, for example, to design and install a complete plant for the production of pure oxygen, new calorimeters, resistance thermometers and accessory apparatus capable of giving results of the highest accuracy attainable at the present time; and the thermal constants must be determined with different sets of apparatus and checked by several methods.

This calorimetric investigation is very far-reaching and involves, for example, the heat values to be accepted as standard in this country for the constituent components of gas burned in every city and town and, therefore, the ultimate basis on which the price of gas as fuel should be fixed. The heats of combustion which are being determined of those substances suitable for calibrating bomb calorimeters, in a similar way, will define the ultimate units on the basis of which coal and oil fuels should be bought and sold. Concrete calorimetric standards are realized in the form of certified samples of pure materials of known heat value, which are distributed to interested parties. The various types of calorimeter are also being compared, and incidentally to the whole investigation, improved methods of measurement and new instruments have been developed.

To the refrigerating industries we have furnished after elaborate experimentation the correct values of the specific heat of brine at low temperatures, a constant of very great importance to them. There is great need for further systemization of units in this field as well as the better determination of other constants fundamental to refrigeration. The question of these units will be gone over very carefully at the next International Congress of Refrigeration which meets in this country next autumn.

The relation of the Bureau to the manufacturers of precision instruments is very close, manifold, and in general most cordial. When it is remembered that previous to 1901 there was no generally recognized standardizing authority in this country for thermometers and that each manufacturer had his own scale tied up in one or more long cherished instruments, the breakage of which was a real calamity for him and his clients, the necessity for the establishment of a standards bureau is self-evident.

Besides the testing of thermometric instruments, and thereby furnishing all manufacturers with a single scale, uniform everywhere—the Bureau, in certain cases, loans its own standards. It has also taken a very active part in the improvement of specifications and the methods and materials of manufacture as well as in testing methods. The resulting improvements in American thermometer manufacture have been such, that in 1906 Gehr. Wiebe, the chief of the thermometer department of the Reichsanstalt, published the statement that the German thermometer manufacturers were complaining more and more of the loss of the American market.

The manufacturers of instruments for the measurement of high temperatures have also been aided greatly, and the condition in this field may be described as having been chaotic before the benefits of uniformity of calibration could be realized. Opportunity is also given for trying out and developing new types of instrument, and a great many suggestions have been given manufacturers which have been incorporated in their instruments. The standards of practically every pyrometer and thermometer manufacturer in the country have been tested at the Bureau.

The past ten years have witnessed great strides in the development of the manufacture and use of pyrometers. Formerly the question usually asked by an engineer responsible for the control of some process involving change of product or output with varying temperature, was "Do I need a pyrometer and is there a reliable one?" Today he knows he needs one, and asks which is the best for his purpose. The Bureau carries on a very heavy correspondence with the users of such instruments endeavoring to furnish them with the information best adapted to each case, although no pretence is made of replying to the often asked question, which is the best pyrometer?

In a similar way, advice has been furnished to Universities and Technical schools regarding the formation of courses of instruction in heat measurements, the installation and design of new apparatus, and the testing with the highest possible accuracy of instruments to be used by them in numerous experimental investigations.

To still further aid in the dissemination of a uniform temperature scale, preparations are being made for the distribution of samples of pure metals and salts of certified melting points, thus permitting anyone to check his own thermometric or pyrometric apparatus in place. As a preliminary to this, a careful survey of the reproducibility of the thermal behavior of such materials from several sources of supply has been made.

Among the other investigations under way or in immediate contemplation, that have a bearing on the control of thermal operations, may be mentioned studies of thermal properties of steels and refractories, including in both instances the purest obtainable ma-

materials and the commercial products; the properties of steam with respect to turbine design; the melting points of the elements and of some pure salts and alloys; gas thermometry; the temperatures of incandescent lamps; the corrections to be applied to optical and radiation pyrometers when sighting on steels, bronzes, clays, and so forth, and the comparison of the laws of radiation on which the estimation of the highest temperatures is based.

Some of these experimental problems have been undertaken at the suggestion of one or more interested individuals or corporations who often are in a position to furnish materials that may sometimes be difficult to obtain otherwise. Indeed, we are at times in the perhaps happy, but nevertheless embarrassing position of being offered a great many more opportunities of this kind than we can embrace.

Still another function the Bureau has occasionally exercised, and one that will probably develop, is to act as referee or court of appeals in case of disputes involving standards, physical properties of materials and the like, and for the interpretation of contracts involving testing methods.

The testing activity of the Heat Division of the Bureau for the past four years ending June 30, 1910, is shown in the following table:

	1906-7	1907-8	1908-9	1909-10
Clinical Thermometers	8,444	8,395	10,955	13,082
Various Thermometers.....	577	798	1,791	1,508
Pyrometers	46	60	31	39
Calorimetric Tests.....	..	..	41	58
Oils: Flash and Viscosity.....	42	17	159	17
Miscellaneous	9	99	6	45

Several of these miscellaneous tests were of the nature of quite elaborate investigations and most of the pyrometers were manufacturers' standards. Nominal fees are charged for all tests except for the United States or State Governments. About one-third of this testing is for the United States Government. The Bureau makes known the results of its investigations by means of its Bulletin, and its testing methods and specifications are described in Circulars of Information.

With the ever-increasing economic competition and the improvement of scientific methods and instruments as applied to the arts and sciences, the demand for higher accuracy of calibration is also constantly growing. Where, for instance, $0^{\circ}.01$ C. was considered an ample precision a short while ago in combustion calorimetry, and $0^{\circ}.001$ C. sufficed yesterday, the demand is now for a certainty of $0^{\circ}.0001$ C. in temperature difference at 25° C. The meeting of such rigid requirements necessitates oftentimes the application of entirely new methods and instruments, and we are witnessing in consequence the passing of some familiar types such as the mercury-in-glass thermometer from the list of instruments of precision.

A most important step is yet to be taken in the unifying of the temperature scales and the values assigned to the various thermal constants in use in

the several countries. Today, a pyrometer manufactured in France, calibrated in England and used in the United States comes in competition with a German-made and certified instrument, and with one of American make and certification. They will each give a different temperature for a furnace at say $1,500^{\circ}$ C. We have the example of the unification of the standards of weights and measures as well as those of electricity and photometry, and it is to be hoped that ere long the standards of heat may also be rendered universal.

The Bureau of Mines has begun the publication of a series of short papers to be known as "Technical Papers," intended to convey to engineers and others quickly the results of certain of the investigations. The first of these, known as Technical Paper No. 1 has just been issued under the title, "The Sampling of Coal in the Mine," and the author is Joseph A. Holmes, the director of the bureau.

In the general statement, the author says:

"In dealing with coals no less than in dealing with ores, the taking of the sample requires as much care as does the making of the analysis or assay. And the difficulties in the way of obtaining, at reasonable cost, a sample of coal that fairly represents the commercial product as found in the mine or, more especially, as loaded in cars or in ships, have seriously retarded the movement for the sale or purchase of coal on a rapid specification basis.

"The purpose of this paper is to describe briefly the method now followed by the Bureau of Mines and United States Geological Survey in an endeavor to take mine samples that fairly represent the beds of coal that are examined and that show, for the places sampled, the commercial possibilities of these beds. It is of the utmost importance that the sampling be done in a systematic manner, according to a prearranged plan, and that the same procedure always be followed where circumstances permit. Wherever it is possible, unless special samples are desired for a particular purpose, only clean, fresh coal should be sampled, and all dried, weathered, or long-exposed coal should be avoided. When weathered coal, either in the outcrop or in pillars, or other special samples are collected, the particular characteristics of each sample should be clearly described."

The production of bauxite in the United States in 1909 amounted to 129,191 long tons, valued at \$679,447. The year 1908 was one of general business depression, and therefore the normal growth of the bauxite industry can be seen by comparing the figures for 1909 with those for 1907 rather than with those for 1908. As compared with the production of 1907, the output in 1909 showed an increase of 32 per cent in quantity and 41 per cent in value. The average price of the ore per ton at the mines was: 1907, \$4.91; 1908, \$5.06; 1909, \$5.26.

THE INDIA RUBBER INDUSTRY.*

The foundations of the rubber industry were laid considerably over a century ago, yet it must, as an industry proper, be regarded as of a comparatively modern and recent origin. Although Samuel Peal in 1791 patented a process for water-proofing cloth by spreading it with hot rubber, Thomas Hancock, of London, and Charles Macintosh, of Manchester, are generally regarded, in conjunction with Charles Goodyear, of New Haven, U. S. A., as being the actual founders of the industry. Macintosh in 1820 produced water-proof garments by a practicable process, and about the same date Hancock invented a masticating machine and process, thus introducing the manufacture of rubber sheet and much improving the method of making "solution." It was not, however, until after the process of vulcanization by the combined action of sulphur and heat, had been discovered, by Goodyear in 1839, and independently by Hancock about 1844, that the rubber industry showed signs of becoming of great importance. In 1846 Parkes found that india rubber may be vulcanized in the cold by the action of sulphur chloride suitably diluted, but this process is only applicable to goods of very moderate thickness. The influence of the discovery of "vulcanization" on the development of the rubber industry may be judged by the following figures representing tons of crude rubber imported into the United Kingdom: 1830, 23 tons; 1850, 381 tons; 1870, 7,656 tons; 1890, 13,200 tons; average 1904 to 1908, 29,389 tons. Only a very small proportion of the total passes to the consumer in the unvulcanized state.

At the present day the total production of raw rubber is probably between 70,000 and 75,000 tons broadly distributed as follows: Brazil, Peru, Bolivia, 40,000 tons; other South American countries, Mexico and Central America, 7,000 to 8,000 tons; Africa, 17,000 to 20,000 tons; Ceylon, Malaya, Java, Sumatra and Borneo, 4,000 to 5,000 tons. Miscellaneous, 1,000 to 2,000 tons.

The most important rubber consuming countries are the United States, the United Kingdom, Germany and France. Italy and Russia also import and manufacture considerable quantities. As a rough approximation it may be said that the United States consume one-third of the world's rubber, the United Kingdom and Germany between them (in approximately equal shares) one-third, and the other countries the remaining third.

The accompanying figures will give a general idea of the importance of the rubber industry in the United Kingdom in 1908.

Nineteen hundred and eight, owing to the American financial crisis, was an abnormal year; the totals of the preceding year are therefore also given.

Of the total imports, rather more than one-half are

re-exported, the average figures for the last five years being:

EXPORTS OF RAW RUBBER			Quantity, cwt.	Value, £.
Average imports 1904 1908.....	587,781			9,302,990
Average re-exports 1904 1908.....	334,129			5,858,488
Balance retained for consumption..	253,652			3,444,502
MANUFACTURED GOODS.			Exports, 1909.	Imports, 1908.
			Value, £.	Value, £.
Miscellaneous goods	1,576,000			482,644
Boots and shoes.....	205,668			123,381
Waterproofed goods	295,184			6,825
Rubber covered cables, other than telegraph and telephone cables.....	289,342			103,230
Telegraph and telephone cables.....	744,140			125,087
Total	3,110,334			841,167

This does not include inferior grades (very resinous) rubbers, such as "Pontianac" or "Jelutong." The annual production of these is probably some 20,000 to 30,000 tons, the United States alone importing some 12,000 tons annually.

There are no published figures on which an estimate of the value of the manufactured articles consumed in the home trade can be based with any certainty, but it may be safely placed at not less than ten millions sterling, assuming normal rubber prices.

The india rubber industry as a whole may be conveniently and naturally subdivided into three sections, namely (1) the production of crude rubber;

IMPORTS OF RUBBER IN 1908			Quantity, cwt.	Value, £.
From.				
Brazil	300,032			5,331,842
Peru	29,622			518,885
French Colonies	58,159			553,615
Other foreign countries.....	112,085			831,819
Total foreign	499,898			7,236,161
British possessions	75,168			1,135,044
Total 1908	575,056			8,371,205
Total, 1907	667,294			10,834,759

(2) manufacture of articles from and with the aid of the crude material; (3) treatment and utilization of waste rubber.

India rubber is contained, in the form of an emulsion, in the latex or milk of the laticiferous system of various plants. It also occurs, but not frequently, in the solid state, as a deposit in the woody fiber of certain species, for instance in the Guayule shrub (*Parthenium argentatum*). The laticiferous system of most rubber bearing species lies between the outer bark and the cambium. By cutting through the outer bark and into the latex cells the latex may be obtained as a white to cream-colored, more or less viscous liquid. In former days this operation of "tapping" was carried out in a very rough and ready fashion, resulting in injury to the wood and so to the life and yielding capacity of the tree. As regards the "native" or "wild" rubber from large tracts of South and Central America and from Africa this still holds good, since under present conditions of collection, adequate supervision of the natives and methodical working is out of the question. For the correct tapping of a tree, the following factors are of cardinal importance, namely (a)

*From Journal Society Chemical Industry.

the age and species of the tree; (b) the method of making the cut; (c) the disposition of the cuts and the general contour of the cutting system; (d) the season; (e) frequency of tapping; (f) methods of collection.

Occurrence of India Rubber Bearing Species.—Rubber bearing species are indigenous to considerable tracts of the tropical and sub-tropical zones. Favorable conditions are a moist climate with high and equal temperatures and a fairly copious rainfall. Although the nature of the soil is of importance, certain species thrive well in soils which would be regarded as poor for planting many other tropical products. Rubber species are very numerous but the following are of greatest industrial importance:

1. *Hevea Braziliensis* or Para Rubber. This is indigenous chiefly to the area watered by the Amazon and its tributaries. It has been almost exclusively employed in the plantations of Ceylon and Malaya and largely in Sumatra, Java, Samoa, etc.

2. *Ficus Elastica*, or "Rambong," is indigenous to Burmah, Ceylon, Malay, Java, India, etc. It has been planted to some extent in the eastern Dutch colonies and in some parts of the German possessions (New Guinea, etc.). Other *Ficus* species are found in Africa and South America.

3. *Funtumia Elastica*.—Indigenous to many parts of Africa, planted experimentally in Uganda, the Cameroons, etc.

4. *Landolphia* species occur profusely in many of the African rubber districts and yield a considerable proportion of the crude product exported from Abyssinia, the Congo, and from the West and East coasts.

5. *Castilloa Elastica*.—The natural rubber par excellence of Mexico and Central America. It has been largely planted in Mexico and experimentally elsewhere.

6. *Manihot Glaziovii*.—Indigenous in the State of Ceara (Brazil). Planted experimentally in the East, the Cameroons, etc. Other rubbers of some technical importance are *Guayule* (*Parthenium argentatum*) the *Sapium* varieties, the *Hancornia* species and resinous rubbers such as *Dyera castulata* or "Jelutong".

Quality and general attributes of different species: There are very marked differences in the quality of various rubbers, attributable partly to the inherent properties of the various latices, and partly to the methods of collection and preparation. Two main factors determine the value of a rubber, namely, its chemical purity and its physical properties. The chief impurities are moisture, resins, proteins, ash, matters insoluble in rubber solvents, and such adventitious impurities as mechanically admixed earth, bark, etc. In regard to physical properties, the main considerations are strength and vulcanizing capacity.

Differences due to inherent properties of the latex: There is, in regard to chemical composition, in the narrower sense of the term, no very great difference between the latices derived from the more important

species; but the physical properties of rubbers prepared from the main species appear to exhibit wider differences than do the chemical properties, but it is probable that improved methods of preparation will tend to minimize these differences also. The best of the *Funtumia* and *Landolphia* species seem to show the greatest mechanical strength, then follow the *Hevea* and *Ficus* species, although some would perhaps place *Manihot* nearer the head of the list. The differences in regard to physical and physico-chemical properties between the various species are of considerable interest inasmuch as it would appear from the work of Harries and Gottlob that the complex caoutchouc molecule in some of the African varieties (probably *Landolphia* and *Funtumia* species) is not identical as regards complexity and disposition of the simple molecule with the molecules of the *Hevea* species. In splitting the ozonide derived from the *Hevea* rubber certain quantities of leavulinic-aldehyde and leavulinic acid were obtained. The fission products derived from the African species and also from gutta percha were constant in quantity relatively, to one another, but the relation of aldehyde to acid was practically the reverse of that observed in the case of the *Hevea* derived ozonides. There are two main factors which must be taken into consideration in connection with the physical properties of pure caoutchouc. In the first place there may be differences in regard to the molecular complexity, i. e., the state of polymerisation of the molecule itself, and, secondly, differences in regard to the colloidal state of the rubber particles, that is as Spence has recently pointed out, the differences of physical aggregation. These differences may well occur even in rubber of one and the same species. My own work on the viscosity of rubber solutions is confirmatory of the view that from one or several of the reasons given there are specific differences in regard to the actual "caoutchouc" contained in *Hevea*, *Funtumia*, *Castilloa*, and *Sapium* varieties.

Anyone inspecting the various forms of rubber emanating from Africa, Asia and tropical America would scarcely believe that most of these are prepared from latices, which if properly treated, would yield rubbers differing little in commercial value from fine hard Para on the one hand, or the best plantation products on the other.

It is mainly due to the work of the chemist, stimulated by the rise of the plantation industry, which in its turn is mainly due to the efforts of economic botanists that this change in our views and this important addition to our knowledge has come about.

Differences Due to Methods of Preparation.—The bulk of the raw rubber is still prepared in a very crude manner, and this is likely to be the case for some time to come. Practically all "wild" rubbers (rubbers prepared by the natives of the rubber regions without skilled white supervision) contain in addition to a considerable proportion of moisture,

a varying quantity of mechanical impurities. Such rubbers have to be thoroughly cleaned before they can be used in the factory. The loss which native or "wild" rubbers show after this process has been completed generally varies from 10 to 50 per cent, or even more. Frequently native rubbers are wilfully adulterated with sand, stones and dirt, not to speak of admixture with inferior rubbers. The foregoing applies chiefly to African and Asiatic rubbers, for the South American rubbers are, on the whole, better prepared, many of those being from the valley of the Amazon being of remarkably fine quality. These rubbers have been, ever since the rubber industry has attained to considerable proportions, a standard of quality, and prices of pure plantation rubbers are still referred to fine hard Para as a standard. The Para rubbers are prepared by coagulating successive films of latex round a core by the combined action of heat and wood smoke. In this way the whole of the non-rubber substances contained in the latex are incorporated.

THE PLANTATION INDUSTRY.

Perhaps the most important occurrence in the rubber industry since the discovery of the process of vulcanization is the foundation and development of plantations. In 1875 Wickham was commissioned by the Indian Government to obtain from the Amazon valley a supply of *Hevea B.* seeds, and from these and from small quantities obtained from other sources, substantially the whole of the 60 to 100 million trees now growing in the East have been raised. In 1876 the first large batch of seeds was planted in Ceylon, but the following 20 years constituted a purely experimental period, and it was not until some years of the present century had passed that planting on a really industrial scale commenced. The evolution of the industry may be gauged by the following figures:

Approximate acreage under Rubber in Ceylon.

Year.	Acreage.
1890	300
1895	550
1898	1,250
1904	11,000
1907	150,000
1910	190,000

The following is an estimate of the total acreage under planted rubber in various parts of the world:

Country.	Acreage.
Ceylon	190,000
Malay Peninsula	250,000
Java, Sumatra and Borneo.....	90,000
India	25,000
German Colonies	38,000
Mexico, Brazil, Africa and W. Indies.....	100,000
Total	693,000

From the experience gained so far, 200 to 250 lbs. to the acre appears to be a safe estimate of yield, but not immature trees, planted under reasonably favorable conditions. During 1909 the Eastern plantations exported roughly 4,000 to 4,500 tons, of which 2,800 came from Malaya.

Plantation Methods.—Wild rubber is generally collected from scattered trees as a rule in forest re-

gions inaccessible except to the natives of the district. In most cases, therefore, skilled superintendence of collection of the latex and preparation of the rubber is out of the question. The planter is able, under favorable circumstances, to produce 250 lbs. of pure, dry rubber from a single acre of land, whereas the native collector in the forests of Africa and South America may have to scour 40 or 50 acres, if not more, to obtain the same quantity of, in many cases, an extremely inferior product. The tree which is most generally, in fact, in the East almost exclusively, planted at the present time is *Hevea Braziliensis*, and it has been found possible and profitable to tap this tree at the end of five years. It would be desirable that tapping should be deferred, say, till the sixth year in most cases, for early tapping produces a comparatively inferior rubber, and is detrimental to the tree. The question as to whether, assuming tapping is conducted in a reasonable manner, the tree may be tapped indefinitely for a successive number of years may be answered in the affirmative, for there are trees in the various botanic gardens in the East which have been successively and successfully tapped for some thirty years or more, and which show no apparent injury.

Cost of Production.—There are two main factors to be considered in this connection (a) cost of bringing into bearing; (b) upkeep and manufacture. The cost of bringing into bearing varies from £18 or £20 upwards, the average being about £40 per acre. At 5 per cent this corresponds to a fixed charge of £2 per annum, or assuming that 240 lbs. per acre are produced to 2d. per lb. of rubber. With regard to upkeep and manufacture, etc., the following figures are based on the actual working expense of a Malay estate which produced about 100 tons of rubber in 1908, at 200 lbs. per acre, and estimated, when mature, to produce 400 lbs. per acre:

	Per lb. of rubber produced.
General charges (salaries, allowances, quit rent, hospital, contingencies)	2-31d.
Cultivation charges (weeding roads, drains, supplying, pests and diseases, etc.).....	1-85d.
Upkeep of buildings.....	0-23d.
Rubber manufacture:	
Tapping and scrapping.....	3-93d.
Curing	0-43d.
Packing	0-33d.
Marking, utensils, transport, etc.....	0-47d.
Duty	1-19d.
Manufacture—total	6-35d.
Total working charges	10-74d.
Total charges:	Per lb.
Fixed	2-00d.
Working	10-74d.
Total	12-74d.

To place the rubber on the European market a further 3 30d. (freight, brokerage, etc.) must be added, making a total of 16-04d., or 1s. 4d. per lb. The product from particularly well placed estates may be put on the European market at 10d. to 1s. per lb.; on

the other hand there is little doubt that a certain proportion of the planted areas will not show any considerable profit at less than 1s. 6d. to 2s. per lb.—or more.

METHOD OF MANUFACTURE.

The following essential desiderata should be aimed at (1) Collection of maximum amount of latex with minimum amount of injury to the tree. (2) Absolute cleanliness of collecting and transport vessels. Metal containers should be avoided, glass, earthenware or enameled collecting cups, pails, etc., being preferable. Bare metal certainly affects the color and possibly other qualities of the finished rubber. (3) The latex to be strained or filtered as soon after collection as possible, so that all mechanical impurities may be removed without delay. (4) To restrict, as far as possible, the formation of "bark scrap."

Coagulation and Curing.—The method employed in coagulating the bulk of the plantation rubber consists, substantially, in the addition of a small quantity of acetic acid to the latex after the latter has been strained and preferably reduced to a certain rubber content by mixing of various batches, or by dilution. In most cases, just sufficient acid is added to render the liquid neutral or faintly acid to litmus. After the coagulum has formed it is removed from the mother liquor and passed through the washing rolls, from which the rubber emerges in the form of crepe, and if this form of rubber is required it passes direct to the drying sheds. If sheet is required the crepe is passed through a sheeting machine, which consists of rolls revolving at even speed. Intermediate grades obtained by subjecting wet crepe to the action of even speed rolls are also turned out. In the manufacture of biscuits, now only produced in relatively small quantity, the latex is directly coagulated in circular pans or basins.

"Worm" and "Lace" rubbers are also produced by passing the coagulum through appropriately designed rolls or dies. In some cases where it is desired to produce rubber of very light color, the crepe or sheet is passed, before it goes to the drying rooms, through water heated to about 180° F., a process due to Mr. Kelway Bamber, who discovered that *Hevea* latex contains an enzyme which causes the darkening of the rubber, and which can be destroyed by heating to the temperature mentioned. This does not apply to all other latices. Freshly coagulated *Funtumia* rubber is, for instance, perfectly white and may be boiled for a long time with water, but nevertheless darkening subsequently takes place. In the drying rooms the rubber is placed on perforated shelves, or as in the modern practice, is hung from racks. In all modern drying houses provision is made for passing a current of warm air through the same. Partially dried air is preferable, but it is doubtful whether any drying process for this purpose is economical. Another method of drying is by means of the vacuum dryer,

in which rubber may be dried in almost as many hours as it takes weeks in the ordinary fashion. It has been said that the vacuum dried rubber is not so satisfactory as air dried, but where inferior results are noticeable, they are probably due to imperfect methods of applying the vacuum. Thick crepe is generally prepared by passing several layers of the thin crepe through the rollers simultaneously. Block rubber is prepared from crepe by subjecting it to pressure in a hydraulic press while it is still warm from the vacuum dryer.

Other coagulants than acetic acid are occasionally used and there are various modifications of the other processes referred to, but substantially the above gives in outline the essential features of the production of Para plantation rubber.

Other Varieties.—With regard to the preparation of varieties other than *Hevea* on the plantation system, such differences as occur in the methods employed are the result of the fact that the inherent properties of different latices vary considerably, and it is therefore necessary to modify the method of coagulation. It may be suggested that this is perfectly obvious, but as a matter of fact, much trouble has been caused by efforts to apply to different latices methods which have been favorable in the case of *Hevea*.

Guayule Rubber.—The production of rubber from the Guayule shrub (*Parthenium argentatum*) is of particular interest, because the problems involved are more intricate than those dealing with the production of rubber from ordinary latex bearing plants, and also because guayule is produced on a very large scale, the amount manufactured during the last year being estimated at roughly 4,000 tons. The Guayule shrub, which prefers an altitude of from 4,000 to 5,000 ft. is distributed unevenly in a belt of territory from one to a hundred miles in breadth extending roughly from Ft. Stockton in Texas to the Tropic of Cancer in Mexico. In this shrub rubber occurs in solid particles dispersed through the mass of the woody fibre. According to M. P. Fox the wood contains from 6 to 18 per cent of rubber, a similar quantity of resin, and about 10 per cent of extractive. Two processes are employed to separate the rubber from the wood, one consisting in extracting the rubber by means of solvents, and the other, which consists in disintegrating the woody mass in such a manner as to separate the rubber; a combination of the two processes is also employed. Two main grades of Guayule are prepared, one containing about 30 per cent of resin, and the other, from which part of the latter has been extracted, containing about 3 per cent of this impurity. Since the extraction of rubber involves the destruction of the shrub it is estimated that within four years from now practically the whole of the standing shrubs will have been destroyed. For the period 1906 to 1909 some 17,000 tons of rubber were prepared from

Guayule, which corresponds on a 7 per cent basis (for wood containing 25 per cent of water) to roughly 328,000 tons of shrubs.

FUTURE OF THE CRUDE RUBBER INDUSTRY.

It is perfectly clear that within a measurable period the crude rubber industry will have undergone a complete change. I believe that within a very few years' time the manufacturers will refuse to have anything to do with any raw material that is not perfectly clean and practically dry. Therefore the bulk of the wild rubber in the market will have to be prepared in a different manner, or it will be unsaleable. It has already been shown in various parts of the world that the plantation system may, under favorable circumstances, be applied to wild trees, and it seems likely that in time the system of rationally working wild as well as planted trees will be extended over favorable areas. This involves expense in the way of cutting roads and paths, and making arrangements necessary for the transports of latex or rubber bearing wood to central stations. It is obvious that only such areas can be worked as those on which the trees occur in fair number. Probably the most difficult question involved is that of labor. The drawbacks to the rational working of wild rubber are considerable, but on the other hand there are advantages. While it is frequently assumed that the first class of rubber to disappear under the stress of competition will be the inferior African grades, I am by no means sure that this will be the case, because on the whole, the question of labor is not so acute in most of the African rubber zones as in South America, and there are already signs that those interested in the West Coast, the Congo, etc., will take the necessary steps to improve the quality of the rubbers from these regions. With regard to the Para grades, the labor problem appears to be a very serious one, and in view of this, and of the extremely unhealthy climate, it is difficult to see how the Amazon valley is indefinitely to maintain its superiority as a producer. Certainly the opening up of the vast forests of Brazil involves the overcoming of apparently insuperable difficulties. The third factor—taking planted and ordinary wild rubbers as the two others—in the crude rubber problem, is the preparation of pure, high class rubbers from inferior, naturally resinous grades by chemical or other processes. That enormous quantities of resinous rubbers are available is well known. That it is possible economically to purify resinous rubbers appears to be proved beyond doubt by the Guayule industry. I think there is a great future for the subsidiary industries which are likely to arise in this direction.

Chemistry of Crude Rubber, and the Question of "Synthesis."—According to the classical work of Harries, india-rubber is a ring compound, namely:



that is a polymer of 1.5 dimethyl-cyclo-octadiene.

I have already referred to the fact that according to the work of Harries, this polymer exists in at least two isomeric modifications. Although there is no apparent flaw in the chain of Harries' chemical reasoning, one is face to face with the fact that india-rubber has on several occasions been synthesised from isoprene by observers whose work must be regarded as beyond doubt. The synthesis by Euler of isoprene makes it apparently certain that this compound possesses the structure:



To reconcile Harries', Euler's, Tilden's and Bouchardat's work it is necessary to assume that the open chain compound isoprene in polymerising forms a ring compound, and that at the same time a somewhat unusual intramolecular rearrangement of linkages takes place. At the present there appear to me to be two main lines of work which are worth following up in this connection, firstly, the synthesis from isoprene and, secondly, that from laevulinic acid or its derivatives. If, as is anticipated, rubber can be obtained ultimately, in practically unlimited quantities, from the plantations and other sources at 1s. to 1s. 6d. per pound, no synthetic process which works at any appreciably higher figure is likely, in the long run, to prove a commercial success. Briefly put, the chemist must be able to produce rubber at something like 1s., otherwise the synthetic product cannot come to stay; such a price does not leave a great margin in the choice of raw material or cost of production. Synthetic indigo is frequently mentioned in discussions of the synthetic rubber problem, but the comparison is misleading from every point of view, but mainly because the lowest price at which natural indigo can be prepared is relatively speaking a very high one. With regard to the synthesis from isoprene, the problem appears to be the production of this substance at a very low cost, for it appears to be not unlikely that a satisfactory conversion of isoprene into rubber will ultimately be achieved. Another factor which may complicate the scientific problem is that it is necessary to produce not only a definite chemical substance, but that, in addition, this substance should have certain definite physical properties. It is, however, possible that these problems may be found to overlap. The differences in regard to mechanical strength which certainly exist as between different species and also as between rubbers of the same species prepared in different ways or derived from different localities, both in the case of the crude and the vulcanized articles, have not yet found a satisfactory explanation. In this direction a great deal of useful work may well be done.

THE MANUFACTURE OF RUBBER GOODS

The main processes employed in the manufacture of articles from crude rubber are still largely empirical, partly because the industry is comparatively

young and partly because the principles governing manufacture are by no means clearly defined.

The india-rubber article in the process of manufacture is subjected to a series of operations which are briefly as follows: First, the crude rubber as it arrives at the factory must be purified, and this is done by the process called washing. The washed rubber is subsequently dried. If ordinary "mechanical" goods are to be made the next process is that of mixing. This consists in incorporating with the rubber the necessary quantity of sulphur and filling materials. The next step as a rule is that of calendering; this consists in rolling out the well mixed and homogeneous dough to sheets of the required thickness. The calendered material passes next to the making-up table where it is cut to suitable sizes either for the hydraulic vulcanizing press or prepared for the molds. The next process is that of vulcanizing, which consists substantially in subjecting the calendered dough to combined heat and pressure, or heat only, in the press, in a mould, or in the open. In the case of manufacturing solid tires the rubber is not first calendered, but the well mixed dough is forced through an appropriate die. The same applies to squirted tubing. In the case of elastic thread the rubber is mixed with flowers of sulphur and "let down" with a solvent and the mass so obtained spread or calendered onto cloth. The above remarks refer to goods which are made by the hot process of vulcanization. There is, however, a large class, such as articles made from cut sheet, waterproof goods, and broadly speaking, all articles in which the rubber layer is very thin, which is prepared by the cold process. In this method the rubber is not first mixed with sulphur and then exposed to heat, but is subjected, in the form of a thin layer or sheet, to the action of a dilute solution of sulphur chloride in a suitable solvent or to the action of sulphur chloride vapors. The cold process cannot be applied to goods of any thickness because the action of sulphur chloride is so powerful that by the time the interior of a piece of rubber of any thickness is properly cured, the outer portions are over vulcanized.

SUBSTITUTES AND RESIDUALS.

Substitutes.—Although in the proper sense of the term no substitutes for india-rubber exist, there is a class of substances which is known in the industry as "substitutes" produced by vulcanizing vegetable oils such as colza, rape, maize and cotton-seed oils, either in the cold with sulphur chloride or hot with sulphur. Substitutes produced by the cold process are generally known as white substitute, those by the hot process as brown or French substitutes. They have been produced on a considerable scale for certainly 40 years and a goodly number of patents has been taken out in connection with their manufacture. As the price of these substitutes varies from about 3½d. to 9d. per

lb. their use effects a considerable saving in the cost of the rubber article in which they are incorporated. They are employed in order to cheapen the cost of production, and where the goods to be produced are of low specific gravity. The advantage of the substitute over mineral fillers is that the plasticity of the mixing is not affected to any extent, although the strength, wearing qualities, and elasticity are naturally much influenced. White substitute (which is the more expensive) is generally employed only for cold cured goods, but the brown may be employed for either. Many other substitutes have been suggested and some are used on a considerable scale, particularly in cable mixings. Gums, bitumen, vulcanized bitumen, nitrated oils, nitrated celluloses, ceresin, etc., etc., all find their sphere of utility in cable mixings, in insulating materials, valve rings, and so on, but a true rubber substitute has yet to make its appearance.

Residuals.—An enormous quantity of waste rubber finds its way into the market. If we assume that the crude rubber employed in the manufacture of goods in the British market yearly amounts to 12,000 tons, and that this reappears in the shape of finished rubber articles containing on the average perhaps not more than 30 per cent of rubber, it is plain that there must be something like 30,000 to 40,000 tons of waste rubber annually. The waste rubber industry is an important one in itself and a special organization exists for the collection of this material. Broadly speaking, waste rubber is utilized in one of three ways: (a) It is ground up (where the article permits of such simple treatment) and used direct in the ground state by the rubber manufacturer in his mixing for the cheaper varieties of goods. (b) It is subjected to one of the reclaiming processes which substantially consists in freeing the rubber from fibre, dirt and particles of metal, and subsequently removing as much as possible of the free rubber and non-rubber substances; and (c) the rubber is submitted to a re-forming process, the reforming being understood a process which employs substantially only heat and pressure.

Reclaimed Rubber.—In the old reclaiming processes the rubber was ground, the metal particles and vegetable fibre removed, and the mass then heated together with mineral oil and subsequently rolled out into homogeneous sheets. By this process no sulphur was removed. The newer processes all involve wet treatment and the removal of sulphur. Some of the reclaimed rubbers so made are of remarkably good quality; indeed, the better varieties, which are made from the best qualities of scrap, fetch appreciably higher prices than some of the inferior crude rubbers. In the oldest of the wet processes fibre is destroyed by means of sulphuric or hydrochloric acid, part of the mineral matter also being removed; the process is chiefly suitable for lightly cured scrap, such as that from proofed goods, as very little sulphur is removed.

In the alkali process, which is the one most generally adopted by large works in the United States and the United Kingdom, the scrap is first sorted, ground, metallic particles removed by magnetic separation, the ground rubber shaven in perforated trays to get it in an evenly divided state, and subsequently heated under pressure in iron vessels with alkali, washed and sheeted. It is claimed that in this way the whole of the free sulphur is removed. Another process is that in which neutral sulphites are employed for the removal of sulphur. A number of reclaiming processes have been proposed (some of which are, I think, working), in which, after mechanical cleaning, the whole rubber mass is dissolved in a suitable solvent; these solvents are generally of a neutral character and of high boiling points (petroleum, terpineol, etc.). From the solution so obtained the rubber is precipitated by means of a non-rubber solvent, such as alcohol. I have not personally come across any reclaimed rubber which does not still contain a very considerable proportion of sulphur.

Re-formed Rubber.—During the past few years a number of patents have been taken out by Gare, Im-misch and others for processes which effect the direct production of moulded articles from ground or flaked scrap mainly by means of heat and pressure. Exceedingly good results are obtained in this way, and it is remarkable that this simple method of dealing with "mechanical" scrap was not discovered at an earlier date. The fact remains, however, that particles of vulcanized rubber in the shape of dust flakes can be moulded by pressure and heat to a homogeneous mass, which on cooling, is to all intents and purposes indistinguishable from an ordinary moulded article.

There are two gypsum factories in the Argentine city of Cordoba, which, in view of the introduction into that community of electric power and a reduction in freight rates from there to Buenos Aires, promise to compete favorably with the gypsum imported from foreign countries. The cost of manufacturing gypsum per 1,000 kilos (2,204.6 lbs.) is shown to be as follows (in United States gold): Quarrying the stone, \$2.12; shipping to the kilns, \$2.46; manufacturing, \$2.97. The profit per 1,000 kilos is figured at \$1.36.

The annual report of an artificial silk factory near Frankfort, Prussia, according to a U. S. Consular Report, shows a clear loss of \$346,052 for 1910. This same company in former years paid as high as 35 per cent. Shares, which at one time brought as high as \$133 on the local stock market, have been as low as \$23 and are now selling at \$30. The price of artificial silk has dropped 60 per cent in the last few years, being now \$1.20 per pound.

DEVELOPMENTS IN PAINT AND VARNISH MANUFACTURE.*

By EDWARD C. HOLTON.†

Fifty years ago the paint and varnish industry of this country was in its infancy and was under no scientific control whatsoever. Even 25 years of ago the paint and varnish manufacturers who employed technical graduates as engineers or as chemists were the exception rather than the rule.

Conditions have greatly changed and today some of the larger companies have so branched out, in their attempts to provide themselves with raw materials of more uniformly good qualities, that they now have geologists, mining engineers, electrical engineers, mechanical engineers, chemical engineers, metallurgists and chemists constantly employed in looking after the various branches of their business. Even the smaller companies must make occasional use of the mechanical engineer and the electrical engineer, and since the recent activity amongst the State and Federal law makers, the chemist has become well nigh indispensable.

The mining, milling and transportation of lead and zinc ores from the mines to the smelter calls for the employment of the various engineers, but the analytical chemist or assayer must always be at hand. At the smelter the metallurgist or chemical engineer assumes charge of operations but he depends on his chemist.

The mining and milling of iron oxides, ochres, siennas, umbers, graphites, barytes, silicates, etc., are mechanical operations and are often carried on in a very crude manner, but somewhere and at some-time before the product is sold it is examined and rated by a chemist.

In the manufacture of the so-called "chemical" and "lake" pigments, chemists and chemical engineers are indispensable. Many of the older and better known pigments have been made for so many years that they could be made without the aid of trained chemists but the margin of profit is so small and the opportunities for losses and waste so great that the manufacturer can afford to be without a chemist. The development of organic chemistry has been marvelous, and to it the paint manufacturer is greatly indebted.

Many of the fast to light aniline, tollidine, anisidine naphthylamine and other lake pigments which were unknown or if known were laboratory curiosities 25 years ago are today being made in American color plants to the extent of thousands of tons per annum. The intermediate organic compounds which enter into these lake pigments are still largely imported from Germany. The mineral bases on which they are built are mostly of American manufacture and the

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diazotizing and combining of the organic compounds with each other and the mineral bases are generally under the direct supervision and control of chemists and chemical engineers trained in American schools.

Although much progress has been made in the manufacture of pigments, still more remains to be done. The field is a broad and interesting one and will offer opportunities for chemists for many years to come.

If the paint manufacturer of today were merely a mixer and grinder purchasing from others all pigments and vehicles which he uses, even under such conditions, he would be forced to maintain a chemical laboratory at his own plant or be in close touch with a commercial laboratory. If he did not do so he would not know the composition of his own manufactured products and would be liable to fine and imprisonment for unconscious violation of State and Federal laws.

The preparation and manufacture of paint vehicles is increasing enormously in kinds and in quantities, and presents an ever broadening field for the employment of chemists and chemical engineers. The lumbering of our Southern forests with attendant decrease of supplies of gum spirits of turpentine and consequent increase in price has brought about the recovery of wood spirits of turpentine from the waste logs, stumps and sawdust. It has also brought about the use in much larger quantities of coal tar distillates and petroleum distillates as substitutes for spirits of turpentine. Here again chemists and engineers have worked out processes of distillation which furnish products far superior to that which were on the market some years ago.

About 15 years ago China began supplying us in a small way with tung oil, and today its use is enormous and we would hardly know how to get along without it.

Two very poor flax crops have recently brought about a world shortage in linseed oil and chemists in all countries have been busy in investigating more thoroughly oils hitherto but little known, in order that they may be used as substitutes for linseed oil if of value, and on the other hand may be detected and identified when so used.

A few years ago the art of varnish making was clouded in mystery. Varnish makers taught their sons or favorite helpers all they had learned from their predecessors and by their own experience. Little by little they absorbed information given to them directly or indirectly by chemists, and profited by it, but rarely did they give anything in return. Today the varnish maker's attitude toward the chemist is rapidly changing. He recognizes the fact, which he is not always ready to admit, that if he would progress in his craft he must seek aid from the chemist. In many plants the chemist and varnish maker are working harmoniously together with the result that rapid advance is being made.

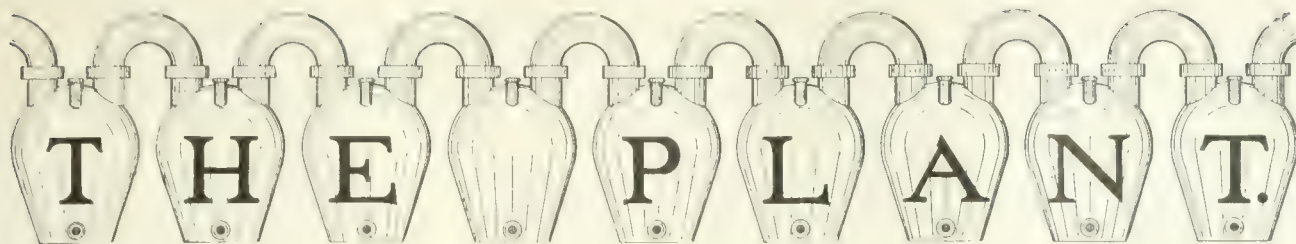
Of late years a great deal has been written and said about the chemical engineer. This is an engineering age and the chemical engineer is beginning to come into his own, and his future is bright.

However, it should not be overlooked that if the chemical engineer is not first and foremost a chemist, he might better avoid the use of the term "chemical." A man well trained in general chemistry with some skill as an analyst and synthesist, and with an active imagination, is fully as well equipped to advance the cause as a chemical engineer who has little knowledge of chemistry. Some chemical engineers are needed, and they are greatly needed, but there is even greater need for many well trained, thorough analytical and synthetical chemists.

A NOVEL METHOD OF OBTAINING PURE OXYGEN.

The chemical laboratory of the National Bureau of Standards recently had need of a large quantity of practically pure oxygen gas in carrying out certain work. The method of obtaining this gas is described in the *Scientific American* as follows: This gas was prepared in one of the physical laboratories of that institution by the electrolysis of water. The positive electrode was suspended in an ordinary round quart bottle with its bottom removed. This bottle was provided with a hollow metal stopper which formed part of the electrical connection between the positive electrode and the outside main. The electrolytic apparatus was placed outside the building, and as the hydrogen given off at the negative pole was not needed, it was allowed to go to waste.

The oxygen gas escaping through the hollow stopper under a pressure of only a few inches of water was conducted through a long copper tube to a purifier containing platinized quartz. Although this purifier was used constantly no impurities were found in the gas. From the purifier the gas was conducted through a valve-head to a two-liter cylinder of heavy brass lined with nickel. This cylinder was suspended in a bath of liquid air at atmospheric pressure. Fresh made liquid air boils at about 9° C., lower temperature than oxygen, and this difference was sufficient to cause the oxygen to liquefy. When the brass cylinder was filled with liquid oxygen, the current was shut off from the electrolytic apparatus, a common steel flask was connected up with the valve head, the liquid air bath removed, and the liquid oxygen at once began to vaporize owing to the heat received from the surrounding atmosphere. When the gage on the valve-head indicated sufficient pressure the valves were closed, and another flask was substituted for the filled one. The oxygen thus obtained was found more than 99.9 per cent pure. It will be seen that oxygen produced in this way is as near absolute purity as is possible. It is an expensive process, however, and cannot compete with the liquid air rectification process.



RELIABILITY OF MATERIALS.*

By WALTER C. FISH.*

The selection of this subject, which is overwhelmingly broad, is a reflection of admiration for the progress which increased reliability has afforded the industrial arts in their attempts to better supply the general needs of the community. A precise treatment should prominently deal, among other things, with statistics involving and contrasting the variations in performances of industrial materials and the scientific cause and cure of errors in them, but as such treatment, except narrowly, is impossible within the scope of these lines, I have preferred to proceed in a much more general fashion and with particular reference to some of the principle factors on which reliability depends. This unscientific treatment is less justifiable on this 50th anniversary of the Technology we honor if it fails in suggestion of the debt we owe our technical institutions and their product. In ordinary discussion, some such practical question as this might be asked: Do we, in the common affairs of life, individually and as a community, suffer marked delay, loss, or discomfort, because of failures of materials when rationally handled, or more simply, do our materials render the service we have the right to expect from them? It seems to me a fair answer to this question and one arising from common experience is that such discomforts and losses are relatively trivial if we omit those due to the deliberate and usually unwise use of materials for which reliability should not be claimed and from which it should not be expected—a social and economic question which is not herein considered—and if we omit the occasional faults of material in initial commercial development.

We travel at increasing speeds in heavier trains, wide rivers are spanned and harbors tunneled, high buildings are erected, intricate mechanisms serve us, and all such common things of today are done with a high degree of reliability from the view-point of the public. Correct design requires and concerns the interpretation of our natural laws and their expressions, and among these are the characteristics of our materials and that the skilled engineer is enabled to surround us with the facilities we enjoy is general proof of the enormous fund of exact knowledge

which science has already given for our safety and comfort.

If it be admitted that the public can view with some complacency the service it obtains from the materials more commonly employed, at least, it is none the less true that such results are far from easy and automatic in accomplishment, so to speak. When we enter behind the scenes and into our industries, we shall find that even those materials which by every good reason should be dependable become the source of loss, and that constant vigilance is necessary to insure the reliability we seek. While such losses may not be commonly large in percentage, it is certain that in the aggregate and over all industries the economic waste is huge, and the elimination of the causes of such losses directly concerns not only the pocketbook, but also the safety, of the public, since undetected error becomes a cause of danger. What then are some of the principal causes of these losses, and how can they be lessened?

When our scientific facts in any case are sufficiently well established, it is axiomatic that the common cause of loss is that which comes from imperfect skill in manipulation and direction. This statement is made with full appreciation of the splendid work of our skilled American artisan, so far as he is concerned, but it is none the less true that there is general failure to realize the opportunities for more precise and intelligent effort on the part of the individual. If we are to generally increase the uniformity of production, our industries must, in one way or another, provide themselves with more operations and methods which are beyond the likelihood of harm from human agency, or the standard of skill throughout our industries must be raised. Some operations may exist which may well be performed by so-called unskilled labor, but all manual operations permit some degree of dexterity, and most can be greatly enhanced in value by increased skill. This is a portion of the doctrine of the exponents of "efficiency" and "scientific management," who are stimulating splendid thought, but as we are occasionally a volatile race and easily pass from the threatened industrial invasion of Europe of a few years ago to a hysterical cry for increased efficiency—over night, so to speak—there is danger that the public—and we all are to be reformed as an essential part of the program—will overlook the time required for any real national change in mental attitude and disposition. A fundamental difficulty today,

*Paper presented before the Congress of Technology at the 50th Anniversary of the Granting of the Charter of the Massachusetts Institute of Technology.

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closely concerning the subject, is that too large a proportion of those annually entering the industrial army are quite prepared and content to undertake and continuously perform minor operations. This is a serious problem which society is not adequately solving. In such matters our progress will bear a distinct relation to the sane forces increasing the opportunities and intelligence of our present and, particularly, of our rising generation. The importance of skill and the good which may come from it is not sufficiently emphasized at any stage of life.

A further opportunity to lessen unreliability is opened by a better appreciation and use of existing knowledge and experience. Such impetus has been given scientific research by reason of the practical value of its own productions and so many facts of recent acquisition result therefrom, that the producer, unorganized to ascertain and use such facts, will continue to repeat the errors of the past, from which often all mysteries of cause might be lifted. The skillful framing of specifications for the control of methods and the selection of materials is by no means new, but it is a function of increasing difficulty and importance if the community is to be better served.

The evolution of new materials is constantly proceeding and surrounds us. There are perhaps one hundred so-called elements of nature with which to deal, and under the pressure of the chemist and metallurgist new combinations of these elements are passing as new materials into public service with such frequency that few, outside of the particular field of their own activity and save in exceptional cases, realize the progress. It is often true that these new combinations are replacing in character, and with slight changes in composition omit or counteract the causes for unreliability in their predecessors. Others appear with totally new characteristics and permit new arts. Even the pure elements of nature under the touch of our research laboratories continue to unexpectedly pass into the ranks of useful and reliable materials. Very recently, the element tungsten, as a pronounced example, had not value when used alone. It was a worthless and unworkable metal unless alloyed. The proposition to produce from it a ductile and tenacious wire might naturally have excited ridicule, but today such fabrication has been accomplished, and tungsten wires for incandescent lamps are used to great economic advantage and increased satisfaction, and such results, of course, are practical measures of reliability. The element boron has recently been separated and found to possess characteristics of such possible value that in due time it may find important uses. If we consider uniformity of performance to be the criterion of reliability, this mere evolution of new materials might appear to have no direct bearing on the subject, but as no such work can be intelligently undertaken and accomplished without adding to our

existing knowledge of materials and their characteristics, there is, after all, a close relationship.

Furthermore, the unwise and premature exploitation of these new materials from time to time is a cause of loss which is seldom excusable. Some materials, like mankind, unexpectedly take on infirmities with age and service and from causes which are not yet understood, but it is a function of the manufacturer to determine all practical characteristics of his new materials before public service is undertaken, and this he can generally do. The public itself, by its premature desires, is at times at fault. To obtain some new facility or pastime it permits itself to be transformed into a great testing laboratory, and attempts the use of undeveloped mechanisms and materials. Such public demand, however, stands for speed in accomplishments. The automobile of a few years ago was a pronounced illustration of premature exploitation, but, to its credit, it forced the development, not only of principles of design, but of the production of more reliable materials without which the art could not safely proceed. The same public desire has existed for increased speed of transportation, and the sanity with which this desire has been met, though involving problems of great difficulty, is shown by the relative absence of accidents of transportation due to failure of material.

If I have not attempted within these limited lines to recite in precise fashion specific measurements of reliability and unreliability, it was to better suggest the good to come to us by the collection, diffusion, increase and use of scientific facts. It was for such purposes, among other things, that the wonderful so called Alexandrian Museum was founded some 2,200 years ago. Few deliberately concern themselves nowadays with Schools of Philosophy and Thought, but it is interesting to quote the scientific methods as they then and there existed, and to suggest that the partial hiatus in those methods over a period of centuries was perhaps responsible for a halt in the benefits coming from applied science.

"The essential principle of the Aristotelion philosophy was, to rise from the study of particulars to a knowledge of general principles or universals, advancing to them by induction. The induction is the more certain as the facts on which it is based are more numerous; its correctness is established if it should enable us to predict other facts until then unknown. This system implies endless toil in the collection of facts, both by experiment and observation; it implies also a close meditation on them."

We should be a nation of blood and brain, and such organisms may resist suddenly applied pressure and demand the exercise of patience. Molecules and atoms are pliable to the hands of the scientist and can continue to be forced to rapidly enhance the conditions of life.

THE CORROSION OF BRASS FOUNDRY PRODUCTS.*

By WILLIAM VAUGHAN.†

Many of us no doubt have had our own experiences with the rusting of iron and steel, the leaking of steam and water pipes and the corroding of the various alloys of copper. The problem is continually arising: what composition or alloy under given conditions will offer the strongest resistance to these corrosive forces. Comparative tests under the exact conditions of service afford the most valuable information, but in most instances the expense of such tests and the length of time necessarily involved are prohibitive. We are therefore forced to obtain a solution of this problem by some quicker and simpler means.

The term "corrosion" we may define as "the effects produced upon a metal by the combined action of air and water, with or without the stimulus provided by the various impurities in the water or in the atmosphere". These corrosive effects are explained on the basis of: (1) Chemical action alone; (2) Electrochemical action, which may be caused by stray electrical currents from an outside source or by galvanic action due either to included impurities or to the varying proportion of the different elements constituting the alloy.

The main difference between two actions is in intensity; the corrosive action itself is practically the same in both cases, unless abnormal results are produced by high current densities. Effects that take months under the first can be accomplished in six or eight hours by the second. By accelerating the chemical action by an electric current, we have therefore a comparatively simple method of determining the relative resistance to corrosion of the various metals.

The relation also between corrosion and the constitution; that is, crystallographic structure, of an alloy is very marked. Shepard & Blough found that when alloys of copper and tin were annealed to equilibrium between 200° C. and 400° C. (Figure 10), there appeared five solid phases. The series of solid solutions from 100 per cent to 87 per cent of copper consists of alpha crystals. From 87 per cent to 74.5 per cent copper there are varying proportions of alpha and beta crystals. From 74.5 per cent to 67 per cent the beta crystals alone exist. Between 67 per cent and 61.3 per cent copper there is a mixture of the compound Cu_3Sn with the beta crystals; from 61.3 per cent copper to 41 per cent, a mixture of the compound Cu_3Sn and gamma crystals; from 41 per cent to 40 per cent copper, the gamma crystals only, and from 40 per cent to 0, pure tin and gamma crystals coexist.

From this we see that there may be a radical change in the crystalline structure of an alloy with a very small change in composition. It is easy to understand

then why the same alloy in the same solution, but varying slightly in composition, may be affected so differently. For example, in changing from a bronze made up wholly of alpha and beta crystals, to one consisting of a mixture of the compound Cu_3Sn and of beta crystals, we have a change from a corrosive alloy to one that is but little affected by a sulphate solution, or change from an alloy made up of the compound, Cu_3Sn and of beta crystals to one of pure gamma crystals, and we change from an alloy that resists corrosion to one that is easily affected. A knowledge of the crystalline structure therefore aids in understanding why alloys of slightly different composition may corrode so differently under similar conditions.

The methods of studying the corrosion of different alloys which naturally suggest themselves are: (1) Visual observation; (2) Loss in weight method; (3) Loss in weight and the analysis of the corrosion product. The first method consists of simply observing the action of different solutions, whether acid, alkaline, or neutral, on metals of different composition. This is only a rough qualitative method and may be very misleading.

The "loss in weight" method consists of weighing the sample before beginning the experiment and again at the end of the experiment, taking care to remove all scale. The figures thus obtained give the loss of metal by corrosion or the percentage of corrosion. For example, practical experiments comparing the corrosive effects on various alloys of sulphite acid, such as is used in pulp mills, were carried out as follows:

The different alloys were placed on a piece of wood in a small digester, covering with sulphite acid and heated for three days at a temperature of 150° C. The results obtained were as follows:

Alloy.	Original weight in grams	Weight after heating, grams.	Loss in weight, grams.	Percentage loss.
(1)	190.372	184.300	6.072	3.189
(2)	179.937	178.750	1.187	.659
(3)	123.222	120.210	3.012	2.444
(4)	96.496	96.480	.016	.016
(5)	100.889	99.930	.959	.955
(6)	89.195	84.900	4.295	4.81

- Notes: (1) This sample dissolved rapidly and left a perfectly clean surface.
 (2) This sample was covered with a heavy coat of copper sulphide, which formed a protecting film.
 (3) Sample badly etched, no protective coating produced.
 (4) A very thin protective film formed almost immediately.
 (5) This metal also had a protecting film.
 (6) Sample corroded very rapidly with no protecting coating.

By means of the third method; i. e., "loss in weight and analysis of the corrosion product," not only the amount of corrosion but also the actual percentage of each element corroded can be determined. It is by this method that some of the most valuable corrosion investigations have been carried out.

By the electrolytic method using low current densities several investigators (Messrs. Curry, Lincoln and Rowland) have carefully worked out the effects of

*Paper presented before the American Brass Founders' Association.

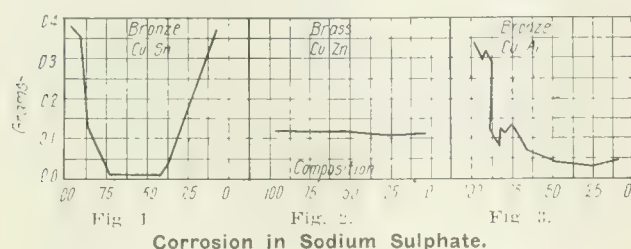
†With Arthur D. Little, Inc., Chemists and Engineers.

different salt solutions in the corrosion of different alloys.

It may be of interest to compare some of these effects as gathered from the data compiled by these men, which is shown by the accompanying diagrams. In the accompanying figures the ordinates represent the amount of corrosion in grams and the abscissae

Alloys of copper and tin; copper and zinc; copper, tin and zinc; and copper and aluminum were tested ranging in composition of copper from 95 per cent to 5 per cent.

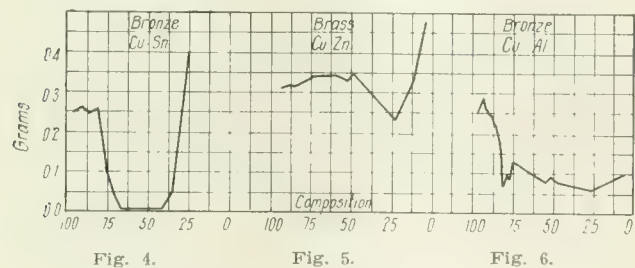
The effect of electrolytic corrosion of the bronzes was determined in 7 per cent solutions of sodium sulphate (Na_2SO_4), sodium nitrate (NaNO_3), and sodium chloride (NaCl), using a current of 25 milliamperes; whereas the bronzes were examined in normal solutions of these salts.



Corrosion in Sodium Sulphate.

The amount of corrosion in the copper-tin bronzes of 95 and 90 per cent copper content is very heavy in sodium solution (Fig. 1), and decreases rapidly until the 70 per cent alloy is reached. From 70 per cent to 40 per cent bronzes there is so little corrosion that the metal remains apparently unchanged; whereas the amount of corrosion from 35 per cent to 5 per cent again increases very rapidly.

The amount of corrosion in the copper-zinc bronzes in sodium sulphate solution (Fig. 2) is practically the same throughout the series, and the composition of the corrosion product is practically the same as the brass itself, from 93.6 to 51.3 per cent copper. Below



Corrosion in Sodium Nitrate.

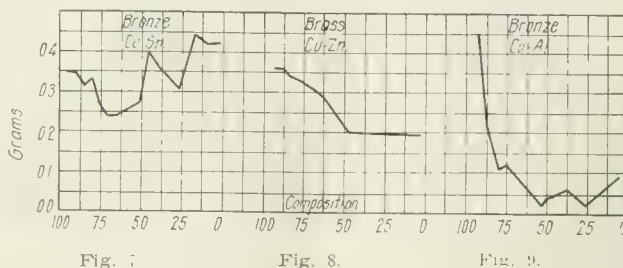
47.6 per cent the corrosion product consists mainly of zinc with very little copper.

The copper-tin-zinc bronzes in the same solution corrode in a manner but slightly different from the straight bronzes, except that there is a decrease in the amount of corrosion. Additions of small amounts of tin to an alloy affect the physical properties by increasing its hardness, rigidity, elasticity and tensile strength.

The aluminum bronzes (Fig. 3) in sodium sulphate solution dissolve almost according to their composi-

tion along the range from 95 per cent to 60 per cent copper. From 55 per cent copper in the alloy to 35 per cent there is a gradual falling off in the amount of corrosion, whence it rises again as pure aluminum is approached.

The copper-tin bronzes (Fig. 4) in sodium nitrate solution act very much as they do in sodium sulphate,



Corrosion in Sodium Chloride.

the 95 per cent to 90 per cent bronzes dissolving very readily; the 85 to 70 per cent dissolve less easily on account of the protecting film that is formed. The 65 to 40 per cent bronzes are affected but little by the nitrate solutions, and the thin film that forms is not easily broken down. The 40 to 5 per cent bronzes all corrode, the corrosion being slight at the 40 per cent, and increasing rapidly as we approach pure tin, due probably to disintegration.

The amount of corrosion of the copper-zinc bronzes (Fig. 5) in the nitrate solution gradually increased with the decrease in the copper content until the 47.6 per cent brass is reached, when it falls off very rapidly to 22.6 per cent copper in the metal. From the

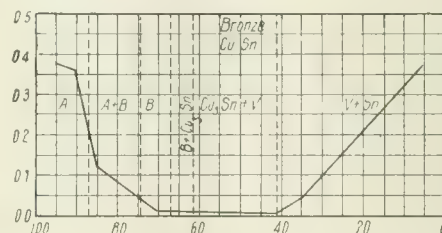


Fig. 10—Equilibrium Balance.

22.6 per cent brass to pure tin the increase in the amount of corrosion is very marked. This seems to be due to the fact that some of the crystals are dislodged by the body of the metal being dissolved away, thus leaving them to fall into the electrolyte.

The amount of corrosion in the 95 to 50 per cent bronzes containing 1 per cent of tin is in general the same as that of the pure bronzes, increasing slightly with the decrease in copper content. (Tin was only added to bronzes of high copper content, 95 to 50 per cent.)

Aluminum bronze (Fig. 6) in sodium nitrate solution: The 95 per cent bronze has the greatest solubility and this decreases slightly with the decrease in copper content to the 75 per cent bronze, whence it rises to the 65 per cent and remains almost constant to 55 per cent. From the 55 per cent bronze it decreases to the 25 per cent, and from there to pure aluminum the corrosion increases rapidly.

The copper-tin bronzes (Fig. 7) in sodium chloride

solutions do not become passive as they do in the other solutions (Figs. 1 and 4). There is a continuous changing in the amount of corrosion from 95 per cent copper in the sample to the 5 per cent copper.

The amount of corrosion in the straight brasses (Fig. 8) in sodium chloride solutions decreases very rapidly with the decrease in copper down to about the 50 per cent brass. From 50 per cent copper to pure zinc the amount of corrosion remains practically constant. On examining the corrosion product of the brasses of high copper content, 95 to 50 per cent, it was found to be a heavy red scale which in most cases was easily removed. The precipitate on the brasses of low copper content was pure white, mostly zinc.

When three-quarters to one per cent of tin was added to brasses they behaved almost exactly like the pure copper-zinc. (Tin was only added to brasses from 95 to 50 per cent copper.)

In this connection the experiments made by Lincoln on brasses using solutions of common salt, bath salt and synthetic sea salt indicated that they corroded practically alike and agreed with his former experiments with solutions of pure sodium chloride.

The amount of corrosion of aluminum bronzes (Fig. 9) in the chloride solution decreases somewhat irregularly, with the decrease in copper content from the 95 per cent copper to 26 per cent copper, whence it rises rapidly to pure aluminum. Between the 40 per cent bronze and pure aluminum there is no copper in the corrosion product.

From these results we can compare not only the action of the same alloy in different solutions, but also the action of different alloys in the same solutions. Take for instance the corrosion of the copper-tin bronze in different salt solutions (Figs. 1, 4, 7). It is apparent that this composition is affected more in the sodium chloride solution than in either of the others. There is also a very striking difference in the way the brasses (Figs. 2, 5, 8) act in different salt solutions. They are least affected by sodium sulphate. In the aluminum bronzes (Figs. 3, 6, 9) there is a marked similarity in the corrosion curves of the different solutions.

A comparison of the different alloys in the same solution is interesting. Comparing the different alloys in the sulphate solution (Figs. 1, 2, 3) it is clear that the copper-tin bronze (Fig. 1) is the least affected, and therefore the best to use in a solution of this kind.

Comparing Figs. 3, 4 and 5, the corrosion in sodium nitrate, we still find the copper-tin bronze the best metal to use, since there is practically no corrosion between 75 per cent and 40 per cent copper content.

Figures 7, 8 and 9 show us that the aluminum bronze is affected the least in sodium chloride. This bears out practical tests reported by Sinclair at the Manchester meeting of the Institute of Metals in 1909 in which he found aluminum bronze the most resistant to salt water.

As a practical example of overcoming condenser tube corrosion trouble, where salt water is used for cooling purposes, we may cite certain experiments carried out by us.

On May 15, 1900, nine different kinds of tubes were installed in a surface condenser, twenty-five (25) of each kind being inserted in vertical rows and designated alphabetically by punch marks at the top of the row. The remaining tubes in the condenser, some 3,400, were of "admiralty metal" (untinned), having the following composition:

	Copper	70%		
	Zinc	29%		
	Tin	1%		
The tubes installed (25 of each kind) were as follows:				
Mark.	Composition.	No. of tubes removed.	Date of removal.	
"A"	Copper	70%	1	Sept. 5
	Zinc	29%		
	Tin	1%		
	Tinned			
"B"	Copper	60%	None	
	Zinc	40%		
	Untinned			
"C"	Copper	60%	None	
	Zinc	40%		
	Tinned with pure tin			
"D"	Copper	60%	None	
	Zinc	40%		
	Tinned with a mixture of 2 parts tin and 1 part lead			
"E"	Pure copper		1	Aug. 15
	Untinned		24	Aug. 21
"E"	Pure copper		1	July 23
	Tinned		1	Aug. 1
			1	Aug. 17
			22	Aug. 20
"G"	Copper	85%	None	
	Nickel	15%		
"H"	Copper	66%	None	
	Zinc	33%	None	
	Untinned			
"I"	Copper	66%	1	June 30
	Zinc	33%	1	July 30
	Tinned		1	Sept. 6
			1	Sept. 7

In all cases the failure took place by severe pitting and grooves running lengthwise of the tubes. In many cases small holes penetrated through the entire thickness of the tube. It will be noted that a large part of the failures were with tinned tubes, due probably to the electrolytic action set up between the metal and the tin wherever defects occurred in the tin coating.

These results bear out in a general way the deductions drawn from the laboratory experiments previously described. The pure copper tubes were particularly susceptible to corrosion, while the copper-zinc tubes were much more resistant. The 60/40 and the 66/33 tubes showed practically no difference, which would be expected as little difference in corrosion was found by the "electrolytic method" previously referred to between brasses of these compositions. The so-called "admiralty metal" appears to resist corrosion better than the other brasses and this too was indicated in the experiments, as the addition of a small amount of tin tended in most cases to diminish the rate of corrosion.

The facts which we have presented indicate that an accurate idea of the relative corrosion of alloys can be obtained by laboratory experiments. A large amount of experimental work still remains to be done on the numerous alloys which are manufactured today, but it is to be hoped that investigations will be carried out to such an extent that in the not distant future the employment of alloys which are rapidly destroyed by corrosion will be due to carelessness or willful intention and not to lack of proper information.

CORROSION TABLES.
BRONZE (Fig. 1).

Percentage of copper in test.	Corrosion in grams.	Grams copper corroded.	Percentage of copper corroded.
95	0.3787	0.3638	96.07
85	0.1289	0.1245	97.03
75	0.0416	0.0400	96.1
65	0.0061	0.0036	
55	0.0037	0.0030	
45	0.0049	0.0039	
35	0.0410	0.0126	30.1
25	0.1579	0.0213	3.3
15	0.2629	0.0224	8.5
5	0.3707	0.0121	3.2

BRASS (Fig. 2).

Percentage of copper in test.	Corrosion in grams.	Grams copper corroded.	Percentage of copper corroded.
93.6	0.1184	0.1069	90.3
86.6	0.1184	0.1012	85.5
83.6	0.1177	0.0888	75.5
73.4	0.1188	0.0845	71.2
60.3	0.1214	0.0688	56.7
51.3	0.1136	0.0357	31.4
47.6	0.1132	0.0015	1.33
22.6	0.1031	0.0010	0.97
10.5	0.1694	0.0003	0.28
3.1	0.1116	0.0002	0.18

ALUMINUM BRONZE (Fig. 3).

Percentage of copper in test.	Corrosion in grams.	Grams copper corroded.	Percentage of copper corroded.
96.0	0.3426	0.3220	94
86.4	0.2945	0.2713	86.01
76.0	0.1082	0.0667	61.7
65.4	0.0703	0.0460	63.0
55.0	0.0430	0.0196	43.00
46.6	0.0401	0.0161	40.14
35.2	0.0352	0.0134	38.08
25.0	0.0364	0.0123	33.79
15.1	0.0408	0.0114	27.94
10.0	0.0451	0.0125	27.71

BRONZE (Fig. 4).

Percentage of copper in test.	Corrosion in grams.	Grams copper corroded.	Percentage of copper corroded.
95	0.2528	0.2452	97.0
85	0.2513	0.2471	92.0
75	0.1143	0.1000	98.2
65	0.0049	0.0038	
55	0.0037	0.0024	
45	0.0027	0.0032	
35	0.0491	0.0077	15.7
25	1.7904	0.3783	21.7
15	1.2990	0.1824	14.0
7	1.0752	0.0433	4.3

BRASS (Fig. 5).

Percentage of copper in test.	Corrosion in grams.	Grams copper corroded.	Percentage of copper corroded.
93.6	0.3091	0.2877	95.2
86.6	0.3191	0.2775	86.9
83.3	0.3173	0.2645	83.4
73.4	0.3325	0.2481	75.2
60.3	0.3452	0.2105	61.0
51.3	0.3375	0.1812	53.7
47.6	0.3504	0.1516	43.2
22.6	0.2352	0.0002	0.0
10.5	0.3316	0.0228	6.8
3.1	0.4804	0.0174	3.6

ALUMINUM BRONZE (Fig. 6).

Percentage of copper in test.	Corrosion in grams.	Grams copper corroded.	Percentage of copper corroded.
96.0	0.2498	0.2440	97.60
86.4	0.2409	0.2151	89.29
76.0	0.0876	0.0642	73.29
72.4	0.1253	0.0908	72.46
55.0	0.0822	0.0460	55.90
45.6	0.0805	0.0242	30.06
35.2	0.0640	0.0217	33.91
25.0	0.0591	0.0141	23.06
15.1	0.0653	0.0129	19.75
10.0	0.0759	0.0123	16.21

BRONZE (Fig. 7).

Percentage of copper in test.	Corrosion in grams.	Grams copper corroded.	Percentage of copper corroded.
95	0.3486	0.3318	94.9
85	0.3216	0.2775	86.3
75	0.2793	0.2694	75.4
65	0.2462	0.1611	65.4
55	0.2598	0.1402	54.0
45	0.4021	0.1802	44.8
35	0.3693	0.0525	14.2
25	0.3051	0.0584	19.1
15	0.4425	0.0448	10.1
5	0.4237	0.0169	3.8

BRASS (Fig. 8).

Percentage of copper in test.	Corrosion in grams.	Grams copper corroded.	Percentage of copper corroded.
93.6	0.7204	0.6756	93.8
86.6	0.7165	0.6263	87.4
83.3	0.6925	0.4993	76.6
73.4	0.6460	0.5117	79.9
60.3	0.5773	0.3417	59.2
51.3	0.4618	0.0990	21.4
47.6	0.4077	0.0077	1.3
22.6	0.4042	0.0006	0.2
10.5	0.3975	0.0004	0.01
3.1	0.3972	0.0001	0.00

ALUMINUM BRONZE (Fig. 9).

Percentage of copper in test.	Corrosion in grams.	Grams copper corroded.	Percentage of copper corroded.
96.0	0.5623	0.5345	95.1
86.4	0.2124	0.1291	60.78
74.8	0.1222	0.0323	26.4
58.8	0.0555	Trace	
57.9	0.0207	Trace	
45.6	0.0511	Trace	
35.2	0.0579	0.0000	00.0
26.0	0.0265	0.0000	00.0
15.0	0.0316	0.0000	00.0
4.5	0.1000	0.0000	00.0

According to publications devoted to wine growing in France, that country is in need of large quantities of nicotine to combat successfully the ravages of the cochylis and other cryptogamic diseases which have wrought such damage to French vineyards in the past. Although effective methods were adopted to destroy these insects (the cochylis and eudemis) by peeling the bark of the vine, by scalding the stubs, and by the application of different chemicals, yet it was impossible to prevent heavy damages. The two remedies that have proved to be by far the most efficacious are arsenate of lead and nicotine sprayed over the entire vine. Owing to the nonvolatile character of the former it can be used only in the springtime before the fruit forms. Nicotine alone is found effective after the fruit appears. It is this quality that gives this alkaloid its special importance; it is indispensable because there is nothing that can replace it.

DIFFERENTIAL DISTILLATION.*

By J. LOUIS FOUCAR.

Some time ago I designed the column, a sectional drawing of which is seen in Fig. 1. It consists, essentially, of a helical septum, or septa (a), which traverses the annular space (b), between two concentric cylinders (c) and (d). The inner cylinder (c) can be used as a thermostat in various ways: for example, by allowing a suitable substance to boil therein under the necessary pressure; so that the column may be used as a constant-temperature still-head for the preparation of specially pure substances, for which there is an ever increasing demand. It is not necessary or desirable to fit the column itself with a reflux condenser, but it is preferable to make this an accessory part of the thermostatic device, and directly attached at (e) to the inner cylinder.

Assuming that we start with the column cold, and without making use of the inner cylinder, the action of the column may be described as follows: On suitably changing the temperature and pressure of the substances contained in the vessel, vapors are evolved, and pass up the spiral passage. The vapors normally tend to condense in the order of their boiling points, that is, the vapor of the substance of highest boiling point will condense first, and the liquid so condensed will flow backwards down the spiral, some upon the upper side, or top, of the spiral passage, and some upon the lower side, or bottom. In condensing, the latent heat of the condensation is transmitted both upwards and downwards through the material of the inclined septum, and the latent heat evolved in one spiral turn causes the re-volatilization of some of the constituent or constituents of lower boiling point in the already condensed liquid which is flowing down the upper side of the inclined plane in the spiral turn immediately above it and in that which is flowing down the underside of the inclined plane immediately below. Thus a continuous, simultaneous differential vaporization of the relatively lighter constituent, or constituents, and condensation of the heavier constituent, or constituents, takes place, with the result that, by choosing suitable conditions, the constituents may be successively isolated. The apparatus can, of course, be used as a continuous still in the same way as modern alcohol and ammonia stills. Its function in this case is substantially as already described.

In Fig. 1 (f) is the vapor inlet, (g) the vapor outlet, (h) the run-off for the liquid in the thermostat, and (i) the return for the refluxed liquid.

One of the many advantages of this column is that if we start cold we may not only work entirely without the usual reflux condenser, but can actually lag the column without any loss of efficiency. Experiments have demonstrated the truth of this assertion.

As it is of considerably smaller size than other column of like capacity and efficiency, it is evident that a great saving in fuel and consumption is effected by its use. It is perfectly continuous in action throughout its whole length, and those mathematicians among us who have studied the counter-current will tell us that this is the only way to obtain the maximum efficiency out of such an interaction tower. I have hence called it the "differentiator" column, and its function "differential distillation."

The advantages of this appliance over others of like character are as follows:

- (1) Smaller size required.
- (2) Moderate cost.
- (3) No clogging or blocking possible.
- (4) No back pressure thrown on to the still, except that due to the mere height of vapor in the column, a

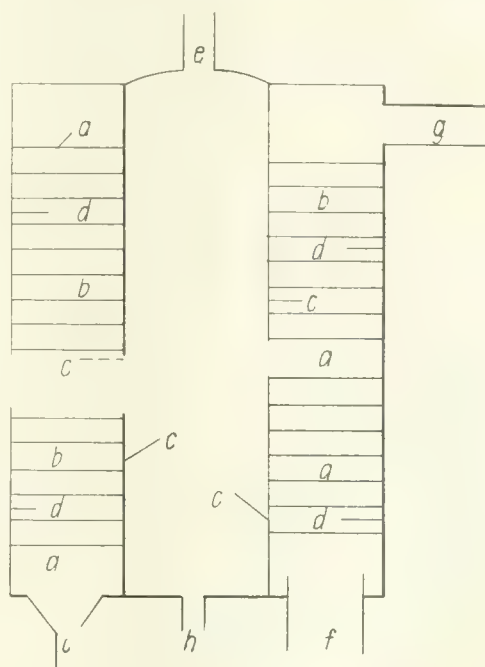


Fig. 1.

great advantage in the distillation of substances which can, or must, be distilled under reduced pressure.

- (5) Saving in fuel.
- (6) Increased purity of products.
- (7) Great sensibility and consequent easy control.
- (8) On completion of the distillation and subsequent cooling of the column, the latter retains no liquid, as it is able to drain completely. This is of great advantage, as every practical distiller must know.
- (9) Simplicity of construction and consequent reliability and lowness of the costs of maintenance.
- (10) Safety.

Other advantages could be enumerated. The only possible disadvantage of which the author has any knowledge or suspicion is the comparative intimacy of contact, which may not be quite so good as in dephlegmators. What is meant in this. The column will have a steady temperature-gradient falling from the still to the outlet of the still-head. At any point

*From Journal Society of Chemical Industry.

in the still-head this temperature will correspond to a certain and calculable composition of the mixed vapors. This is what one may term the static equilibrium.

When the apparatus is at work, however, the two counter-currents will cause the equilibrium to lag behind its corresponding temperature, and the measure of this lag, which the author intends to investigate under various conditions, should be of very great interest. In dephlegmators, properly designed and properly used, the kinetic equilibrium—if one may so call it—should very nearly approach the static, and in this one respect they may possess the only advantage over the “differentiator” column.

I have effected a most difficult separation of three isomeric substances by systematic fractional distillation with the use of my apparatus, a result which, so far as I am aware, has not been previously achieved by distillation alone.

Professor Sydney Young used for the test of a still-head a mixture of equal weights of benzene and toluene. It was, however, more convenient for me to work with equal volumes of those substances, pre-

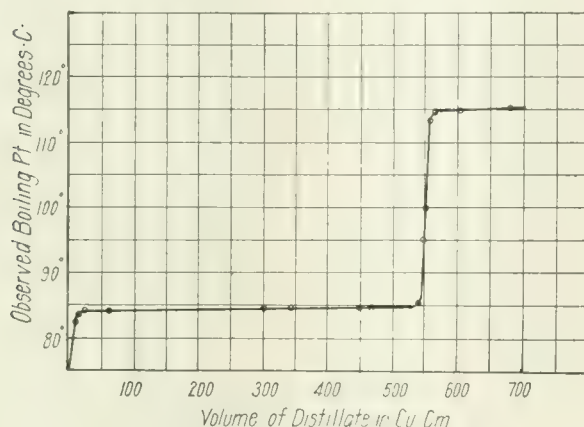


Fig. 2—Distillation of Equal Volumes of Benzene and Toluene.

Volume = 550 + 550 c. c. s. Vol. of dist. up to mean of B. Ps. = 551 c. c. Vol. of dist. residue beyond mean of B. Ps. = 545 c. c. Vol. of dist. below 79° C. = 25 c. c. Vol. of dist. B. P. 79-80° C. = 522 c. c. Vol. of dist. B. P. 80-109° C. = 15 c. c. Vol. of dist. and res. B. P. 109-110° C. = 536 c. c. Loss in distillation less than 1½%.

pared from commercial 90 per cent benzol by means of a “differentiator” column, which is not fitted with any thermostatic device.

The curve (Fig. 2) gives the volumes of the distillate in cubic centimeters as ordinates and the temperature in degrees centigrade of the vapors condensing as abscissae. The intermediate fraction, whose volume is independent of the volume of the material with which the experiment was conducted, was only 15 cubic centimeters, or in this particular case less than 1½ per cent by volume of the whole. Had I used twice the volume, the intermediate fraction would have been no larger, but the proportion would have been less than ¾ per cent by volume.

The rate of distillation was about 250 cubic centimeters per hour, but considerably faster than this when only pure benzene or toluene was being run.

IMPROVEMENTS IN COTTON BLEACHING.*

By WALTER S. WILLIAMS.†

Bleaching is the whitening or removing of color from a substance, but in the textile industry the term is broadened to include the removal of all foreign bodies and coloring matter adhering to or included in the original fiber. In the case of fabrics, it also covers the elimination of all material, such as starch, oil and the like, added either intentionally or accidentally in the process of manufacture.

Cotton was used for the production of cloth thousands of years before the Christian era, and bleaching of the fabrics so made must have been more or less crudely performed even at that early age. While interesting, the scope of this paper will not permit the tracing of the early history of the art. Ancient processes will therefore be cited only as they may have some bearing upon modern practice.

An efficient process of bleaching requires a thorough understanding of the nature and chemical composition of the fiber and of the substances to be removed, as well as a knowledge of the effect produced by the different agents to be employed. It is a remarkable fact that very few complete studies of the process as a whole exist, and that the few careful investigations of different steps in the procedure have not received the attention and consideration they deserve at the hands of the practical manufacturers. There is no doubt that great progress will be made in the improvement of both processes and results by careful and scientific research in this field.

At the very outset of our study we are met by the fact that the chemistry of cellulose, which composes 83 to 90 per cent of the crude cotton fiber, is still after many years of investigation far from being clearly defined. Two series of analyses conducted by the Department of Agriculture gave average results as follows:

Fertilizing constituents:

Water	6.07
Ash	1.37
Nitrogen	0.34
Phosphoric acid.....	0.10
Potash	0.46
Soda	0.09
Lime	0.19
Magnesium	0.08
Ferric oxide.....	0.02
Sulphuric acid.....	0.6
Chlorine	0.07
Insoluble matter.....	0.05

Proximate constituents:

Water	0.74
Ash	1.65
Protein	1.50
Cellulose	85.71
Nitrogen-free extract.....	5.79
Fat	0.61

Ash from cotton:

Potassium carbonate.....	44.8
Potassium chloride.....	9.9
Potassium sulphate.....	9.3
Calcium phosphate.....	9.0

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Calcium carbonate.....	10.6
Magnesium phosphate.....	8.4
Ferric oxide.....	3.0
Alumina and loss.....	5.0

Similarly, the precise change produced by alkalis and acids has not been determined and the extent to which such treatment may be carried without injuring the strength and lasting properties of the fabric is now rather a matter of observation and practical experience than an actual science.

For a knowledge of the impurities present in crude cotton, we are indebted to a study of these bodies by E. Schenck. About 5 per cent of the original fiber is removed by boiling in soda ash for eight hours and may be thrown down from the resulting dark brown liquor as a copious, light brown precipitate by adding an excess of dilute sulphuric acid. If dried, it produces a brittle, horn like mass which leaves upon incineration an ash varying from 2.3 to 6.9 per cent. From this residue Schenck succeeded in isolating substances which he classifies as follows:

1. A wax-like body which he named *cotton wax*. It is lighter than water, is soluble in ether and alcohol, but insoluble in water and only slightly dissolved by boiling dilute caustic soda.

2. A *fatty acid* which melts at 55.5 degrees Centigrade and may be either margaric or a mixture of stearic and palmitic acids.

3. Two nitrogenous, amorphous, brown *coloring matters*, one of which is easily soluble and the other sparingly soluble in alcohol.

4. *Pectic acid*, obtained in the form of a light yellow amorphous substance resembling gum or gelatine. This body forms much the larger part of the precipitate, the other constituents being present only in extremely minute quantities.

5. Albuminous matter.

Many others have studied more or less thoroughly the nature of the substances extracted by water, ether or other solvents, as well as the nature of the ash present in the raw cottons, but without more definite results.

In cotton bleaching, we are confronted with the problem of the removal of these various impurities and of the decolorizing or whitening of any yellow or brown tint still remaining. Broadly speaking, the process consists of boiling with alkali liquors to remove all impurities made soluble by such treatment and a subsequent oxidation for the removal of residual tint.

Cotton may be bleached in almost any form from loose or raw cotton to cotton piece goods, the difference in treatment required in each case being purely mechanical. As the bleaching of cotton piece goods far exceeds the importance of the bleaching of any other form of cotton, we will confine our attention to that form. We will also assume in the succeeding remarks a knowledge on the part of the reader of the terms and processes of practical bleaching. The

complete process used with slight modifications for the past 50 years may be outlined as follows:

1. *Singeing*.
2. *Grey Wash*: Wet out in washing machine, allow to lie for 24 hours and wash.
3. *Lime Boil*: Pass through milk of lime and boil for 12 hours under pressure.
4. *Lime Sour*: Wash and pass through hydrochloric acid at 3 degrees. Twaddle, squeeze and allow to lie in sours for 6 to 8 hours. Wash thoroughly.
5. *First Lye Boil*: Boil 6 to 8 hours with soda ash and resin soap. Wash in kier and through washer.
6. *Second Lye Boil*: Boil in kier as for first boil using soda ash and a small amount of caustic. Wash in kier and through washer.
7. *Chemic*: Pass through hypochlorite of lime solution at $\frac{1}{2}$ to 1° Twaddle. Allow to lie in chemic from 6 to 8 hours.
8. *Sour*: Wash and pass through sulphuric acid at 1 to 2 degrees Twaddle.
9. *White wash*: Wash until all traces of acid are removed, open and dry. This process in practice will occupy from 5 to 10 days.

The first step in the process of bleaching cloth for printing or other purposes requiring a smooth face is the operation of singeing, which has for its object the removal of all loose hairs from the surface of the cloth. This step is largely mechanical, but requires close attention to produce satisfactory results without damage to fiber or undue cost for gas or fuel. The most approved method, known as gas singeing, passes the fabric at speed several times through the flame of Bunsen burners extending the width of the cloth. Improvements in this step must come from proper regulations of the gas mixture and mechanical arrangement of rollers and speed of travel to produce the most efficient results.

The next operation is known as steeping or grey washing and consists in passing the goods through a washer or otherwise saturating, preferably with warm water, and allowing it to remain piled in a warm room from 18 to 24 hours. This treatment removes any bodies soluble in water and, as a result of fermentation, the starchy matters added as sizing.

Modern practice has made the mistake of considering this operation unnecessary and omitting it entirely even when the caustic boil is used. Lime probably removes the starchy matters completely, but it is extremely doubtful if more than a relatively small portion is extracted in the caustic boil. Steeping may be replaced by a malt treatment, using by preference one of the diastase preparations now on the market. The solution is prepared by using 15 to 20 lbs. of diastase or similar product to 100 gallons of water and warming to 140 to 150° F. The goods are wet out in this solution and the action allowed to continue for 15 to 30 minutes, then washed and passed to the next step in the process. Solubility of the

starch may also be produced by wetting out with very dilute hydrochloric or sulphuric acid and piling for a short time. This process must be carried out with extreme care, owing to danger of the acid injuring the fiber.

Lime boil.—The cloth is saturated with milk of lime of such a strength that it takes up about 4 per cent of its weight of CaO , run into one of the stand-kiers and boiled with constant circulation of the liquor through the goods. This operation decomposes the fats and similar matter contained in the cloths and forms as a result insoluble lime soaps. These latter are broken up by the scouring which follows and completely removed by the subsequent operations. In addition, the other natural impurities of the raw cotton, as well as starch, are so changed that they either become soluble or are more easily acted upon by the chemicals employed in the succeeding processes.

The gray soaps remove the excess of lime and other metallic oxides, if present, besides breaking up the insoluble soaps formed on the fiber.

The lye boils continue the reactions already begun in the lime boil and further saponify and remove any fatty acids resulting from the decomposition of the lime soaps in the previous steps.

Resin soap is employed by some bleachers and is supposed to remove certain indefinite constituents of the fiber which are said to have a deleterious effect on the whites of prints and are not fully removed by the other agents employed.

Bleaching or Chemicing.—The destruction of the coloring matter now left in the fabric is the object of this final step. Previous to the 19th century this was always accomplished by the exposure of the moist cloth to the action of air and sunlight and extensive bleaching greens formed an important part of every establishment. The use of chlorine for this purpose was first proposed by Berthollet, about 1785, but its use even after the introduction of the hypochlorite of lime was very slow in becoming general. Today, chlorine, either as calcium or sodium hypo, far exceeds all other agents in importance and in amount used. With proper regulation of strength of solutions and duration of action, this bleaching agent may produce the most satisfactory whites without appreciable injurious action on the strength or other desirable properties of the fiber.

Large quantities of chlorine are annually produced electrolytically either as a gas to be absorbed by lime to form ordinary bleach, or as hypochlorite of soda solution to be used direct. On a large scale the latter form cannot compete successfully with the cheaper and more common product, but it finds extensive use in laundries and the small establishments. The sodium hypochlorite solution formed in this form of apparatus contains an excess of sodium chloride which causes the chlorine to be given off freely either to the fabric or the atmosphere. The

solutions are therefore unstable and cannot be made in advance. The cost of production is also higher except in the case of cheap electric power and low freight rates for salt.

The much claimed superiority of electrolytic bleach seems to have been based on the increased rate at which the chlorine was given off in the presence of the chloride and may be produced, if desired, in bleach from other sources.

The use of hypochlorite of soda made from bleaching powder and soda ash is highly to be recommended and in many cases will more than repay the increased cost. The following receipt for its manufacture is given by Thies and Herzig:

Two hundred and ninety kilos of bleaching powder of 33 per cent available chlorine are ground to a paste with 1100 liters of water, and to this are added 175 kilos of soda ash dissolved in 500 liters of water, and the resulting solution is immediately made up to 2,000 liters. After stirring for one-half hour, the mixture is allowed to settle over night and the clear liquor is decanted. The sediment is mixed with clear water, the clear liquor being decanted and this process repeated until in all 5,000 liters of liquor are obtained, which should stand at from 6 to $6\frac{1}{2}^\circ$ Twaddle. This solution is alkaline and is said to keep well.

In order to neutralize the free alkali and start the liberation of the hypochlorous liquor, sulphuric or hypochloric acid is added to the liquor just before using. Thies and Herzig recommend 6 to 7 grams of sulphuric acid of 60° Beaume for every 10 liters of the above bleaching liquor.

The kiers used in bleaching have had much to do with the progress of the art. Starting from open kettles heated by direct fire, the natural tendency led to a large open kier, but heated by steam. A great improvement in this kier was the introduction of a central vomit pipe taking the liquor from the bottom of the kier and throwing it by means of a steam ejector over the goods at the top, thus producing a circulation through the goods. The next improvement was the introduction of the closed kier, by means of which the pressure and consequent temperature of boiling was increased.

The first distinct system was that of Barlow, whose kier was introduced about 1870. This system employs two kiers working in pairs and may use steam up to 40 lbs. pressure. The liquor circulates from the bottom of one kier to the top of the other. In working with the Barlow system, both kiers are filled with cloth as usual, the requisite quantity of liquor admitted and steam blown through both to expel the air through vent pipes. The kier is then closed and steam let into one only, so that the pressure at the top forces the liquor through suitable connection to the top of the other, where it is sprayed over the cloth. When most of the liquor has been forced into the second kier, the valves are changed so that the pressure

is led to that kier, changing the circulation to the top of the first kier. This reversing process is continued until the goods are sufficiently boiled, when the kiers are blown out, washed and the process continued as usual.

The next great step in advance was the advent of the Mather kier, which was especially constructed by Messrs. Mather and Platt to carry out the process devised by Horace Noechlin. In this system the lime boil is entirely dispensed with, and this, as well as the two lye-boils of the old process, are replaced by a single boil in caustic soda and resin soap. The kier used for this boil is a large horizontal cylinder with convexed ends, one of which may be removed to allow the entrance of the goods. Suitable means are provided for raising the door and for making the same steam tight when closed. The pieces to be bleached are piled on wagons made of sheet iron with perforated bottoms and running on tracks leading inside the kiers. When ready the wagons are run into the kier, the door is closed, the air driven out by steam and circulation started. While the kier is boiling, other wagons may be loaded and those already boiled unloaded, thus increasing greatly the production of a single kier.

The process aims to subject the fabric to the action of caustic liquor and steam at the same time, and for this purpose the amount of solution used is small compared with the size of the kier. The circulation is maintained by means of a centrifugal pump, which takes the liquor from the space under the wagon and showers it over the top of the goods. It then percolates through the pieces and collects in the bottom of the kier, where it is heated by closed steam pipes and is again taken up by the pump, which maintains a continuous circulation. The kier may also be heated by the introduction of live steam, but the resulting dilutions of the liquor is more objectionable in this than in other kiers.

Several ingenious continuous processes have been proposed for boiling cotton piece goods, but as yet no system of this character has succeeded in prolonging the treatment long enough to produce results comparable with those obtained by the more usual method.

The Thies kier and the process of which it forms a part are the result of long, careful research by Thies and Herzig. By designing the kier in such a manner that no steam comes in direct contact with the liquor and thus avoiding the action due to the oxygen contained in the air always present in steam, it is impossible to use strong caustic at a high pressure without fear of tendering the fabric. The latest form of kier consists of three parts—the kier proper, the vacuum boiler and the superheater. The vacuum boiler consists of an upright cylindrical vessel similar in size and shape to an ordinary kier, while the superheater is of the same height as the kier, but much

smaller in diameter. The latter is provided with closed coils for heating. For the circulation and transfer of liquors, use is made of a Grindle pump. The boiling consists of two parts and the process is described by the authors as follows:

The goods saturated with alkali (liquor blown off from the previous bowking) are packed into the kier and then by means of the Grindle pump, the old kier liquor is drawn from the vacuum boiler (which is quite full to start with, and closed at the top) and sent in at the bottom of the kier until the latter is quite full, the air escaping through a blow-off valve at the top. As soon as the kier is full, the air valve is closed. By pumping the liquor from the vacuum boiler into the kier, a vacuum equal to about 10 pounds is produced in the former vessel. A pipe provided with a throttle valve connects the top of the kier with the top of the vacuum boiler. This valve is now slightly opened, so that a circulation of the liquor takes place, while any air contained in the kier rises with the liquor, passes through the throttle valve into the vacuum boiler, where the liquor falls to the bottom and expands into the vacuum. The pressure in the kier is now gradually increased to three atmospheres by checking the valve, when any air remaining in the goods is mechanically dissolved out of the goods (air being more soluble in water under pressure) and transferred to the vacuum boiler. After the air in the vacuum boiler has been expelled, steam is turned on in the superheater and the bowking continued for 2 hours at 45 pounds pressure. The pump is now stopped, the steam turned off and the liquor blown off under its own pressure. Fresh lye, consisting of caustic soda at 6 to 7½° Twaddle and the resin soap, is now run into the kier (which is free from air), the pump set in motion again, steam turned on in the superheater and the bowking proper continued for 6 hours at 45 pounds pressure.

As free hypochlorous acid is much more efficient as a bleaching agent than any of its salts, we find many attempts to make use of this property.

Mather and Thompson pass the cloth continuously through closed chambers in which the pieces are run through an atmosphere of CO₂ after being saturated with the ordinary bleaching solution. Acetic acid vapors have also been used in the same manner. The addition of small amounts of sulphuric or hypochloric acid or of acetic or formic acid to the chemical solution will produce similar results, and the improved action of the liquors will be at once apparent.

Chlorine in cotton bleaching may be replaced by other oxidizing agents and will no doubt be superseded at some later date by some agent acting less injuriously on the fiber itself. Such a body is found in sodium or hydrogen peroxides and it is only the higher cost which prevents its more general use. For cotton and silk goods, laces and other special fabrics, it finds extensive application.

Drawing freely from all published information, especially the results of the researches of Koechlin, Thies and Scheurer, the following process has been evolved as a working compromise of the theoretical and practical.

The goods are marked and sewed as usual and piled on trucks. They are then run through the gas singer and directly through a lukewarm solution of diastafaer and into a continuous piling chute. This latter is a more or less complicated inclined trough so arranged that while cloth is entering at the upper end and leaving at the lower, a large amount of slake cloth fills the length of the incline and allows time for the action to take place. To increase the time of action, several of the chutes may be arranged in series, the cloth passing over a reel between each. From this step the goods pass directly to a small washer and are then squeezed and piled in a bin. As wanted they are drawn through a saturating machine, where they are impregnated with caustic soda at 3° Twaddle, containing 0.5 per cent sodium bisulphite solutions and packed directly in the boiling kier. This kier should be of the vertical pressure type with outside vomit pipes and pump circulation. The perforated false bottom should be conical in shape and so arranged as to give a much larger space at the bottom than is usually allowed, and should be provided with both closed and open steam pipes. The closed coil should be sufficient to maintain the kier at boiling during circulation after that temperature has been reached, and must be provided with a suitable trap to remove the hot water and return it to the hot well or feed water heater. The goods are evenly plaited and tramped in the kiers by boys in the usual manner, to within 2½ or 3 ft. of the top. The air is next replaced by admitting steam and a charge composed of 3° Twaddle caustic containing 3 per cent of resin soap run in to cover the goods. The kier is brought to a boil with the open coil at a pressure of 15 lbs., a small vent pipe being open to allow the escape of air.

When the goods are boiling the open pipe is nearly closed and the boiling is continued for 6 to 8 hours with the use of the closed coil as far as possible.

The charge is washed in the kier, care being taken to prevent access of air to the hot fabric. The pieces are then run out of the kier through a washer, squeezed and returned to another kier, where the boil is repeated, but with the omission of the resin soap.

Experience seems to indicate that two boils are necessary to produce a thorough bleaching without over treatment in the subsequent steps.

The goods are next run through the chemic and piled in open bins for 2 to 6 hours. The best results are produced at the greatest economy by the use of hypochlorite of soda prepared as already described, but with experience and care very satisfactory results are obtained, by the use of hypochlorite of lime solution at ½ to 3° Twaddle.

Continuous processes which subject the fabric for

a short period to a very strong chlorine solution or which sour the cloth after chemicing without previous washing are necessarily injurious to the fiber and unsatisfactory in other important respects.

The pieces after lying exposed to the air for a suitable length of time are washed, squeezed through sulphuric at 1 to 2° Twaddle, washed twice, and are then ready for opening, drying and finishing as desired.

Bleaching today is not an exact science, and depends for success upon experience combined with close observation and understanding of the various processes. If the heads of the industry in this country could be brought together for an interchange of experience and frank discussion of the most promising lines of research, a great advance would be produced in the art, but in any case we may expect sooner or later more economical and efficient processes to replace many of those now in use.

A. U. S. Consular Report states that the old fashioned "beehive" coke ovens, to be seen on nearly every hillside by the traveler through the Sheffield district of England at night, are being rapidly superseded by the modern retort oven from which valuable by-products—i. e., tar, benzol, and ammonia—are recovered. The remarkable growth of this industry in the last 10 years as seen in Yorkshire, Derbyshire, and the Northumberland districts was the subject of a lecture before the department of applied science of the Sheffield University recently, wherein it was stated that the proportion of coal carbonized or coked in by-product ovens in Germany was 30 per cent in 1900 and 82 per cent in 1909; in England, 10 per cent in 1900 and 18 per cent in 1909; in the United States, 5 per cent in 1900 and 16 per cent in 1909. This activity stimulated designers to improvement in oven construction and led to simplification and improvement of by-product recovery. Dealing with the treatment of waste liquors the speaker said that experiments which he had made, but which were not sufficiently advanced for publication, led him to believe that the phenols could be completely recovered by oxidation with Weldon mud, but he had not yet succeeded in entirely removing the sulphocyanides. A most important advance in by-product recovery was to be found in the direct process for the recovery of ammonium sulphate, the successful introduction of which would partially if not completely solve the question of effluent. These processes had for their object the recovery of ammonium sulphate without subjecting the gases to previous water scrubbing.

The production of copper in Chile during the year 1910 amounted to 17,987 tons, of which 9,271 tons were shipped in the form of copper bars and the balance as regulus and ores. The largest amount of the copper was shipped to England.



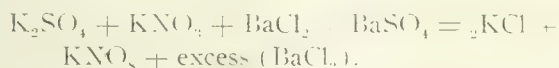
TITRATION METHOD FOR DETERMINING SULPHUR IN COAL.

By E. A. CUNNINGHAM.*

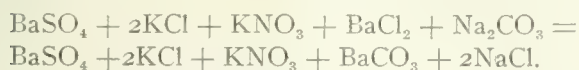
This method was devised to facilitate the determination of sulphur in coal from the washings of the Atwater Mahler bomb calorimeter after the determination of the B. T. U. has been made.

The method is based on the following:

"If we take the bomb washings which have been titrated in the usual manner with an alkali to determine the calories produced by the combustion of sulphur and nitrogen and if we assume this alkali to be potassium hydrate, the solution will contain the sulphate and nitrate of potassium. If we then add a known quantity of BaCl_2 and if this known quantity is in excess of that required by the highest percentage of sulphur found in any coal it is evident that the following will take place in this solution:



If we now add to this solution a small quantity of Na_2CO_3 we shall precipitate the excess of BaCl_2 as BaCO_3 , and the solution will contain the following:



This solution is to be filtered which separates the $\text{BaSO}_4 + \text{BaCO}_3$ from the other soluble constituents; the filter and contents are then placed in any convenient vessel and titrated with HCl . A blank has also been run on the known quantity of BaCl_2 taken and the number of cc. used for the sample subtracted from the cc. used for the blank, gives the sulphur. The solution having been previously standardized with a Na_2SO_4 solution, the sulphur of which has been carefully determined by gravimetric methods.

The following gives the detailed procedure:

To the bomb washings which have been titrated as usual to determine the calories is added 9 cc. of an $\text{N}/2$ solution of BaCl_2 , this amount being measured accurately from a burette. It will be noted that 9 cc. of an $\text{N}/2$ BaCl_2 solution, would represent 0.072 grams of sulphur and as the quantity of coal taken in the determination of the B. T. U. is about one gram, this

would represent 7.2 per cent of sulphur, an amount far in excess of that usually found in coal. After the addition of the above quantity of BaCl_2 , a small quantity, about 5 or 6 cc., of a normal solution of Na_2CO_3 is added. This need not be measured accurately.

After this addition, the solution is decanted through a filter paper and the precipitate washed twice with hot water, after which the filter and its contents are placed in an Erlenmeyer flask and about 15 cc. of water added. The flask is then corked with a good fitting rubber stopper and the contents shaken violently in order to break up the filter paper, after which the rubber stopper is removed and washed off into the flask, the sides of which are also washed down. One or two drops of methyl orange are then added and the contents titrated with $\text{N}/16$ HCl and this figure subtracted from the figure found in running the blank, gives the sulphur in 0.001 gram per cc., provided the HCl solution is accurately $1/16$ normal.

The blank determination need only be run once when the solutions are first made up, as they are quite stable and this figure found used in all the subsequent titrations of the samples.

The solutions should of course be standardized by using a solution of Na_2SO_4 , the sulphur content of which has been carefully determined by gravimetric methods.

The bomb washings need not be filtered before adding the solutions as the solid particles have no effect in the subsequent operations.

It will also be found that the presence of BaCO_3 with the BaSO_4 entirely prevents the tendency of the BaSO_4 from running through the filter paper.

The time for the complete operation is about ten minutes, which is very much faster than the gravimetric method.

The accuracy of the method is shown by the table given below:

By Titration, Per cent.	Gravimetrically, Per cent.
0.830	0.830
0.660	0.660
1.090	1.070
0.860	0.810
3.540	3.600
1.296	1.289
0.600	0.556
0.590	0.585
0.752	0.780
1.072	1.069
3.856	3.789

*Chemist American Bridge Co., Pencoyd.

COMPARISON OF METHODS FOR THE DETERMINATION OF NICKEL IN STEEL.

By JAMES J. BOYLE.*

Having noticed that many discrepancies occur in determining nickel in steel by the old ether separation method, and the subsequent precipitation of the nickel with hydrogen sulphide, converting the nickel sulphide formed to nickel oxide, and weighing as such, an investigation of some recent methods was made, and their accuracy noted.

DIMETHYLGLYOXIME METHOD.

O. Brunck in Stahl & Eisen, states that dimethylglyoxime, an organic compound, has proved a very efficient reagent for the determination of nickel in various substances, and with certain modifications, can be used to determine nickel in iron and steel.

After conducting a number of experiments for the determination of nickel in steel, with this reagent, the following procedure was found to produce the best results:

Weigh out one gram of the nickel steel, place in a 250 cc. beaker, cover, add 20 cc. of hydrochloric acid (sp. gr. 1.12), heat gently till steel is dissolved, keeping beaker covered; oxidize with 6 to 7 cc. of nitric acid (sp. gr. 1.20). Continue heating a few minutes longer, to complete the oxidization, add twenty grams of tartaric acid dissolved in 20 cc. of water, remove from fire and cautiously add a strong solution of potassium hydroxide till solution is nearly neutral, a change of color indicating this condition. Volume of solution should now be about 200 cc.

If solution is made alkaline, slightly acidify with acetic acid, heat to boiling and add 15 to 20 cc. of a 1 per cent alcoholic solution of dimethylglyoxime with stirring.

Ammonia is then cautiously added until the solution smells *slightly* of it; while still hot, filter on a weighed Gooch crucible, wash thoroughly with warm water, dry in oven at 110-120° C. for 45 minutes, cool in desiccator, and weigh.

Nickel glyoxime is a red crystalline salt, contains no water of crystallization, sublimates at 250° C., without decomposition, and contains 20.32 per cent nickel.

To test the accuracy of the method .2 gm. of cp. nickel ammonium sulphate, and 1 gram of common steel were weighed out and put through the above procedure, and the following results obtained, check analysis being run:

	Nickel Present, Per cent.	Nickel Found, Per cent.	Error, Per cent.
Test No. 1.....	2.972	2.970	-.002
Test No. 2.....	2.972	2.981	+.009
Average	2.972	2.975	+.003

In both cases the Gooch crucible containing the nickel glyoxime was heated until constant weight was obtained.

The results show this to be a very accurate method. It is easy to operate, and can be readily completed in 1½ hours. The only objection to the method is the cost of the reagent, which is at present, 10 cts. per gram.

A reduction of this price is in view. The oxime is readily recovered by mixing the nickel salt with water to a paste, warming with potassium cyanide, filtering hot and precipitating the oxime at once with acetic acid.

CYANIDE METHOD.

This is the method most used now in steel laboratories, having superseded the old sulphide method.

After reviewing the work of Chas. M. Johnson of Pittsburg and Geo. S. Jamison of Yale on this subject, the following scheme, which is essentially a combination of both writers' methods, was found to give good results:

Weigh out 1 gram of the nickel steel in a 150 cc. beaker, dissolve in 20 cc. of (1:1) hydrochloric acid. When action ceases, add 10 cc. of nitric acid (sp. gr. 1.20); reduce to volume of 15 cc. by boiling, keeping beaker covered, remove from fire, add 32 cc. of 25 per cent sulphuric acid in water. Transfer to a 400 cc. beaker, add 4 grams of anhydrous sodium pyrophosphate, and 5 grams of citric acid, stir well and then slowly add ammonia, till the precipitate just dissolves and the solution acquires a *faint* odor of ammonia.

If too much ammonia is added nearly neutralize by the careful addition of nitric acid; dilute to about 150 cc., cool to below 20° C., add 2 cc. of a 10 per cent solution of potassium iodide, and enough tenth-normal silver nitrate solution, the volume of which must be noted, to produce a distinct turbidity. Slowly run in the standardized potassium cyanide, stirring thoroughly till the turbidity *just* disappears, and the solution assumes a light golden yellow color.

The solution should remain bright for five minutes, otherwise the titration is incomplete.

The end point is best observed when the beaker is placed upon a white paper having an elliptical hole in it under which is placed a black glazed paper for contrast.

Make the proper deduction for silver nitrate in calculating results.

STANDARD SOLUTION.

Prepare an exact tenth-normal silver nitrate solution, 1 cc. of which equals .002935 gram of nickel.

Standardize the potassium cyanide against this solution by taking 20 cc. of cyanide in a 250 cc. beaker, diluting to 150 cc. with cold water, adding 2 cc. of a 10 per cent potassium iodide solution, running in silver nitrate until a distinct turbidity is produced, and then slowly adding the cyanide till the turbidity just disappears. From the amount of silver nitrate used, the strength of the cyanide in terms of nickel is readily found.

The potassium cyanide should be made up approxi-

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mately tenth-normal and 5 grams of potassium hydroxide added per liter in order to improve its stability, frequent standardizations being necessary.

It is perhaps better to standardize the potassium cyanide against a steel of known nickel content, but the theoretical value used in the tests given below indicate very satisfactory results:

Two-tenths gram of cp. nickel ammonium sulphate and 1 gm. of common steel were weighed out and put through the above procedure, and the following results obtained for six tests:

	Nickel, Present, Per cent.	Nickel, Found, Per cent.	Error, Per cent.
Test No. 1.....	2.97	3.00	.03 +
Test No. 2.....	2.97	2.99	.02 +
Test No. 3.....	2.97	2.99	.02 +
Test No. 4.....	2.97	3.00	.03
Test No. 5.....	2.97	2.98	.01 +
Test No. 6.....	2.97	2.99	.02 +
Average	2.97	2.99	.02 +

Check analyses by this method can be completed in fifty minutes.

RESUME.

The dimethylglyoxime is more accurate than the cyanide method, and should be employed in all umpire and check work where exact determinations are desired.

The cyanide is a good, quick, accurate method, particularly adapted to routine laboratory work. It is easy to conduct, though the operator should have some experience with the end point, the greatest source of error in the method.

Plans have been drawn for the Hawaiian Sugar Planters' Association for one of the finest laboratories in Hawaii Territory. The laboratory will cost \$18,000 and is to be used for the delicate and important work carried on at the present experiment station.

The Department of Mines of New Zealand aids prospectors by leasing to them state-owned prospecting drills. The department has three diamond drills and one Keystone driller, which may be leased on the following conditions: The lessee must take delivery of the plant at any place decided upon by the minister of mines. In addition to making a deposit of £50, the lessee must furnish bond and security for the payment of all charges and the proper observance of all conditions. All damages to the plant outside of reasonable wear and tear must be paid by the lessee. A superintendent appointed by the minister of mines has sole control of the plant and the workmen employed, and the lessee must comply with his requests regarding the housing and care of the plant. The salary of the superintendent is paid by the department in the first instance, but is recovered from the lessee in monthly installments. The lessee must pay the cost of all carbons used in boring and all working expenses of the plant.

EXAMINATION OF TIN IN AN ORE.*

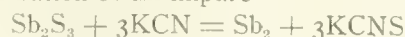
By R. J. MORGAN.

It is a curious fact that for the last 25 years tin-assay methods have made little or no advance, while those now in general use in connection with copper and lead ores can scarcely be compared with those then obtaining, so greatly have they been improved upon. Though it is not altogether expected that the method herein advocated will at once meet with general approval, the writer hopes the resultant discussion may bring out points that have hitherto escaped detection.

The old Cornish method of smelting with anthracite is now obsolete. The cyanide assay, although unreliable, is still much used because of its simplicity.

In the Malaya many buyers of tin ores use the cyanide assay, the smelting being carried out generally in a benzene furnace.

It is essential that the smelting with cyanide be performed at a low temperature. Even then, impurities, such as Sb_2S_3 and Bi_2S_3 , if present, are reduced, rendering the button of tin impure—



Oxides of iron are readily reduced, the iron going in with the tin, and if the button contains impurities the percentage of tin in the button must be determined by wet methods; therefore it is desirable to use a wet method, and so avoid the doubtful fusion connected with the cyanide process.

In technical practice, volumetric methods are preferable, being quite as accurate and much more rapid than gravimetric methods. Consequently this paper will be confined to discussing volumetric methods. Many schemes are in use, but professional men appear to differ as to which is the most practical and satisfactory volumetric method. The two in general use at the present time are:

1. Titration of stannous chloride with standard iodine solution.
2. Titration with a standard solution of ferric chloride.

Both these titrations give satisfactory results, but the writer prefers No. 1, as the operations of preparing the assay solution are simpler.

The following method, which is a modification of that commonly known as the Pearce-Low scheme, was found to give very satisfactory results. It is rapid, and answers every ordinary requirement:

Place from 6 to 8 gr. of stick sodium hydroxide in a thin-spun iron crucible provided with an iron cover, and gently heat to expel moisture. Weigh out about 5 grm. of the finely powdered ore and add to the iron crucible after the smelted hydroxide has cooled sufficiently to permit of its setting. Cover the crucible, and again heat, gently at first, and afterwards to dull redness, until fusion is complete. Remove the cover

*Paper read at the New Zealand Meeting of the Australian Institute of Mining Engineers.

and allow the "melt" to cool, placing the cover in a 5¼-in. casserole, or in an ordinary porcelain evaporating basin. When the crucible has cooled, place it on its side in the casserole, add a little water, and heat to boiling point. Next, loosen the "melt" with a glass rod, wash the crucible well until all the "melt" has been removed, and then lift the crucible from the casserole, giving it a final rinse: Rinse the crucible with a few cubic centimetres of weak hydrochloric acid to insure that absolutely all the "melt" has been removed, adding the acid to the solution in the casserole, which must be covered to prevent loss by effervescence. Remove the crucible cover from the casserole, and if any globules of the "melt" are still attached, remove them with a dilute acid wash.

It is important to keep the washings in the above operations as small in volume as possible.

Next add strong HCl, a few cubic centimetres at a time, keeping the casserole well covered until effervescence ceases. Then add 20 cc. of strong HCl—in excess, heat to boiling point and transfer the solution to a suitable flask. No undecomposed residue should remain in the casserole, except perhaps a little gelatinous silica. Add three horseshoe nails to the solution, the nails being bent to form the letter U. Boil for 15 minutes, and filter through glass wool, washing the filter with about 50 cc. of distilled water.

Horseshoe nails, being made of a very pure and soft iron, act efficiently in removing Cu, Sb, As, etc., and at the same time minimize the risk of introducing other impurities.

To the filtrate, which has been caught in a 6-oz. conical flask, about 5 gm. of finely powdered antimony is added. The solution is then heated to boiling point, and the boiling is continued for five minutes. The flask is next cooled in an atmosphere of CO₂ and titrated with a standard solution of iodine, using a starch solution as an indicator. It is essential that the solution, after boiling with antimony, should be cooled as rapidly as possible, and to accomplish this the flasks can be placed in a tray and cold water circulated round them. As the stannous chloride oxidizes very rapidly to stannic chloride, especially when the solutions are hot, the solution must be cooled in an atmosphere of CO₂. This may be done by placing a pea of marble in the flask. This last instruction cannot, however, be recommended, as the results are generally low—probably owing to the introduction of air.

When doing a number of determinations it is better to cork the flasks during the operation of boiling, the corks being fitted with a delivery tube. On removing the flasks from the hot plate, connect the delivery tubes to a constant supply of CO₂.

Mr. J. J. Berringer, in an article written to the *London Mining Journal*, advocates the use of zinc fume or clippings for reducing the tin compounds, in place of fusion with sodium hydroxide.

The writer has used this method and found it to be an efficient one, giving results which agreed well with

the method already described under the heading of "Method Advocated."

The hydrogen reduction method championed by Parry, and claimed by him to be the method, is, in the writer's opinion, cumbersome, and decidedly inferior to the above methods for accuracy, simplicity and speed.

Many methods are in use for reducing the stannic solutions to the stannous state ready for titration.

Iron horseshoe nails are effective if the solution is kept strongly acid and boiled for at least 25 minutes.

Sheet nickel may be used in place of iron, but it is expensive, and the strong color imparted to the solution tends to mask the appearance of the first tinge of starch iodide, and this is objectionable when testing for small amounts of tin.

Zinc reduces tin chloride to the metallic state, and it can then be filtered off and redissolved in strong HCl. Experiments showed that whilst being redissolved there was a tendency for the tin to oxidize to stannic chloride, and in this one is obliged to assume that the tin dissolves wholly as stannous chloride. As this assumption is not justified, the method is quite unreliable. The writer considers the best reducing material to be finely powdered antimony, which is effective and rapid, and the excess of Sb that remains does not interfere with the iodine titration, which is performed in the cold. It is essential that the antimony should be extremely fine and free from interfering metals. Kahlbaum's pure pulverulent antimony answers well, the rapidity and completeness of reduction well repaying the cost of the pure metal.

In the ferric chloride method of titration the strong yellow color of ferric chloride indicates that the conversion of the stannous chloride to the stannic condition is complete. With this method antimony cannot be used as a reducer, metallic zinc being generally employed, and, as pointed out above, this is open to error. The titration must be performed above 70° C., and at such a high temperature it is difficult to prevent oxidation.

After a considerable number of experiments, the writer is of opinion that fusion with NaOH in an iron crucible, followed by reduction with iron nails and antimony and the iodine titrations, gives the most accurate and consistent results with the minimum of trouble.

Reduced output of balata gum is reported from British Guiana. This year's shipments up to May 25 totaled 104,000 lbs., against 148,000 lbs. for the same period last year. To this shortage should be added 46,000 lbs. shipped this year, but which were bled in 1910. Lack of laborers is given as the chief cause. The United States imported 535,195 lbs. of balata, worth \$340,569, in 1910, against 803,569 lbs., worth \$385,625, in 1909. British Guiana furnishes the largest supply, the balance coming mainly from Venezuela and the West Indies.

SOME OBSERVATIONS ON LABORATORY PRODUCTION OF ALUMINUM*

By H. K. RICHARDSON.

It has been the writer's privilege to make a number of runs on the production of aluminium in the laboratory. These have proved so interesting that it has been decided to publish the results. All the work was done by students during the regularly scheduled laboratory periods. These students were chemists and metallurgical, chemical and electrochemical engineers. A laboratory process to be successful in the hands of these men must be simple, for some had had little or no experience in handling electrical apparatus for real work.

From the start it has been our desire to approximate as closely as possible the conditions of practice. In this work the student is expected to consider himself as working a commercial process. In his final report he is required to hand in a detailed estimate of the cost of running the process, using the laboratory data. Our earliest attempts to make aluminium were

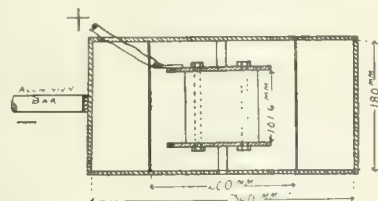


Fig. 1.

made in an apparatus similar to that of Tucker,¹ but results were not satisfactory, for two reasons:

1. The yield was seldom over 30-40 grams for a three or four hour run. This corresponds to an ampere-hour efficiency of 20-25 per cent.
2. The apparatus and method of working are not comparable to practice.

DESCRIPTION OF FURNACE USED.

To more nearly approximate practice, a new cell was constructed according to the figures of a commercial cell given by Winteler.²

This cell, for 3,200 amperes, had the following inside measurements:

1,050 mm. (3 ft. 5 $\frac{3}{8}$ ins.) long.

550 mm. (1 ft. 9 $\frac{5}{8}$ ins.) wide.

6,500 amp. per sq. meter (4.2 amp. per sq. in.).

As we had only 500 amperes available, we designed a cell of the following inside dimensions:

360 mm. (14.18 ins.) long.

180 mm. (7.09 ins.) wide.

110 mm. (4.33 ins.) high.

The material used was $\frac{1}{4}$ -in. (6.35 mm.) wrought iron plate, bolted to angle iron on all edges. The bottom was covered with an artificial graphite plate $\frac{7}{8}$ in. (2.22 cm.) thick. A plan of this furnace is shown in

Fig. 1, while vertical sections are shown by Figs. 2 and 3. It was our intention to use for electrodes three carbons 18 ins. (45.7 cm.) long by 3 ins. (7.62 cm.) diameter, thereby duplicating American practice. Before our cell was set up, the article by Neuman & Olsen³ appeared, and it was decided to use two square electrodes 4x4 ins. (10.16 x 10.16 cm.). However, we found two electrodes inconvenient to work with, so we finally made our runs with one electrode of above

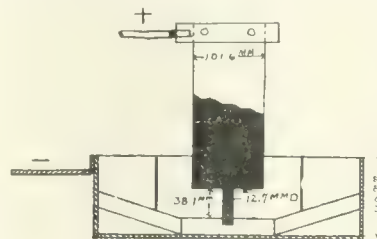


Fig. 2.

dimensions and 9 ins. (22.9 cm.) long. This electrode was of common electrode carbon stock obtained from the National Carbon Co. The method of fusing the electrolyte used by Neuman & Olsen was employed. The heat was obtained by inserting a 2-in. (5.08 cm.) piece of $\frac{1}{2}$ -in. (1.27 cm.) carbon between the electrode and the bottom plate. See Fig. 2. This left 1 $\frac{1}{2}$ ins. (3.81 cm.) of the carbon exposed to do the melting. This length of carbon corresponds to 0.0125 ohms resistance. At the same time the furnace was shortened by placing asbestos board partition on each side 2 ins. (5.08 cm. from the electrode. The remainder of the space was filled with sand.

Fig. 2 illustrates this furnace set up ready to melt.

Fig. 3 illustrates this furnace set up with heating carbon removed and electrolysis taking place.

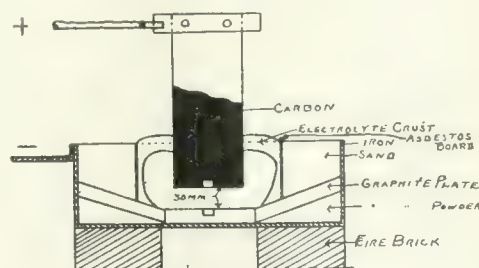


Fig. 3.

INSTRUCTIONS ISSUED FOR THE RUNS.

These instructions are the combined results of our experience and those of the various experimenters cited by Neuman & Olsen.³

The instructions are as follows:

1. Make 4 kg. of a mixture:

	Runs 1 and 2, Per cent.	Runs 4 and 5, Per cent.	Run 3 old charge
Alumina	20	15	
Natural cryolite	80	85	

2. Melt up 3 kg. of charge at once. Add the remainder as the bath becomes diminished.

*Paper presented at the 19th General Meeting of the American Electrochemical Society in New York City, April 6-8, 1911.

1. Electrochemical & Metallurgical Industry, 7, 315.

2. Borchers-Solomon; "Electric Furnaces," p. 210.

3. Remove the heating carbon as soon as the bath is thoroughly fluid.
4. Add alumina to bath every 30 minutes. Quantity to be calculated on assumption of 50 per cent current efficiency.
5. Use anodic current density of 3 amperes per square centimeter.

8. Keep crust broken on at least one side of electrode. Crushed coke around the electrode will tend to keep crust soft.
9. Take voltage and current readings every five minutes.

RESULTS OF RUNS.

Fig. 4 shows the current readings in each of the 5

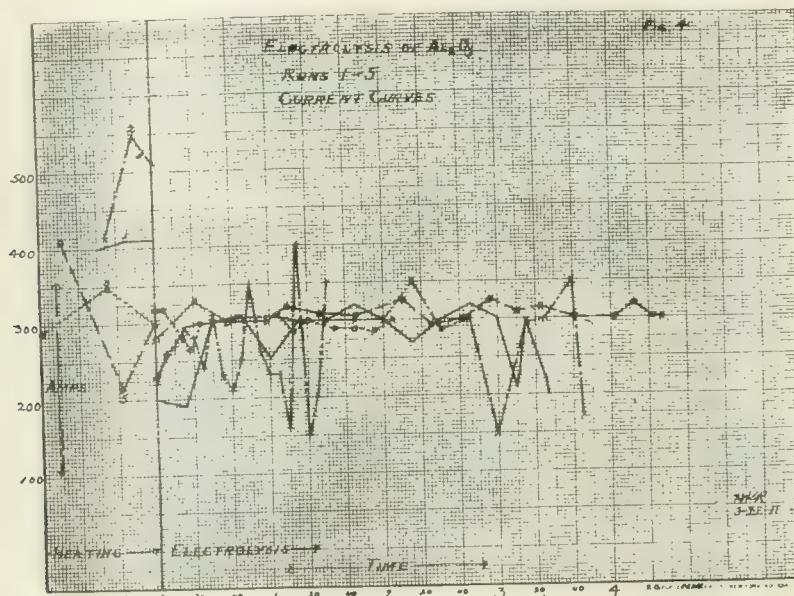


Fig. 4.

6. If anode effect causes trouble, i. e., current suddenly drops

- (1) Open and close circuit quickly
- (2) Add a charge of alumina
- (3) Tap electrode sharply

runs, 15-minute readings only being plotted.

Fig. 5 shows the voltage readings of each of the 5 runs, 15-minute readings only being plotted.

Fig. 6 gives the complete history of run No. 5. The foregoing instructions were more closely followed

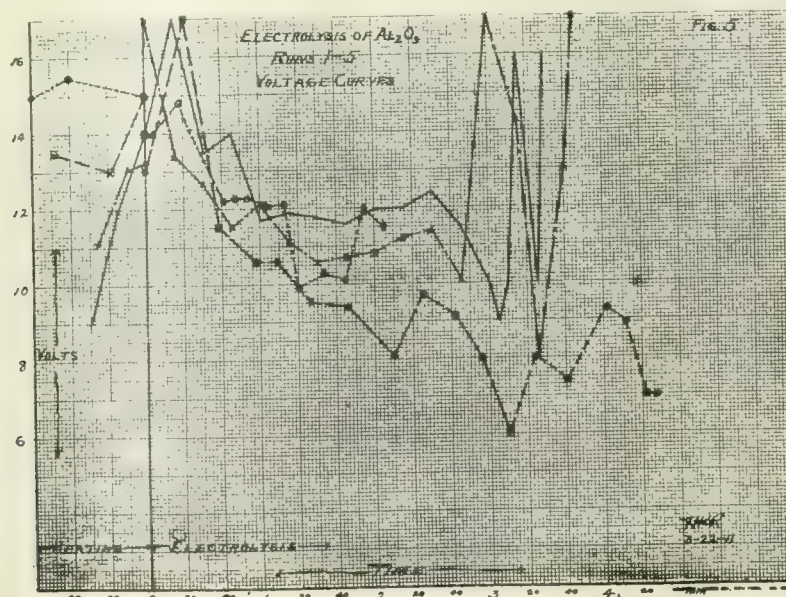


Fig. 5.

- (4) If none above effective, try adding cryolite.

7. If voltage gradually rises at constant current, add, in order named, alumina, mixture, or cryolite, whichever brings the voltage down.

in this run than in any of the others, and it thus can be taken as a sample of the possibilities of the method.

The results of all five runs are given in Table I.

Same apparatus used in every case.

Area of carbon anode = 103 sq. cm.

TABLE I.

Run.	Time of Electrolysis.	Average Volts.	Average Amps.	Average Hours.	Yield Grams (a)	Efficiency (b) Current, %	Efficiency (b) Energy, %	Anode Current Density - A/sq. cm.	Anode Current Density - A/sq. in.
1	3h. 27m.	12.5	269.4	992.4	168	50.3	11.2	2.62	16.8
2	3h. 45m.	12.5	281.6	1,086.0	277	75.8	17.0	2.76	17.8
3	3h. 5m.	12.3	295.2	615.3	96	46.3	10.6	2.86	18.4
4	1h. 30m.	9.8	257.4	383.8	120	93.0	26.6	2.49	16.1
5	4h. 27m.	9.5	307.1	1,346.0	331	73.6	21.7	2.98	19.2

Mixture:	Runs 1 and 2. Per cent.	Runs 4 and 5. Per cent.	Run 3 old charge	TABULATED RESULTS OF VARIOUS WRITERS ON THE SUBJECT.
Alumina	20	15		In this connection it is of interest to tabulate the results of previous experimenters. (See Table II.)
Cryolite	80	85		Data given other than writers' is taken from the article of Neuman & Olsen.

(a) In most cases the aluminium was found in one

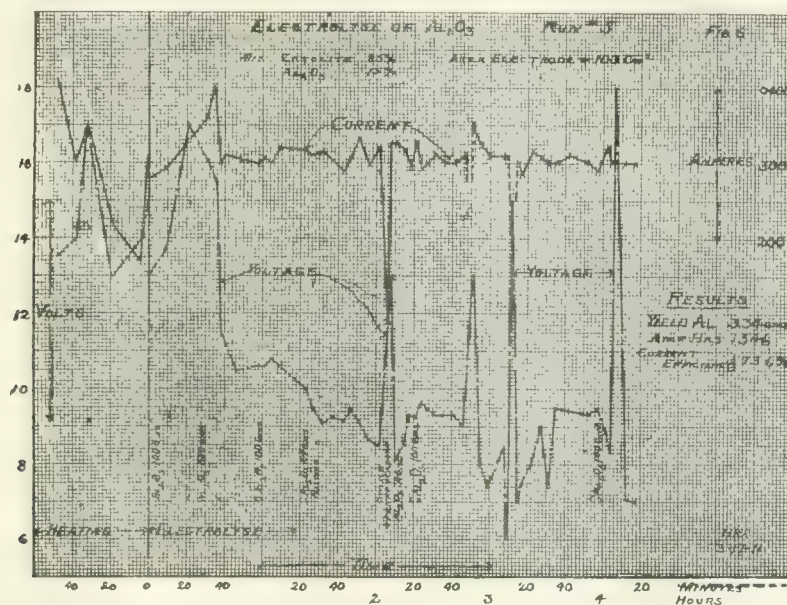


Fig. 6.

or two large pieces; in two runs tappings were made. However, in all runs small pieces were lost in the mix.

(b) Assume theoretical voltage 2.8.

No temperatures were taken, as a thermo-couple was not available at the time. The mix, however, was a bright cherry-red, temperature 900°-1000° C.

Only the range of voltage and current is given, no attempt being made to keep the different experiments of each writer separated.

OBSERVATIONS.

I. A satisfactory laboratory run can be made upon aluminium, following instructions as given. The best

Experimenters.	Type of Apparatus		Method of Heating	TABLE II. — AlO per cent.	Charge, Cryolite per cent.	Total kgms.	Volts.	Amps.	Average C. D. A/sq. cm.	Efficiency per cent.
Haber and Geipert	Crucible	anode		16.5	83.5	1.2	8	300	10.0	54.3
	Carbon	Round	Arc				7	400		14.8
Thompson	Carbon	Round		20.0	80.0		11.8	180		43.6
Thompson	Graphite	Special	Arc				13.5	390		38.0
	Box	Divided					8.8	500	10.0	49
			Arc				10.7	1,010	(Approx.)	
Richardson	Carbon	Round	Arc				7.0	148		22.6
	Carbon	Round		15	85		17.0	150		
Tucker	Carbon	Round	Arc				10			26.0
	Carbon	Carbon		10	90	12.0	12	225		(Approx.)
							8.0	150	1.17	31.8
Neuman & Olsen	Iron Box	Square	Resistance				13.6	280	1.48	45.5
	Carbon	Carbon							1.70	49.8
	Lined	Carbon							2.15	49.6
									2.68	60.8
									4.15	53.0
Richardson & Students	Iron Box	Square	Resistance	20	80	4.0	8.1	150	2.62	50.3
	Carbon	Carbon					17.0	320	2.76	75.8
	Lined	Carbon		15	85	4.0	5.8	160	2.49	93.0
							17.0	400	2.98	73.6

results are to be expected under the following conditions:

Mix: 85 per cent natural cryolite; 15 per cent alumina.

Current Density: Approximately 3 amperes per sq. cm. of anode surface.

2. The anode effect appears to consist of two kinds:

(a) Gaseous envelope which opens the circuit. This can be remedied by opening and closing the circuit or tapping the electrode.

(b) Non-wetting of the electrode, thereby causing numerous small arcs to form from electrode to electrolyte. This causes the voltage necessary to be increased, and appears to be remedied by addition of alumina or cryolite, although it is present in some degree during the entire run.

3. In the hands of students, using the apparatus as above designed, and following the instructions as closely as students usually do, we should expect to obtain an ampere efficiency of about 60 to 70 per cent.

Fruit waste, particularly apple and pear peelings and cores, is imported into Germany on a considerable scale, notwithstanding that the duty of 4 marks per 100 kilos (95 cts. per 220 lbs.) is the same as the duty on sound dried fruit. The total importations amounted to 3,117 tons in 1910, as against 2,219 tons in 1909, of which the United States furnished 2,979 tons in 1910, as against 2,201 in 1909. The principal and probably the only users of evaporated peelings and cores are the syndicated jelly manufacturers, who buy their raw materials through their own managing director. These manufacturers are able to use not only apple and pear waste, but also apricot and plum waste, all of which must be properly dried, the apple and pear waste being shipped in casks and sacks and the other waste in pressed bales and sacks, package to weigh at least 110 lbs. gross.

The chief mineralogist of the Canadian Geological Survey for the past few months has been studying the olivine-bearing rocks of Canada, which often contain asbestos, chromite, platinum, and, in some cases, diamonds. Some of his examinations were into chromite ore from the Montreal pit, near Black Lake. Until last year this mine was worked for chromite and he has established beyond all doubt that this ore contains an appreciable proportion of diamonds in small crystals. The discovery may prove of economic value. The diamonds are too small to be of use as gems, their size being microscopic, but there are so many of them that it is a question whether or not it would pay to separate them as a by-product in the concentration of chromite and sell them as diamond dust, which is much used in the cutting and polishing of diamonds and other gems.

ECONOMY TESTS OF A TURBINE-DRIVEN FURNACE BLOWER.

In the power plant of the Boston Woven Hose & Rubber Co., Cambridgeport, Mass., steam turbine driven blowers, made by the Exeter Machine Works, Exeter, N. H., have been installed, with a reduction in coal consumption of about 11 per cent, states the *Iron Age*. Before the installation of the blowers 378 tons of coal were burned per week, costing \$1,512. After the blower installation the coal consumption was 336 tons, costing \$1,344, which is equivalent to a saving of \$168 per week, or \$8,736 per year. Exhaust gases from the furnace contained 14 per cent of carbon dioxide.

In an evaporation test on one of the boilers fitted with one of these blowers, reported by J. C. Long, chief engineer of the Boston Woven Hose & Rubber Co., the coal used was a mixture of half anthracite wharf screenings and half New River coal. The water evaporated per pound of coal as fired was 8.64 lb. The equivalent evaporation from and at 212° F. was 10.35 lb. The cost of the fuel used in evaporating 1000 lb. of water from and at 212° F. was 14.6 cts. The reduction of coal cost due to the turbine blowers is \$27.60 per day. The boiler tested was a 72 in. x 18 ft. horizontal return tubular boiler, with 1,614 sq. ft. of heating surface and 36 sq. ft. of grate surface. The forced draft pressure was 1½ in. The blower on this boiler has been operated at as low as 30 lb. pressure, and at this pressure the blower made 6.05 revolutions per minute.

Advantages of using the Exeter turbine blower are the saving in coal, the reduction of the amount of clinkers and smoke, the increase in the capacity and the ability to burn cheaper fuel. The blower is small and space can easily be found for it. It is recommended that it be placed against the side of the boiler and connected with a small steam pipe for the supply and arranged to discharge under the grate. A damper may be placed on the discharge to regulate the supply and deflect it to the ash pit floor so that the distribution is uniform. With or without such a damper a pressure regulator can be applied on the steam supply which will take care of the turbine operation automatically so that it requires no attention, or a damper regulator can be used and a connection made to a lever valve in the supply pipe. When the turbine cannot be placed in the position suggested it may be located in the rear or on top of the boiler or on a bracket on the wall or anywhere that is handiest for piping the air to deliver under the fire.

Leather waste treated with tar has been used for an English road. It is stated that after nearly a year's traffic there were no signs of wear.

METALLURGY.

METALLOGRAPHY AND ITS INDUSTRIAL IMPORTANCE.*

By ALBERT SAUVEUR.†

This paper deals with a subject which was not taught at the Massachusetts Institute of Technology during my stay at that institution, and which was not even mentioned in the class room. Indeed, metallography at that time had just been born and had not attracted greater attention than is generally bestowed on infants outside the family circle. If I have contributed a mite to the growth of the child—to the progress of metallography—why, then, should my Alma Mater be entitled to any credit? Why should I not claim for myself and retain undivided whatever merit be attached to the contribution of my mite? The reply is obvious. While I did not acquire, even the rudiments of metallography, in passing through the institute, I entered it with an untrained mind and left it with a mind satisfactorily trained for scientific pursuit. This is my greatest debt of gratitude to my Alma Mater. It is why the institute is entitled to the lion's share of the aforesaid credit. This incident is only a passing illustration of what, to me, is becoming more and more evident, as the years go by, namely, that it is not the knowledge acquired at the school or university which is later of vital importance to us, but rather the training of the mind which the acquirement of that knowledge implies. It is this training—this peculiar attitude of the mind—which distinguishes an educated from an uneducated man. But such mental attitude is to be secured only through the efforts involved in the acquirement of knowledge. Have not most of us forgotten a large portion—shall I say three-fourths—of the knowledge obtained at college, ten years after graduation? Do we cease then to be educated men? Are we less educated? Clearly not, for we retain, and possibly have developed further, that mental attitude—that way of looking at things—which distinguishes the educated men. If this be so, does it matter then what we do study while at the University or Technical School. While I am not prepared to say that this is immaterial provided we study enough to train our mind, I do not believe that the selection of courses has the importance generally attached to it. I do believe that the educational value of a subject depends more upon its mind training power than upon the specific kind of knowledge which it aims to impart. Hence the value

of mathematics, even for those who never expect to be called upon to perform after leaving college, more than the four elementary operations. It is, of course, evident that the natural bend of the mind should be followed in the selection of studies—that persons with decided scientific inclination should take up chiefly scientific studies, those with literary taste, classical subjects, etc.—but the value of greater specializing for the purpose of education is, to me, becoming daily more doubtful. A scientific education should be obtained in four years during which a sound knowledge of the pure sciences and the rudiments of the most important applied sciences should be acquired, thereby securing the desired mental attitude and scientific way of thinking. If the student then desires to take up as his life's work the application of science in some well recognized field such as one of the various engineering professions, his general scientific training should be followed by a course of two years of as practical a character as possible and devoted exclusively to applied science in his chosen field. During these two years specializing could hardly be carried too far. The student for instance, should make up his mind, not only to specialize in mining and metallurgy but in mining *or* metallurgy, and in the latter case whether his special training will be in the metallurgy of iron and steel or of the non-ferrous metals. Should he select iron and steel, his two years of post graduate work should be devoted exclusively to that one subject or practically so. In short, for the purpose of education and general fitness to take up a man's burden let us provide a program of study as broad as possible avoiding specialization, while for preparation in any one field of applied science let us frankly face the need of specializing; at this stage it could hardly be too excessive. The prevailing practice is, of course, very different, the young man being made to specialize upon entering college or possibly upon beginning his second year, and, therefore, at a period when very few can make an intelligent choice. Should he then decide to qualify himself for useful work in the manufacture of iron and steel he is required to follow for three or four years a number of prescribed studies of mining methods, ore dressing, the metallurgy of the non-ferrous metals, while learning relatively very little of his chosen subject, iron and steel. His general scientific training is narrowed while the specialized knowledge acquired is not the kind he wanted. He leaves college a less broad man than what he might have been with scientific information in his chosen field too limited for immediate useful application. Nor can it be reasonably argued that six years is too

*Paper presented before the Congress of Technology at the 50th Anniversary of the Granting of the Charter of the Massachusetts Institute of Technology.

†Professor of Metallurgy, Harvard University, Cambridge, Mass.

long a period of time to devote to preparation for engineering work, for it would be an admission that professional work in applied science does not call for as broad and careful training as that now required by the best schools for the practice of law or of medicine.

Twenty years ago the science of metallography was practically unknown and it is only within the last 15 years that it has been seriously considered by metal manufacturers and consumers as a valuable method of testing and investigation. That so much has been accomplished in so short a time is highly gratifying to the many workers, practical or scientific, who have contributed by their efforts to the progress of metallography.

To realize the practical importance of metallography, it should be borne in mind that the physical properties of metals and alloys, that is, those properties to which these substances owe their exceptional industrial importance, are much more closely related to their proximate composition than to their ultimate composition, and that microscopical examination reveals, in part at least, the proximate composition of metals and alloys, whereas chemical analysis seldom does more than reveal their ultimate composition. It will bear repeating that from the knowledge of the proximate composition of a certain industrial metal or alloy we are able to infer its properties and, therefore, predict its adaptability, with a much greater degree of accuracy than if we knew only its ultimate composition. The analytical chemist may tell us, for instance, that a steel which he has analyzed contains 0.50 per cent of carbon, without our being able to form any idea as to its properties, for such steel may have a tenacity of some 75,000 lbs. per square inch or of some 200,000 lbs., a ductility represented by an elongation of some 25 per cent or practically no ductility at all; it may be so hard that it cannot be filed or so soft as to be easily machined, etc. The metal microscopist, on the contrary, on examining the same steel, will report its structural, *i. e.*, its proximate, composition, informing us that it contains approximately 50 per cent of ferrite and 50 per cent of pearlite and we know at once that the steel is fairly soft, ductile and tenacious, or he may report the presence of 100 per cent of martensite and we know that the steel is extremely hard, very tenacious and deprived of ductility. Which of the two reports is of more immediate practical value, the chemist's or the metallographist's? Surely that of the metallographist. Nor is it only in the domain of metals that we find such close relationship between properties and proximate composition, for, on the contrary, it is quite true of all substance. How many organic bodies, for instance, have practically the same ultimate composition and still are totally unlike in properties because of their different proximate composition, *i. e.*, different grouping and association of the ultimate constituents? If we were better acquainted

with the proximate composition of substances many unexplained facts would become clear to us. Unfortunately the chemist too often is unable to give us positive information in regard to the proportion of the ultimate constituents only his reference to proximate analysis being of the nature of speculation. Ultimate analysis has reached a high degree of perfection in regard to accuracy as well as to speed of methods and analytical chemists have built up a marvelous structure calling for the greatest admiration. Their searching methods never fail to lay bare the ultimate composition of substances. But how much darkness still surrounds the proximate composition of bodies and how great the reward awaiting the lifting of the veil! The forceful and prophetic writing in 1890 of Professor Henry V. Howe naturally comes to mind. Speaking of the properties and constitution of steel, Professor Howe wrote

"If these views be correct, then, no matter how accurate and extended our knowledge of ultimate composition, and how vast the statistics on which our inferences are based, if we attempt to predict mechanical properties from them accurately we become metallurgical Wigginses. * * *

"* * * ; ultimate analysis never will; proximate analysis may, but by methods which are not yet even guessed at, and in the face of fearful obstacles.

"How often do we look for the coming of the master mind which can decipher our undecipherable results and solve our insoluble equations, while if we will but rub our own dull eyes and glance from the petty details of our phenomena to their great outlines their meaning stands forth unmistakably; they tell us that we have followed false clues, and paths which lead but to terminal morasses. In vain do we flounder in the sloughs and quagmires at the foot of the rugged mountain of knowledge seeking a royal road to its summit. If we are to climb, it must be by the precipitous paths of proximate analysis, and the sooner we are armed and shod for the ascent, the sooner we devise weapons for this arduous task, the better.

"By what methods ultimate composition is to be determined is for the chemist rather than the metallurgist to discover. But, if we may take a leaf from lithology, if we can sufficiently comminute our metal (ay, there's the rub!), by observing differences in specific gravity (as in ore dressing), in rate of solubility under rigidly fixed conditions, in degree of attraction by the magnet, in cleavage, lustre, and crystalline form under the microscope, in readiness of oxidation by mixtures of gases in rigidly fixed proportions and at fixed temperatures, we may learn much.

"Will the game be worth the candle? Given the approximate composition, will not the mechanical properties of the metal be so greatly influenced by slight and undeterminable changes in the crystalline

form, size, and arrangement of the component minerals, so dependent on trifling variations in manufacture, as to be still only roughly deducible?"

The above was written before the days of metallography or at least when metallography had barely appeared in the metallurgical sky and when no one yet had fancied what would be the brilliant career of the newcomer. Metallography has done much to supply the need so vividly and timely depicted by Professor Howe, precisely because by lifting a corner of the veil hiding from our view the proximate composition of metals and alloys it has thrown a flood of light upon the real constitution of these important products. Has the game been worth the candle? Will anyone hesitate to answer in the affirmative Professor Howe's question?

Professor Howe with his usual acumen and conscious of the fact that proximate analysis while likely to reveal a great deal more of the constitution of metals than ultimate analysis ever could, might still leave us in such ignorance of their physical structure as to throw but little additional light upon the subject. His fear was certainly well founded and surely if the proximate composition had been obtained by chemical analysis it would indeed have told us little of the structure or anatomy of the metals. In the domain of proximate composition chemistry can not do more for the metallurgist than it does for the physician. Invaluable information chemistry does give without which both the physician and the metallurgist would be in utter darkness, but it throws little or no light upon the anatomy of living or inanimate matter. Its very methods which call for the destruction of the physical structure of matter show how incapable it is to render assistance in this, our great need. The parallel drawn here between metals and living matter is not fantastic. It has been aptly made by Osmond who said rightly that modern science was treating the industrial metal like a living organism and that we were led to study its anatomy, *i. e.*, its physical and chemical constitution; its biology, *i. e.*, the influence exerted upon its constitution by the various treatments, thermal and mechanical, to which the metal is lawfully subjected; and its pathology, *i. e.*, the action of impurities and defective treatments upon its normal constitution.

Fortunately metallography does more than reveal the proximate composition of metals. It is a true dissecting method which lays bare their anatomy, that is, the physical grouping of the proximate constituents, their distribution, relative dimensions, etc., all of which necessarily affect the properties, for two pieces of steel for instance, might have exactly the same approximate composition, that is, might contain, let us say, the same proportion of pearlite and ferrite and still differ quite a little as to strength, ductility, etc., and that because of a different structural arrangement of the two proximate constituents; in other words because of unlike anatomy.

It is not to be supposed that the path trodden during the last score of years was at all times smooth and free from obstacles. Indeed the truth of the proverb that there is no royal road to knowledge was constantly and forcibly impressed on the mind of those engaged in the arduous task of lifting metallography to a higher level. Its short history resembles the history of the development of all sciences. At the outset a mist so thick surrounds the goal that only the most courageous and better equipped attempt to pierce it and perchance they may be rewarded by a gleam of light. This gives courage to others and the new recruits add strength to the besieging party. Then follow the well known attacking methods of scientific tactics and strategy and after many defeats and now and then a victorious battle the goal is in sight—but only in sight and never to be actually reached, for in our way stands the great universal mystery of nature; what is matter? What is life?

Nevertheless there is reward enough for the scientist in the feeling that he has approached the goal, that he has secured a better point of vantage from which to contemplate it. The game was worth the candle. And if scientific workers must necessarily fail in their efforts to arrive at the true definition of matter, whatever be the field of their labor, they at least learn a great deal concerning the *ways of matter*, and it is with the ways of matter that the material world is chiefly concerned. Hence, the usefulness of scientific investigation, hence the usefulness of metallography.

Like any other science with any claim to commercial recognition, metallography has had first to withstand the attack and later to overcome the ill-will and reluctance of the so-called "practical man" with a decided contempt for anything scientific. He represents the industrial philistine clumsily standing in the way of scientific application to industrial operations. Fortunately, while his interference may retard progress, it cannot prevent it. Had he had his own way neither the testing machine, nor the chemical laboratory, nor the metallographical laboratory, nor the pyrometer, would ever have been introduced in iron and steel works.

In metallography as in other fields of research, American workers, with very few exceptions, have been quite willing to let Europeans perform the arduous and general unrewarded task of the pioneer, being content to wait, before entering the field, until practical results were fairly in sight. Such course, which is never to be concealed, becomes intolerable when accompanied, as it so frequently is, by the coarser boasting attitude of the man believing himself smarter than his neighbor whom he regards in the light of a cat drawing the chestnut from the fire. America, barring brilliant exceptions like Richards at Harvard, and Noyes at Massachusetts Institute of Technology, does not as yet do her share of the pioneer's work in investigations which do not give evident indications of quick commercial returns. The

unselfish, nay self-sacrificing spirit of the true scientist is of far rarer occurrence in the United States than it is in Europe and especially in France. America has not yet produced a Pasteur nor a Berthelot, intellectual giants, profound scientific thinkers, whose conception of the duty of the scientist as a man is so lofty that they have despised the wealth within their easy reach to devote themselves unreservedly to the betterment of their country or rather of the world, for they are morally so great that the entire world becomes their fatherland; humanity claims them.

Speaking in 1904 of the practical value of metallography in iron and steel making, I wrote the following, which it may not be out of place to reproduce here: "History, however, must repeat itself, and the evolution of the metallographist bids fair to be an exact duplicate of the evolution of the iron chemist; the same landmarks indicate his course: distrust, reluctant acceptance, unreasonable and foolish expectation from his work, disappointment because these expectations were not fulfilled and finally the finding of his proper sphere and recognition of his worth. The metallographist has passed through the first three stages of this evolution, is emerging from the fourth and entering into the last. For so young a candidate to recognition in iron and steel making, this record is on the whole very creditable."

We may say today that he has definitely entered the last stage and that the adverse criticism still heard from time to time, generally from the pen or mouth of ignorant persons, are like the desultory firing of a defeated and retreating enemy.

In the United States alone the microscope is in daily use for the examination of metals and alloys in more than two hundred laboratories of large industrial firms, while Metallography is taught in practically every scientific school or technical school.

Arthur D. Little, Inc., Chemists and Engineers, Boston, Mass., have been selected to organize and direct an Engineering Department for the General Motors Co., Detroit, Mich. Mr. D. T. Randall, the resident director, has been connected with the former organization for some years, and has had extended experience in engineering problems as well as in the equipment and direction of several testing laboratories. Mr. Randall was formerly professor of mechanical engineering at the University of Illinois, and later was in charge of the fuel investigations and producer and gas engine tests carried on by the U. S. Government. He will be assisted in his new position by several men detailed from the Arthur D. Little, Inc., organization.

The total quantity of Portland, natural and puzzolan cements produced in the United States during 1910 amounted to 76,934,675 bbls., valued at \$68,052,771. This is an increase of 10,244,960 bbls. over the 1909 production.

TROPENAS CONVERTER WITH DOUBLE DOOR BOTTOM.

The Tropenas Converter Co., New York, according to *The Iron Age*, recently transformed a crucible steel foundry into a converter foundry without suspending operations. Up to Feb. 21 the Union Steel Casting Co., Boston, was making small steel castings by the crucible process, having four batteries of 10 crucibles each. To make room for the converter two of the furnaces were torn out, while the other two remained in continuous operation. On Feb. 22 the plant was shut down, the finishing touches were put to the new equipment, and on Feb. 23 the first blow was made, which proved to be perfectly satisfactory.

One of the main reasons for changing from the crucible to the converter process was the high price of the finished product due to the cost of crucibles, the company's allowance for this item being an average of 1 cent per pound. As against a monthly crucible bill of \$600 for a small foundry, the necessary repairs to the lining of a small converter are put at less than \$100 a year.

After 100 blows of the new "baby" converter, only one casting weighing a few ounces was returned. Tests showed also that the converter steel met the specifications readily and most of the time exceeded them.

With the installation of this Boston plant the Tropenas Converter Co. brought out its new drop bottom converter. The criticism had often been made that in a one piece converter it was practically impossible to cool the lining quickly enough after a day's blowing, so as to allow a man to enter the converter the next morning and do the necessary patching. For a foundry wanting to cast every day it meant that a second converter had to be installed. Finally A. Tropenas, inventor of the process, thought of equipping the converter with a double door bottom, similar to that in use on the cupolas of iron foundries. The plan had been tested out in Europe, but the converter in Boston was the first one of this kind in the United States.

The procedure is as follows: After blowing, the bottom doors are dropped and the lining forming the bottom of the converter is knocked out. The opening thus created allows the air to circulate freely and the converter is perfectly cold every morning. To use the converter again the doors are closed, a special composition bottom lining rammed up on top of them, and by the time the converter is heated enough to receive the iron the bottom is entirely dried out. The new bottom has an added advantage in leaving a very large opening through which a man can enter the converter and do the necessary patching without being cramped for room.

The Union Steel Casting Co. makes a specialty of very small castings and employs only 12 molders. The following is the daily schedule:

7:00 a.m. arrival of workmen.
 7:00 a.m. to 7:45 a.m. patching up of converter and cupola.
 7:45 a.m., fire started in converter.
 8:15 a.m., fire started in cupola.
 10:25 a.m., blast on cupola.
 10:45 a.m., first tap from cupola.

No. of blow.	Blow started.	Blow ended.	Duration of blow.
1.....	10:54 a.m.	11:05 a.m.	11 min.
2.....	11:24 a.m.	11:35 a.m.	11 min.
3.....	11:57 a.m.	12:08 p.m.	11 min.
4.....	12:28 p.m.	12:38 p.m.	10 min.
5.....	12:59 p.m.	1:09 p.m.	10 min.
6.....	1:30 p.m.	1:40 p.m.	10 min.
7.....	2:02 p.m.	2:12 p.m.	10 min.
8.....	2:35 p.m.	2:45 p.m.	11 min.
3:00 p.m., pour off.			
3:00 to 5:00 p.m., molders resume molding and laborers shake out molds and temper the sand.			

The approximate capacity of the newly installed converter is 1,200 lbs., making the amount of steel poured in a day 9,600 lbs., all taken out of the converter in bull ladles.

It has been discovered by a man in Hanover, Germany, states the *Australian Mining Journal*, that if hard water, previous to being fed into a boiler, were allowed to run over a plate of aluminum set at an angle and exposed to sunlight or diffused daylight, no scale would be formed through the action of such water on the boiler plates or tubes. The solid matter contained in the water deposits as a fine mud, which is easily removable on the lowest part of the boiler, consequently the tedious process of chipping off the scale is obviated. This process was tried at the Proprietary Company's Port Pirie Works, Broken Hill, Australia. Two plain aluminum sheets, each 4 ft. by 2 ft., were fixed in a frame at an angle of 59 degrees, facing the sun. Water from the main was then allowed to flow over the plates from a perforated pipe fixed at the upper edge. A sufficient quantity of the water was run over the plates in the daytime and stored to supply a Lancashire boiler for 24 hours. Mr. J. F. H. Hill, responsible for this treatment, states that the cause of the phenomena has not been ascertained.

The world's production of zinc for the last three years, according to M. M. Rudolf Wolff, Kreuger & Co., was as follows:

	1910. Tons.	1909. Tons.	1908. Tons.
Belgium	169,998	164,511	162,189
Silesia	138,100	140,625	138,092
United Kingdom of Great Britain	86,262	79,152	72,874
France	50,586	49,237	48,701
Roumania	60,876	58,799	53,586
Holland	20,661	19,233	16,983
Poland	12,500	12,926	13,941
Austria	13,200	13,146	11,889
Spain	6,440	6,011	6,263
Italy			75
Total Europe	558,623	543,646	524,593
Australia	600		1,169
United States	238,770	239,478	183,040
Total production	797,993	783,124	708,802
Average price in London.....	23-0-0	23-3-0	20-3-6

AN ELECTRIC FURNACE FOR HEATING BARS AND BILLETS.*

By THADDEUS F. BAILEY.

The possibility of an electric furnace for heating metal without melting it was suggested by the writer some years ago on noting the very wasteful and thermally inefficient oil-fired furnaces for heating forging stock used in a comparatively modern plant in northern Ohio.

A calculation of the thermal efficiency of the furnaces in question, which were said to be of the most modern construction, showed that less than 4 per cent of the heat value of the fuel was delivered into the steel. The technical press and various publications at the time contained a number of descriptions and tests of electrical furnaces then in use in the chemical industry, and also of furnaces for melting and refining steel. The thermal efficiency of these furnaces was given at from 40 to 80 per cent, and it was at once apparent that if a thermal efficiency of even 40 per cent could be realized in a furnace for heating stock such as was heated in oil furnaces at a thermal efficiency of only 4 per cent, that a very considerable loss could be allowed in the means for converting the heat value of any fuel into electrical energy for use in electric heating furnaces, and still effect a saving.

Taking even a case where the least efficient of the modern prime movers—the non-condensing steam engine—was used for generating power, allowing a net thermal efficiency of 5 per cent for conversion of the heat value of the local into electrical energy, and assuming an electrical furnace efficiency of 40 per cent, the net over-all efficiency for the entire heating equipment would be 2 per cent. The fuel for creating the steam, however, would cost only one-third as much per heat unit as would the fuel oil for oil furnaces. The commercial efficiency, then, as far as cost of fuel alone was concerned, would be $33\frac{1}{3}$ per cent in favor of the electric equipment, in spite of the low efficiency of the steam engine. Considering steam engine—low pressure turbine units, or gas engine driven units in place of the steam engine unit above considered, the probable efficiencies of electrical furnace equipments seemed favorable enough to deserve a place in the industry, while in plants where there was a large supply of exhaust steam available, as is the case in the majority of forge plants, electric energy could be so cheaply produced by means of the low-pressure turbine that no type of direct-fired furnace could hope to compete with such equipment in cost of heating. The only question, then, seemed to be whether an electric heating-furnace could be designed to successfully heat metal in thermal efficiencies above a given limit.

The greatest difficulties encountered in the operation of the first experimental furnaces were due largely to lack of sufficient electric current and con-

*Paper presented at the 19th general meeting of the American Electrochemical Society in New York City, April 6-8, 1911.

trol devices. Suitable equipment was, however, obtained by installing the 120 kw. generator and transformer equipment mentioned later at the plant of the Transue & Williams Co., Alliance, O., which company very generously supplied the power for driving the generator from their gas engine for all the runs on the furnace.

The following is a description of the furnace evolved out of the experiments commenced over four years ago. Reference may be made to Fig. 1, a right-left section; to Fig. 2, a front-back middle section; and to Fig. 3, sectional plan of the furnace.

The furnace is of the resistance type, and consists

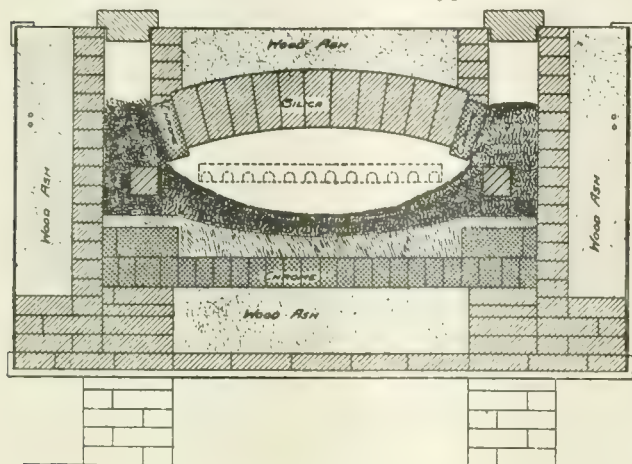


Fig. 1.

essentially of two carbon electrodes, spaced from each other, and an intermediate resistance body of a carbonaceous composition in which the heat is generated. In the space directly above the resistance material and directly under the roof of the furnace is placed the metal to be heated, a ledge at the rear

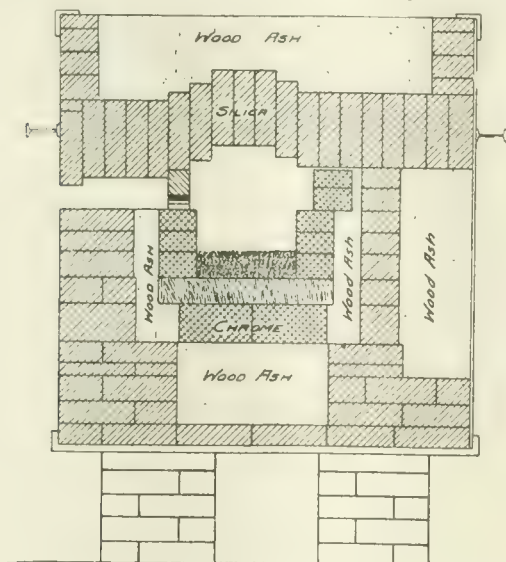


Fig. 2.

of the furnace and one at the opening supporting the bars or billets as the case may be. The electrodes enter the furnace through the rear wall, and are placed slightly convergent, so that the path of the electric current will be shorter from electrode to electrode at the front of the furnace than at the back.

This arrangement compensates for the cooling effect, which is greater in the front of the furnace on account of the opening and the charging of the cold material. The electrodes entering the furnace from the rear present comparatively large contact surfaces to the resistance material, without the use of special electrode sections. The electrodes are also placed in a plane above the resistance material, which throws the shortest path for the electric current in the upper part of the resistance body and nearest the metal. This is a feature that seems almost indispensable to the successful operation of a furnace of this character, since in the furnaces constructed without this feature the cooling of the upper part of the resistance body by the cold steel caused the electric current to take a lower path, in which portion of the body the temperature became very high and melted out the bottom linings and the bottom of the furnace itself, and at the same time did not maintain the heating chamber at a temperature high enough to heat the metal.

The resistance material is composed of coke or coal. Crushed foundry coke passing over a 0.25-in. (6mm.)

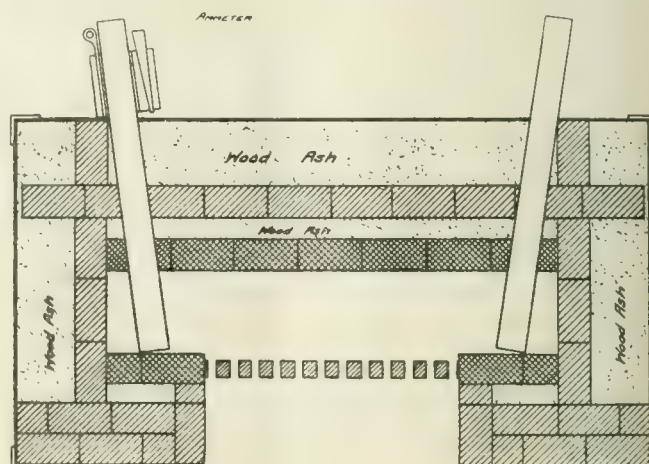


Fig. 3.

ring and through a 0.37-in. (9 mm.) ring giving the most satisfactory and uniform results. For a 40-kw. furnace, heating 150 lbs. (69 kg.) of metal per hour, the distance between electrodes should be 36 ins. (90 cm.) and the cross-sectional area about 24 sq. in. (150 sq. cm.). To maintain an electrical input of 40 kw., a voltage of 200 would be required with a coke resistance body; if the body is composed of coal, the voltage should be about 150 volts. It is extremely difficult to determine the effective cross-sectional area of the material through which the electric current passes, as at working temperatures the lining materials become almost as good conductors of electricity as the resistance material itself. The resistance body as it approaches the electrodes has a very greatly increased cross-section, so as to provide a better contact with the electrodes, prevent undue heating at the point of contact, and prevent, to as great an extent as possible, the dissipation of heat through resistance to the passage of the current except at that part of the resistance body directly under the metal. Thus in a

way these increased cross-sections perform the mission of electrodes, as they conduct the current from the electrodes proper, which are located in a protected part of the furnace, to the part where the transformation into heat is desired, and they do this without any considerable heat being generated in these portions themselves.

The most serviceable lining material so far obtained is made of chrome brick and chrome ore, and while a certain reducing action takes place between the chrome ore and brick and the carbon of the resistance material, it is not of such nature as to seriously impair its usefulness as a lining material. This chrome lining is carried up to at least one course above the resistance material. The lining is readily accessible for repairs or inspection by removing the three courses of silica brick directly under the furnace opening.

The top of the furnace is composed of silica arch brick, and is 9 ins. thick, and clamped from front to back. A row of chrome arch brick is placed at each side of the roof to prevent contact of the silica brick with the carbon in the hoppers above the electrodes.

The sides of the furnace are composed of silica brick and, in the larger furnaces, are incased in a sheet steel frame. Insulating material is placed between the silica brick and the casing.

The furnace base consists of a heavy cast iron plate supported by cast iron legs or brick piers.

The cable terminals are connected with the electrodes by means of iron sleeves, copper straps and iron wedges. In making the connection, the iron sleeve is placed over the end of the electrode and copper straps placed against the four sides of the electrode. The copper cable terminal is then placed between one of the copper strips and the iron sleeve, and wedges driven in on the opposite side between the strap and sleeve, and a firm contact thus secured. Wedges are then driven in on the other two sides between the copper strips and the sleeve. In this way the current flow is equally distributed on all four sides of the electrode by means of the encircling iron sleeve.

The furnace is controlled by means of a regulating transformer and controller. The transformer gives full rated output at the lowest voltage rating. While hand regulation is usually provided, the regulation may be made automatic at a slight additional cost. A voltage range of from 33 to 50 per cent is usually provided, depending on the operating and starting requirements.

Each furnace is provided with a single-panel switchboard having mounted on it a voltmeter, ammeter, watt-hour meter, circuit-breaker and oil switch. The voltmeter and ammeter indicate at all times the conditions in the interior of the furnace, as irregularities are readily noticeable on the instruments. The watt-hour meter enables accurate data to be kept on the current consumption for heating any particular class of work.

The temperatures that may be obtained are limited solely by the electrical input and the temperatures allowable by the refractory linings of the furnace. The temperature usually maintained in a furnace for the rapid heating of stock for automobile forgings is such as to cause the fusion of the ends of the silica brick forming the roof, and is probably not less than 3200° F. (1765° C.). The resistance material, of course, reaches a much higher temperature. When working at the highest temperatures usually met in forge practice, a renewal of the lining of the bottom of the furnace will be required every two to three weeks.

The replacing of the lining or portions of it and the replenishing of the resistance material are the only items that may be classed as renewals, and should not exceed, in furnaces heating 150 pounds of metal per hour, working at the highest temperatures, an average of 20 cts. per day for lining material and 5 cts. per day for resistance material.

The metal being heated is maintained in a reducing atmosphere, the high temperature carbon resistance material giving off a gas of a reducing nature. The electric furnace is the only furnace built in small units in which both a high temperature and a reducing atmosphere may be maintained. The saving in oxidation loss is very considerable in small furnaces of this character, as the loss from oxidation in the usual combustion furnace is the cause of the loss of a large number of forgings, due to the fact that a piece of stock, having lost a considerable amount of metal by the oxidizing flame of the combustion furnace, does not contain enough metal to fill the dies.

The thermal efficiency of the furnaces varies with the size, the percentage of operation at full capacity and the ruling temperature required. Efficiencies of from 33 to 65 per cent may be expected in furnaces with heating capacities of from 120 pounds (55 kg.) per hour to 1,000 pounds (455 kg.) per hour.

Reference was made in the first part of this paper to the experimental furnace installed at the plant of the Transue & Williams Co., and a description of the apparatus and results obtained may be of interest.

The power for driving the generator was supplied by a 12 by 16-in. (30 by 40 cm.) Buckeye four-cylinder twin single-acting gas engine, which not only drove the generator for supplying current to the furnace, but also for driving a portion of the motors in the machine shop, so that only part of the time was sufficient power available for furnace operation. Belted to the fly-wheel of the gas engine was the 120-kw., 2-phase, 440-volt revolving field alternator, only one phase of which was connected up. The leads from this phase were connected with the low-tension terminals of a 37 5-kw., 440 to 2,200-volt transformer. The current was then stepped down by means of a 2,200-220 or 110-volt transformer, the coils on the low-tension side being so connected through a double-throw switch for either series or

parallel operation, so that a normal voltage of either 110 or 220 volts could be obtained. The variations between 110 volts and 220 volts on the lower voltage and also on the voltages above 220 were obtained by varying the field of the alternator. By this arrangement a possible voltage range could be obtained of from 90 to 300 volts, which was more than ample for conditions of operation. Inability to readily obtain a 2,200-volt generator of proper size accounted for the use of the 440-2,200 step-up transformer. From the double-throw switch the current was conducted to the furnace, located in the forge shop, some 75 ft. (22 m.) distant.

The instruments mounted on the switchboard consisted of a 150-volt voltmeter and a 600-ampere ammeter, both made by the Westinghouse Electric & Mfg. Co. The voltmeter was so connected that when the low-tension coils of the step-down transformer were thrown in parallel, the instrument gave a direct reading, and when in series, the indicated voltage was half the actual voltage.

The furnace was constructed substantially in accordance with the description given in the first part of this paper, except that air spaces were used in place of the insulating material shown in Fig. 2, and it was rated as a 40-kw. unit, having a heating capacity of 120 pounds (55 kg.) per hour, the heating capacity being rated somewhat lower than the usual practice on account of the high temperature required.

The electrodes were 4x4x48 ins. (10x10x120 cm.) long, composed of carbon, and were spaced 36 ins. (90 cm.) from each other. The cross-sectional area of the resistance material was approximately 24 sq. ins. (150 sq. cm.). The walls were composed of 9 ins. (22.5 cm.) of silica brick, and were of rather loose construction, owing to the numerous changes that were made and to the jar from the hammers in the plant. The runs on the furnace were made under rather unfavorable conditions, since the forgings that were made under the hammer that the furnace supplied were very difficult to make, and unless the metal was very hot, it would not fill out in the dies. Much delay was also caused by trouble with both the hammer and the dies, any such delay affecting, of course, the amount of metal heated in the furnace.

The stock heated included 1¼-in. (31 mm.) square, 1½x5⁄8-in. (37.5x15.6mm.) flat, 1½ and 7⁄8-in. (37.5 and 21.9 mm.) round bars in lengths from 22½ ins. (56 cm.) to 6 ft. (180 cm.) long; 4 to 12-in. (10 to 30 cm.) cuts were taken off at each heating and the bars replaced in the furnace. From 7 to 9 bars were heated at a time.

The current efficiency taken over periods of 9 hours when heating the heavier of the above mentioned stock was from 3 to 3½ pounds (1.36 to 1.52 kg.) of metal per kw. hour. Tables I, II, III and IV are logs of typical runs of some of the sizes of stock heated.

A voltage of 250 volts was usually used in starting

the furnace in the morning. From one to two hours was usually required to bring the furnace up to temperature. The voltage was then reduced to about 200 volts when using a coke resistance body, or 150 volts for a coal body, which tensions enabled the required electric input to be maintained.

Some of the various linings used before the chrome lining was selected were magnesia, carborundum and silica. The chrome lining, however, gave by far the most satisfactory and uniform service, lasting three weeks without renewal or repair. As only the bottom lining is subjected to very high temperature, it is practically the only part requiring renewal, and as the usual bridge wall used in oil furnaces is not required, that saving compensates for the cost of renewals of the lining of the electric furnace.

The principal advantages of the electric furnace are its high thermal efficiency, its non-oxidizing atmosphere in the furnace chamber at all temperatures, and its freedom from soot, smoke and the hot contaminating gases of the products of combustion.

There seems little reason to doubt that the electric furnace will soon occupy an important place for heating metal such as is now heated in small direct-fired furnaces.

TABLE I.
Run Made With Solid Magnesia Lining.

Time of Charge-ing.	Volts.	Amp.	Stock.	1st Cut.	2d. Cut.	All Out.
9.50	150	270	6-1¼"□x132"	10.15	10.25	10.45
10.55	160	260	3-1¼"□x 66"	11.11	11.15	11.28
12.23	150	270	8-1¼"□x180"	12.38	12.44	12.55
1.00	153	260	8-1¼"□x180"	1.12	1.20	1.37

TABLE II.
Run Made With Air-Cooled Magnesia Lining.

9.25	127	340	8-1½"x5⁄8"x5' 10"	9.33	9.38	10.42
10.51	122	350	8-1½"x5⁄8"x5' 10"	10.58	11.01	11.19
12.32	128	330	{ Metal unforged } at noon time	12.38	12.41	1.25
1.27	132	330	8-1½"x5⁄8"x5' 10"	1.35	1.38	2.44

TABLE III.
Run Made With Air-Cooled Chrome Lining.

1.20	230	180	5-1½"Ox2' 0"	1.40	1.51	2.07
2.08	230	180	7-1½"Ox2' 0"	2.30	2.34	3.08
3.17	208	200	7-1½"Ox2' 0"	3.30	3.36	3.52
3.55	204	200	7-1½"Ox2' 0"	4.01	4.07	4.29

TABLE IV.
Run Made With Air-Cooled Chrome Lining.

3.55	240	140	8-7⁄8"Ox27"	4.00	4.02	4.25
4.16	172	150	8-7⁄8"Ox 9"	4.30	4.33	4.36
4.37	252	120	9-7⁄8"Ox22"	4.40	4.44	4.55
4.57	280	120	8-7⁄8"Ox 9"	5.01	5.04	5.07

During the year 1910, 266 samples of bituminous road materials used in construction were submitted to certain chemical and physical tests by the chemist of the Massachusetts Highway Commission. A large amount of special experimental work was also carried on to test and prove, if possible, the value of the various analytical methods used in this and other laboratories engaged in such work. As a result of the investigations of this kind carried on since the commission began to study the composition of these materials, certain tests have been worked out that appear to show with a fair degree of satisfaction, from a chemist's point of view, the relative value of the materials for road work.

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NOTES AND COMMENTS.

Ethics in the Practice of Chemistry and Chemical Engineering.

A paper presented at the summer meeting of the American Chemical Society, under the above title, aroused a great deal of discussion, and was, perhaps, the most commented on paper of any delivered during this meeting.

The author brought out facts with which all of us have been familiar; that there were many men practicing as commercial analysts, and as consulting chemists, who ought not to be doing so. Comparison was made between the legal and medical professions, and our own. Suggestions were brought out that possibly existing conditions might be remedied by some method similar to that applied in the case of the other professions mentioned. The author of the paper said that remedies to be applied may come in one of two ways: either outside the profession, that is by the passing of some law regulating

the practice of the profession, or from the inside, that is, by some body with sufficient standing, which could only be The American Chemical Society, to formulate certain regulations to govern the practice of public analysts and consulting chemists. These regulations to be enforced by a committee appointed from and by this Society. The discussion of the paper was mainly along the lines of meeting the difficulties which would be encountered in attempting any such regulation. All of the speakers were agreed that such regulation, if possible, was highly desirable. Most of the speakers however were of the opinion that there were practical difficulties in the way of formulating and enforcing any such rules, which would make it impossible to solve the problem in this way. Hoping that some feasible method might be found the Division of Industrial Chemists recommended to the Council of the Society the appointment of a Committee to consider the question of regulating the practice of the public analyst and the consulting chemist, this committee to report at the winter meeting of the Society, which will be held in Washington.

Briquetting Lignites.

The results of the investigations into the briquetting of lignite have just been published by the Bureau of Mines in Bulletin No. 14. Charles L. Wright, who conducted the tests and who is author of the bulletin, declares that enough testing has been done to indicate that some American lignites equal German lignites in fuel value and can probably be made into briquets on a commercial scale without the use of binding materials.

"Three samples of lignites," says the author, "one from Texas, one from North Dakota, and one from California, were made into satisfactory briquets without the addition of a binder. It was proved that some lignites after having slacked by exposure can be made into briquets without the use of binding material, notwithstanding a general opinion that this could not be done. Cohesion and weathering tests demonstrated that good briquets endure handling and resist weathering much better than the lignite from which they are made.

"The tests described apparently show that the cost of briquetting run-of-mine lignites with a German plant, which was used, would be from \$1.35 to \$1.75 per ton, according to the location of the plant. The cost per ton of briquets, loaded on cars, from a briquet plant at the mine would be, in Texas, \$2.51; in North Dakota, \$3.53; and in California, \$5.24. It must be borne in mind that these figures are only approximate and are subject to wide changes because of local conditions. They apply to briquetting run-of-mine lignite to improve its heat value and weather resisting properties rather than to briquetting slack or waste coal. Since the tests have shown that at least some lignites slacked by exposure to the weather can be made into excellent briquets, it may be possible to utilize lignite slack as well as bituminous slack

and anthracite screenings for briquetting, the two latter materials having been made into briquets on a commercial scale both in this country and abroad.

"Of four samples of raw lignite, three samples contained about 40 per cent moisture and had a fuel value of 6,079 to 6,241 B. t. u., while a Texas lignite, with a moisture content of 33 per cent had a fuel value of 6,840 B. t. u. The percentage of moisture removed in the process of briquetting ranged from 24 to 32 per cent and the heat value of the briquet was 36.5 to 54 per cent higher than that of the raw lignites.

"Excessive moisture in fuel not only causes a waste of useful heat during combustion, because the moisture is vaporized and the vapor superheated, but also is a source of expense to the consumer, who pays freight charges on useless water. For both these reasons lignite briquets have the advantage over raw lignite. In the case of one of the North Dakota lignites the removal of 32 per cent of moisture during briquetting permits a decided lessening of the cost of supplying a consumer with a given number of heat units. The advantage of the briquets in this respect is of especial importance when transportation to a distant market is involved. If the briquets possess no other advantage over raw lignite than their higher heat value, they would be worth 50 per cent more than the raw fuel.

A NEW PROCESS FOR PRODUCING A NEW LOW-PRICED GAS MIXTURE FROM COAL.

United States letters patent have been just issued to Mr. George A. Bronder of 309 Broadway, New York, for a process securing a gas rich in British thermal units, which seems worthy of more than passing mention. This mixture is obtained by extracting all the gas contained in the treated coal and coke, reducing the same to ashes and clinkers, the ammonia and tar being extracted at the same time. The calorific value is about 435 B. t. u. per cubic foot, the specific gravity is about 0.48, and it is non-condensable and highly compressible. In the treatment 1 lb. of coal yields about 20 ft. of gas.

The plant for making this mixture is simple and its cost is hardly more than half that of an ordinary plant of like capacity, and as the cost of manufacture is reduced considerably in consequence, the mixture can be produced at low price.

About 27 ft. will yield 1 h. p., and its heating quality is as stated above. Although it has low candle power (like natural gas) it gives a high flame temperature in the mantle for equal quantities of gas.

Its main use, however, is intended for the heating of residence houses and other edifices, where it compares favorably with the cost of coal. Being easily regulated by a thermostat, the temperature of the

rooms can always be kept even and does away with all storage of coal, the handling of furnace ashes and clinkers and attending the furnace.

In connection with this gas mixture a heater has been tested in a modern house for the past winter, using gas as fuel, and applied to existing steam coils, in the so called indirect way, where fresh air is taken from the outside, heated by the steam coils and distributed over all the rooms in the house. The particular house is a detached one, having 37 windows and 40 doors. It stands on a corner plot of 6 city lots, next to an open plaza. It required 75,000 cu. ft. of gas per month to heat it through the cold winter season.

The heater is built to assure absolute safety, the air required for the gas burners being taken from the outside through a pipe. The heater is absolutely tight, so that no gas can possibly escape into cellar or house, and all waste gases are led into the chimney. The waste gases have a temperature of only 92° to 104°. The heater can be applied to existing heating systems, either of steam or hot water types.—*American Gas Light Journal*.

Statistics of the stocks of cotton in spinners' hands throughout the world on March 1, 1911, show that, compared with 12 months ago, the stocks are smaller in practically every country. In Great Britain the total supplies amount to 399,021 bales, as compared with 415,182 bales at the same time last year. The figures for the United States are 1,525,000 bales, against 1,674,000 bales 12 months ago. The figures for all countries are 4,060,740 bales, as compared with 4,166,688 bales in 1910. The analysis of the statistics on the basis of stocks in each country calculated per 1,000 spindles gives Great Britain as 8.20 bales, against 8.50 last year and 9.72 in 1909. The figures for Germany are 30.62 bales, as compared with 34.51 bales last year and 04.86 in the year before. For the United States the figures are 53.51 bales, against 59.79 one year ago and 65.78 in the year before. The country which holds the largest stocks is Japan, the figures being 166.79 bales, as compared with 120.85 last year and 131.77 bales in 1909. Out of the estimated spinning spindles of the world in work of 135,596,724, returns have been secured from firms owning 122,226,091 spindles. In Great Britain returns have been sent in from the owners of 48,688,061 spindles, out of a total of 53,859,247 spindles. There are 35,565,127 spindles engaged on American, East Indian, and sundry cottons, while the spindles engaged on Egyptian cotton number 13,122,934.

It is stated that 137,700 tons of phosphates were exported from Christmas Island in 1910, as compared with 105,481 tons in the previous year.

RECENT PATENTS.

The following patents relating to industrial and engineering chemistry are reported by C. L. Parker, solicitor of chemical patents, McGill building, Washington, D. C.:

994,120—*Manufacture of Sodium Bichromate.*

Thomas John Ireland Graig, Manchester, England, assignor to Peter Spence & Sons, Limited, Manchester, England. Pat. June 6, 1911.

994,139—*Paint and Varnish Remover.* Carleton Ellis, White Plains, N. Y., assignor to Chadeloid Chemical Company, New York, a corporation of West Virginia. Pat. June 6, 1911.

994,273—*Safety Powder for Blasting.* Gershon Moore Peters, Cincinnati, and Milton Fletcher Lindsley, Kings Mills, Ohio. Pat. June 6, 1911.

A safety powder consisting of nitrite of ammonia, nitro-cellulose, a nitro-hydro-carbon solution of asphalt and antacid, and a heat reducing substance.

994,435—*Process of Making Ammonium Sulfate and Sulfite.* Franz Wolf, Bochum, Germany. Pat. June 6, 1911.

994,437—*Process of Preparing Camphor.* Ossian Aschan, Helsingfors, Russia, and Wilhelm Kempe, Berlin, Germany, assignors to Chemische Fabrik auf Actien (vorm E. Schering), Berlin, Germany. Pat. June 6, 1911.

994,446—*Leather Dressing.* Thomas Alex. Drake, Jersey City, N. J. Pat. June 6, 1911.

994,502—*Process of Utilizing Zinc-Sulfite Sludge.* Alan A. Claffin, Medford, Mass. Pat. June 6, 1911.

994,508—*Process of Bleaching and Softening Jute Fiber.* Samuel A. Flower, New York, N. Y.; Charles E. Flower, administrator of said Samuel A. Flower, deceased. Pat. June 6, 1911.

This is a process of treating jute fiber with a hot solution of water, an alkali, chloride of lime and soap to loosen the mechanical impurities and convert the jute pigment into a soluble form, then washing with water and wringing out, next treating the fiber in a solution of water, chlorid of lime, carbonate of soda and glycerin, then treating the fiber with a mixture of glycerin, chlorid of lime, soap and carbonate of soda.

994,555—*Process of Condensing Milk.* James Christian Alexander, Roseburg, Ore. Pat. June 6, 1911.

The process comprises placing the milk in a closed receptacle, creating a vacuum within the receptacle to remove the air from the milk, agitating the milk after the vacuum is created and the air removed from said milk whereby such agitation will not cause the milk to foam, and applying a freezing medium to the exterior of the receptacle to cause the water in the outer portion of the mass of the milk to freeze, the other elements of the said outer portion of the milk becoming mixed with the unfrozen milk.

994,646—*Process of Purifying Raw Juices in the Sugar Manufacture.* Mieczyslaw Kowalski, Lemberg, Austria-Hungary. Pat. June 6, 1911.

994,841—*Explosive.* Harold Hibbert, Wilmington, Del., assignor to E. I. du Pont de Nemours Powder Company, Wilmington, Del., a corporation of New Jersey. Pat. June 13, 1911.

994,889—*Method of and Apparatus for Condensing Zinc Vapor to Liquid Metal.* Charles Thierry, Paris, France, and John Thompson, New York, N. Y. Pat. June 13, 1911.

994,931—*Method of Water-Proofing Fabrics.* Pierre O. Keilholtz, Baltimore, Md. Pat. June 13, 1911.

995,038—*Acid Ferric Phosphor Tartrate.* Carl Sorger, Niederlahnstein-on-the-Rhine, Germany. Pat. June 13, 1911.

995,105—*Process of Obtaining Potash Salts from Feldspar.* Firman Thompson, Newark, Del. Pat. June 13, 1911.

The process consists in reducing the rock to powdered form, mixing therewith an alkali metal acid sulfate and an alkali metal chlorid, heating the mixture, and separating the soluble portion from the remaining portion of the mixture.

995,113—*Aluminum Alloy.* Conrad Hubert Heinrich Claessen, Berlin, Germany. Pat. June 13, 1911.

995,187—*Explosive.* Joseph Sayers, Walter Atkinson Wilson and James Thorburn, Ardeer, Scotland, assignors to E. I. du Pont de Nemours Powder Company, Wilmington, Del. Pat. June 13, 1911.

995,366—*Method of Smelting.* John W. Nesmith, deceased, Denver, Colo., by Harper M. Orahood, executor, Denver, Colo., assignor to Colorado Iron Works Company, Denver, Colo., a corporation of Colorado. Pat. June 13, 1911.

This is a method of smelting coarse and fine ores consisting in generating a smelting heat in a combustion chamber, converting the coarse ore to a fused or molten condition by such heat, and projecting fines horizontally into the combustion chamber where they are treated in suspension.

995,373—*Explosive.* Joseph Sayers, Walter Atkinson Wilson and James Thorburn, Ardeer, Scotland, assignors to E. I. du Pont de Nemours Powder Company, Wilmington, Del., a corporation of New Jersey. Pat. June 13, 1911.

995,481—*Process of Effecting Reduction and Producing Ferrochromium.* Edgar F. Price, Niagara Falls, N. Y., assignor by mesne assignments, to Central Trust Company of New York, trustee, a corporation of New York. Pat. June 20, 1911.

995,510—*Method of Producing Transparent Camphor in Shaped Pieces.* Otta Rudolf Daniel Witt, Hamburg, Germany. Pat. June 20, 1911.

995,542—*Method of Treating Finely-Divided Substances Containing Iron Compounds.* Tom Cobb King, Marion, Ala., assignor to National Metallurgical Company, Jersey City, N. J., a corporation of New Jersey. Pat. June 20, 1911.

995,579—*Explosive Compound and Process of Manufacturing the Same.* David M. Stirton, Cascade, British Columbia, Canada. Pat. June 20, 1911.

995,724—*White Enamel*. Carl Rosenzweig, Vienna, Austria-Hungary. Pat. June 20, 1911.

995,740—*Process for the Manufacture of Incandescent Mantles*. Jacques Visseaux, Lyons, France. Pat. June 20, 1911.

995,820—*Beer and Method of Preparing Same*. Leo Wallerstein, New York, N. Y. Pat. June 20, 1911.

995,852—*Process of Strengthening Cellulose*—Xavier Eschaliér, Villeurbanne, France. Pat. June 20, 1911.

995,894—*Process of Manufacturing Manure from Raw Phosphates*. Wilhelm Palmaer, Stockholm, Sweden. Pat. June 20, 1911.

The process consists in subjecting mineral phosphate to the action of hydroxid acid and adding a solution of the hydroxid of alkali metal to the acid solution of the phosphate for precipitating the latter as bicalcic phosphate.

995,897—*Process of Making Phosphorus Pentoxid and Titanium Compounds*. Samuel Peacock, Chicago, Ill., assignor to American Cyanamid Company, New York, N. Y., a corporation of Maine. Pat. July 20, 1911.

996,011—*Method of Making Nitrogen Compounds*. Albert R. Frank, Halensee, near Berlin, Germany, assignor to Societa Generale per la Cianamide, Rome, Italy, a corporation of Italy. Pat. July 20, 1911.

996,094—*Method of Making Fusible Compounds of Aluminum and Recovering Aluminum Therefrom*. Lucius Richard Keogh, Ottawa, Ontario, Canada, assignor of one-half to Clifton Ashton Douglas, Ottawa, Canada. Pat. June 27, 1911.

996,179—*Process of Producing Metals from Ores*. Raymond Patterson Wheelock, Searchlight, Nev. Pat. June 27, 1911.

996,182—*Process of Making Incandescent Lighting-Bodies*. Oscar Weiderhold, Jersey City, N. J. Pat. June 27, 1911.

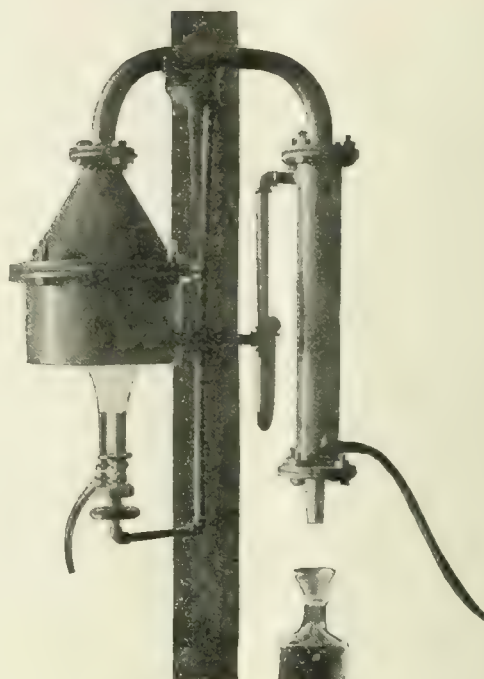
996,380—*Process of Purifying Tin Scraps Preparatory to Dctinning the Same*. Carl Von Der Linde, Crefeld, Germany. Pat. June 27, 1911.

996,400—*Process for the Manufacture of Alcohol*. Aquiles Ernesto V. Castro, Merida, Mexico. Pat. June 27, 1911.

996,491—*Magnetic Preparation of Ores*. Alfred Arthur Lockwood, London, England, assignor to Murrex Magnetic Company, Limited, London, England. Pat. June 27, 1911.

996,509—*Tanning Process*. Ottoka Henry Nowak, Chicago, Ill., assignor to Dermiforma Company of America, a corporation of Illinois. Pat. June 27, 1911.

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THE DEBT OF THE MANUFACTURER TO THE CHEMIST.*

By HARVEY J. SKINNER.†

The enormous progress and changes which have taken place in industry and commerce in the course of the past century may to a large extent be justly attributed to the work of chemists. Such a statement will undoubtedly be regarded by many as a most extraordinary one and open to question, since the proper relation of the chemist to industrial welfare is not generally appreciated. Let us review for a moment some of the achievements of chemistry so that the truth of the above statement may become more apparent.

The great mechanical developments and the more recent electrical ones depend upon the extraction of the metals from their ores by purely chemical processes. The manufacture of steel, so essential in the construction of our railroads, bridges and large buildings, is a most advanced chemical industry and is under strict chemical control at every step of the process. Alloy and tool steels, which are now finding such extended use in the arts, are the direct result of chemical research. Electrical developments have been almost entirely dependent upon copper, and without the chemical processes of refining this metal it is safe to say that electricity would find very limited application in manufacturing work.

The history of aluminum is another instance of an important metal being extracted from the ores by chemical means. In 1856, aluminum sold for \$90 a pound. As a result of improvements in the processes, the price gradually dropped to \$5 in 1886, but was still too high to allow of its general use. About this time, Charles R. Hall, then recently graduated from Oberlin College, made a discovery which forms the basis of the present aluminum industry and which brought the price down to a point where the remarkable characteristics of this metal can be utilized in the arts.

In the textile industry, almost every step is dependent upon chemical science. The use of aniline colors, made possible by the discovery of Perkin, have given to the manufacturer more brilliant and more

permanent colors for his fabrics and at a lower price than the natural colors previously used. This single discovery has led to the manufacture of over 2,000 artificial dyestuffs and become the foundation of an industry in which it has been estimated that \$750,000,000 are invested. It has further given to the producers of coal tar a valuable outlet for a waste product which was not only comparatively useless, but difficult of disposal.

Researches in the chemistry of cellulose have given to us, among other things, artificial silk and smokeless powder. The latter has already grown to a huge industry within the memory of the youngest of us and the development of the former bids fair to become equally extensive. The processes of leather manufacture, paper making and many others which could be mentioned are entirely dependent upon chemical principles. And so we could go on and point out that every manufacturer is dependent in some way or another upon chemical science; and why should it not be so when one considers the true basis of manufacturing?

Manufacturing deals with the modification of material and since all material is subject to chemical laws and its properties are governed by these laws, it becomes apparent that the majority of the manufacturer's problems are those in applied chemistry.

Unfortunately, the average manufacturer, especially if his process is a mechanical one, regards chemistry as something which has to do with drugs and chemicals and has no direct bearing upon his own problems. That manufacturers fail to appreciate their indebtedness to the chemist and how he can improve the efficiency of their processes by studying the chemical properties of their materials is due largely to the fact that the older generation of manufacturers started as factory hands and have worked themselves up through the various grades to managerships and presidencies. Their methods have been rule-of-thumb methods and science has had no meaning to them. Their aim was to make money and the efficiency of their processes was a secondary consideration.

With the growth resulting from the combination of capital and the technically trained men which our universities are turning out, conditions are taking on a new aspect. The larger manufacturers, realizing their debt to the chemist and also that there are still unsolved problems in every factory, are securing the benefits of scientific advice. The smaller manufac-

*Paper presented before the Congress of Technology at the Fiftieth Anniversary of the Granting of the Charter of the Massachusetts Institute of Technology.

†Vice-President, Arthur D. Little, Inc., Boston, Mass.

turer will soon be forced to the same procedure or he will lose in the struggle for industrial existence. The rule-of-thumb method is passing. Guesswork is being replaced by scientific knowledge, and more and more consideration is being given to the underlying principles of the manufacturing processes.

Manufacturing operations based upon chemical processes require control at each step to maintain efficiency. These based upon mechanical processes but still dependent upon "material" demand rigid inspection and control of every material entering into or affecting the cost of the finished product.

All this is the work of the chemist or the testing engineer. It should be his duty to see that every material is purchased on a basis of quality and not of brand, that the finished product meets the proper requirements, and that the yields are as near theoretical as possible.

A laboratory is just as essential to a factory as is an office, and the chemist is just as necessary as the auditor. The records of manufacturing concerns using scientific knowledge will bear out this statement. One mistake common to both the manufacturer and the chemist themselves should be pointed out. Many manufacturers, having been converted to the idea that a chemist can be of assistance in the operation of their plants, oftentimes will employ a recent technical graduate and expect him to solve any question in chemistry. This is an injustice to the young chemist and to the profession itself.

Alan A. Claffin in a recent article has said:

"The employment of a scientific man does not mean the engaging of a recent technical graduate at a salary of \$15 to \$20 a week to test raw materials and report results which are probably erroneous to a foreman who does not understand them, but it means having a man of mature experience as a chemical adviser with two or three recent graduates as working assistants."

No words could be truer or better expressed. The manufacturer does not hire a bookkeeper without actual experience to keep his accounts, neither does he engage a lawyer just out of law school to look after his legal affairs. Then why should he expect the young graduate with a large amount of theoretical knowledge and with little experience to be able to solve effectively the problems which have been troubling him for years?

This condition of affairs is really a serious one and has much to do with the attitude which the average manufacturer takes toward the chemist. It also accounts for the diffidence of the manufacturer in applying chemical science to his problems, and not until the true relation between the chemist and material is more fully realized will the real debt of the manufacturer to the chemist become appreciated.

Owing to the discontinuance of the subsidy of the Japanese Government to sugar-beet growers in Korea most of the land in that country now devoted to beets will be used to grow cotton.

THE PREPARATION OF COAL FOR THE MARKET.*

By PROF. HENRY LOUIS, M. A., D. SC., ETC.

Coal, as it occurs in a coal seam, contains inorganic impurities, consisting of shale or clayey or sandy matter, which may be either finely divided and disseminated through the masses of coal, in the same way, for instance, as the ash of wood is disseminated through the wood, or they may occur in some more concentrated form, often in layers, varying in thickness from sheets of the thinness of paper up to dirt bands which may be several feet in thickness. Furthermore, in the act of coal getting, portions of the adjoining strata, and especially of the floor, are apt to be cut up and mixed in the coal proper. When the inorganic matter is very finely disseminated, its separation is from a practical point of view impossible; when it occurs in larger particles, the coal can be freed from it to a greater or lesser extent, the methods to be adopted depending essentially upon the size of the fragments of dirt.

As regards its composition, the inorganic matter is either of the earthy character referred to above, and called by the miner "shale," or more often "dirt"—and I propose to keep to the word "dirt" as a convenient generic term in this lecture—or else it may consist of iron pyrites. The coal, after it has been mined, is wound up the shaft in small wagons or tubs, one or more of which may be contained in the cage in which it is hoisted. These tubs vary greatly in size, and may carry anything from 5 cwt. up to two tons of coal. When the cage has arrived at the shaft top, or bank, the tubs are run out of the cage, this, in modern installations, being often done by mechanical appliances. They then run over the weighing machine where their weight is recorded, and then they pass on to the tippler, which is an automatic appliance for turning the tub over and emptying the coals out gently and evenly. It is at that point I think we may fairly take the preparation for the market to begin. I include everything up to this point as part of the mining operations. The first step the coal has to undergo is screening or separating into suitable sizes. All coal above a certain size—which may be anything from 2 to 4 ins. according to the market requirements and the local custom—is looked upon as lump coal or round coal, whilst the undersize from the screen is generally spoken of as small coal, and each group is treated independently. Occasionally there is a screen which makes merely two sizes; sometimes a screen which makes a number of different sizes, which are spoken of as "cobbles," "nuts," "peas," "duff," and so on, both the names and the exact dimensions varying widely in different parts of the country.

In some parts of the country the bad practice prevails of only taking the large coal out of the pit. This is very often done by loading underground with a

*From the Journal of the Society of Chemical Industry.

fork, the times of which may be anything from 1 in. to 1½ ins. apart, anything that goes through these being of course left in the pit and wasted. From the point of view of heat generation, the coal that is left in the pit is quite as good as the coal that is taken out, and there is very little to be said in defense of such waste of a valuable national asset.

The screens used are of various kinds; the most common is the ordinary jigging screen (Figs. 1 and

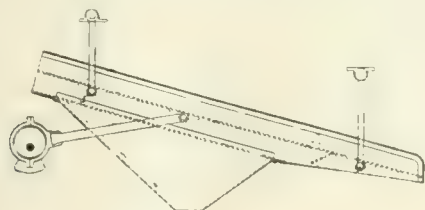


Fig. 1—Type of Jigging Screen.

2). The figures represent two of the best types of ordinary jigging screen, which merely consists of a shallow iron trough, the bottom of which forms the screening surface, suspended from girders by a number of radius rods, and worked by an ordinary eccentric, the revolution of the eccentric swinging the screen to and fro. Care must be taken that the eccentric acts upon the screen at a suitable point near its center of gravity, otherwise a very objectionable amount of jerk is apt to be developed. The screening surface may consist of iron plates with holes—round holes, diamond shaped, or square, according to local practice—or it may consist of woven wire, these being the two methods in most general use. This is a cheap

screen, fairly efficient, there is no excessive amount of wear and tear about it, but it requires a considerable amount of room at the heapstead; it makes a great deal of noise, and communicates a large amount of vibration to the structure in which it is placed, in addition to which it is also rough on the coal; it does very

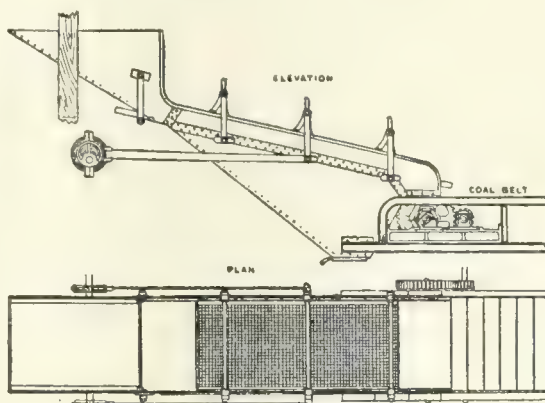


Fig. 2—Type of Jigging Screen.

well for most of the hard, strong coals of this country, but for some of the more tender coals, especially like some of those met with, for instance, in the north of France, it is much too rough, and other methods of screening have to be used.

A form of screen which has been used to some extent in this country is the "Vibromotor," in which the jigging motion is produced not by an eccentric, but by having a vertical shaft which swings from a joint, and to it a weight is attached eccentrically, so that the

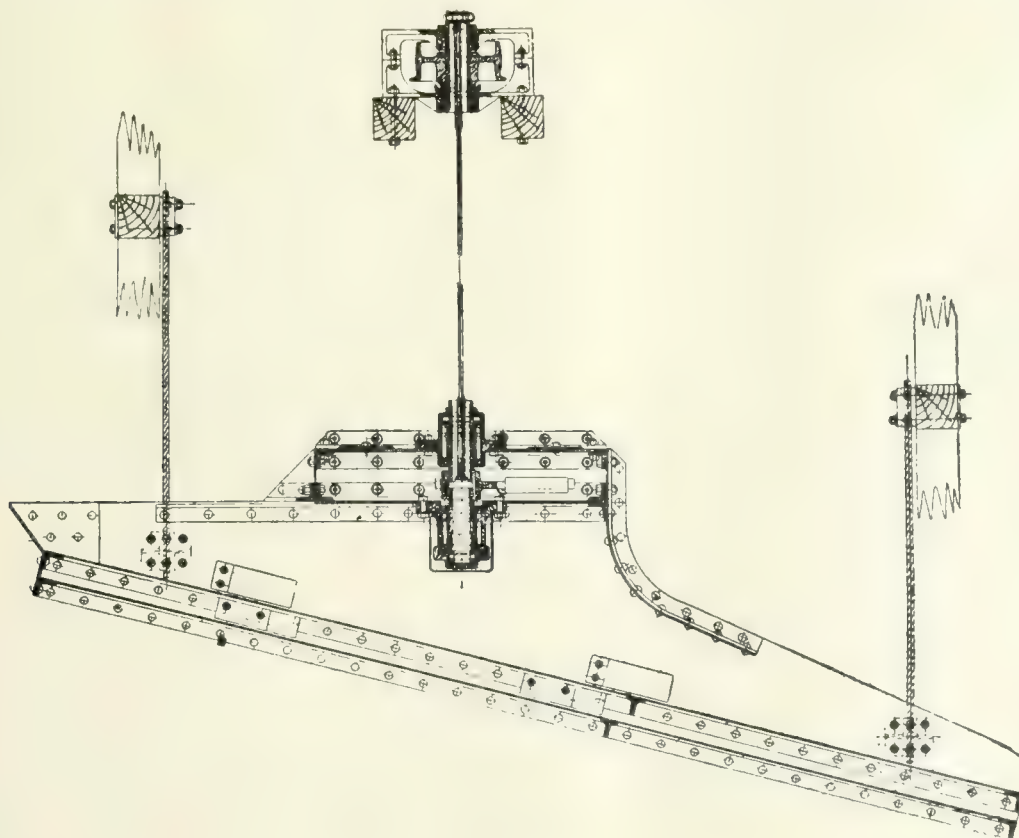


Fig. 3—Sectional Elevation of Vibromotor Screen.

movement of the weight swings the screen. It is really an appliance which makes use of the fact of the machinery being out of balance. The objectionable results of a wheel running out of balance are well

the independent motion of those bars moves the coal along it very smoothly and very gently.

Fig. 5 shows the "Chambers" screen. Here again are a number of bars supported upon pins; alternate

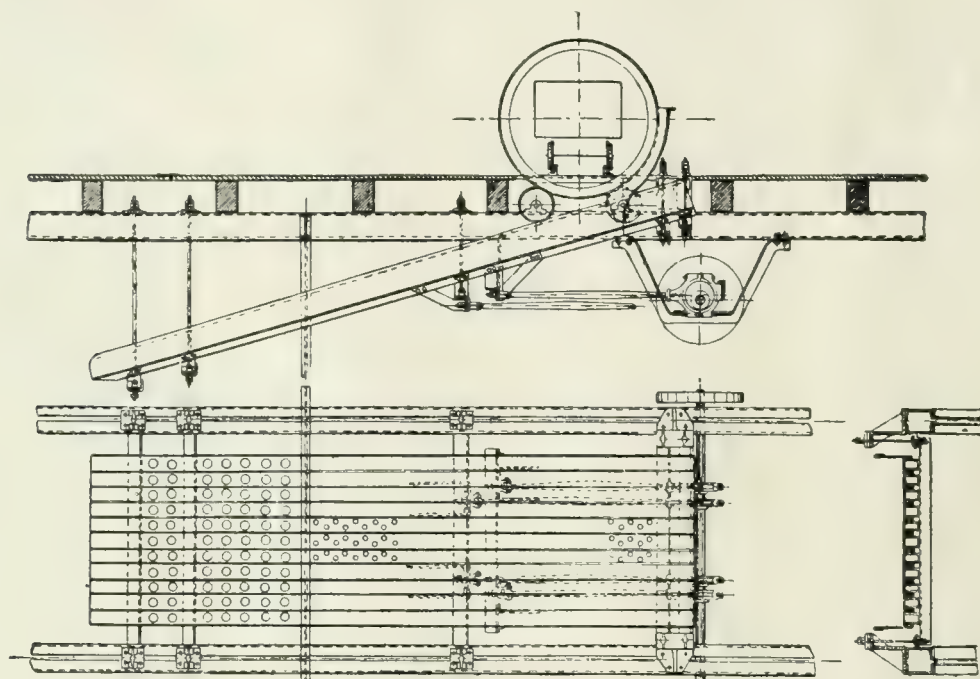


Fig. 4—German Form of Briart Screen.

known; in this case the want of balance is utilized, and the injurious effects are absorbed in doing useful work. The result is that the "Vibromotor" runs very smoothly and very easily, and communicates very little jar even to rather a light structure, so that in cases where a screen has to be placed high up in the heapstead it has great advantages. It has not come into very general use in this country, probably for commercial reasons, but it is certainly a very satisfactory type of screen. One form is shown in Fig. 3.

In order to get over the difficulty of dealing with tender coals, a number of screens have been devised in which the screen frame is fixed, but the bars of the screen have independent motions and thus cause the coal to travel over it. There are many of them. Fig. 4 is a modern German form of the Briart screen made by the Maschinenbauanstalt Humboldt; originally used in Belgium, it has come into extensive use in Belgium, Westphalia, and France. It consists of a fixed frame of iron plates, in which rest a number of bars, which are simply channel irons with perforations in the upper plate of the channels, thus producing the screening action. They are hung by radius rods at their bottom ends, and at the top alternate bars are moved alternately by a pair of eccentrics, so that one of the bars is always moving upwards and forwards while the neighboring bar is moving backwards and downwards. The result is that the coal resting on the bars which are moving upwards and forwards, is gradually moved down the screen by that action, so that, although the screen is fixed,

bars are carried on alternate pins, and are attached to a rocking lever, actuated by one or other of two eccentrics. The net result is that alternately a bar is lifted and swung forward whilst the neighboring bar drops and moves backwards. As in the case of the Briart screen, the coal resting on these bars is slowly moved along them. This screen has the advantage over the Briart screen that the bar is always horizontal at all times of its travel, and thus there is very little jar or jerk indeed, so that it works very smoothly.

Fig. 6 shows the Borgmann and Emde screen, simple in construction, and made up of longitudinal and transverse bars, the longitudinal bars being made of flat iron bars on edge, with semi-circular notches in the upper surface. The transverse bars which run

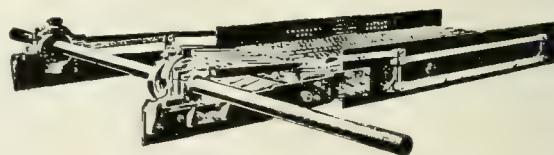


Fig. 5—Chambers Screen.

across them are round iron bars or tubes, and they are all driven by a simple mechanical contrivance, so that the pieces of coal resting on these transverse bars, which really form rollers, are caused to travel along by the revolution of these bars. It works smoothly and quietly, perhaps a little too quietly, and small coal riding on top of a big flat piece does not get sufficient screening.

A multiple screen for making a number of different sizes, the "Coxe screen," is intended for anthracite, which accounts for the large number of different sizes that it makes. The nest of screens is supported upon

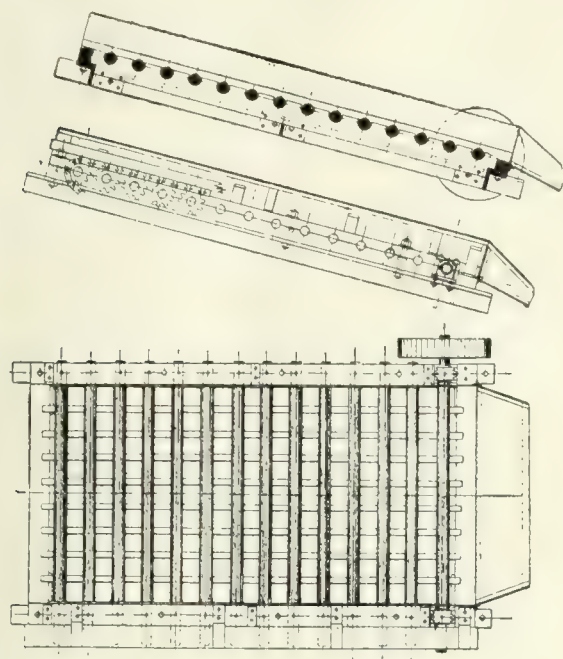


Fig. 6—Borgmann and Emde Screen.

four double cones and is driven by a vertical shaft which carries an eccentric; the result is that the whole nest gets a motion of gyration about an axis, so that when the coal is charged on at one end it moves over the surface of the screen in a series of circles. Every portion of that screen is in fact revolving in a circle, no two of which are concentric, and a very efficient screening action results. The screen is self-contained

and does not jar its foundations. I have seen a screen running, resting on two balks of timber without being bolted down, but of course this is not to be recommended.

Occasionally it happens that a colliery which screens its coal is asked for through-an-through coal, or un-screened coal. That is simply done by rendering the screens "dumb," or "plating" the screens—replacing the perforated plates by plain plates, or laying plain iron plates over the perforated ones.

The products of the main screening are thus round coal and small coal, and entirely different methods of treatment are required in dealing further with each of these. The treatment of round coal is exceedingly simple: The lump coal, the over-size of the first or main screen, is delivered on to a picking belt, this being the appliance which is most generally used in this country. It is made up of a series of iron plates bolted to two or three longitudinal chains driven by a suitable motor, generally by means of a sprocket wheel, the plates themselves resting on a series of rollers. Such a belt will be from 50 to 100 ft. long, and it will travel along at a speed of about 50 ft. a minute. Lads standing on either side of the belt pick out the dirt, and throw it onto a traveling belt to take it away or into tubs by means of which it can be removed. The pickers usually stand about 10 ft. apart, and a belt of this kind will deal with from 20 to 50 tons of ordinary coal per hour. Instead of a picking belt, a picking table is sometimes used, which consists of an iron table carried on a vertical spindle, or on rollers, revolving at about the same rate as the picking belt would travel, the pickers standing round the outside. In some forms they can stand round both outside and inside. The advantage of the picking table is that when only one is required, it requires

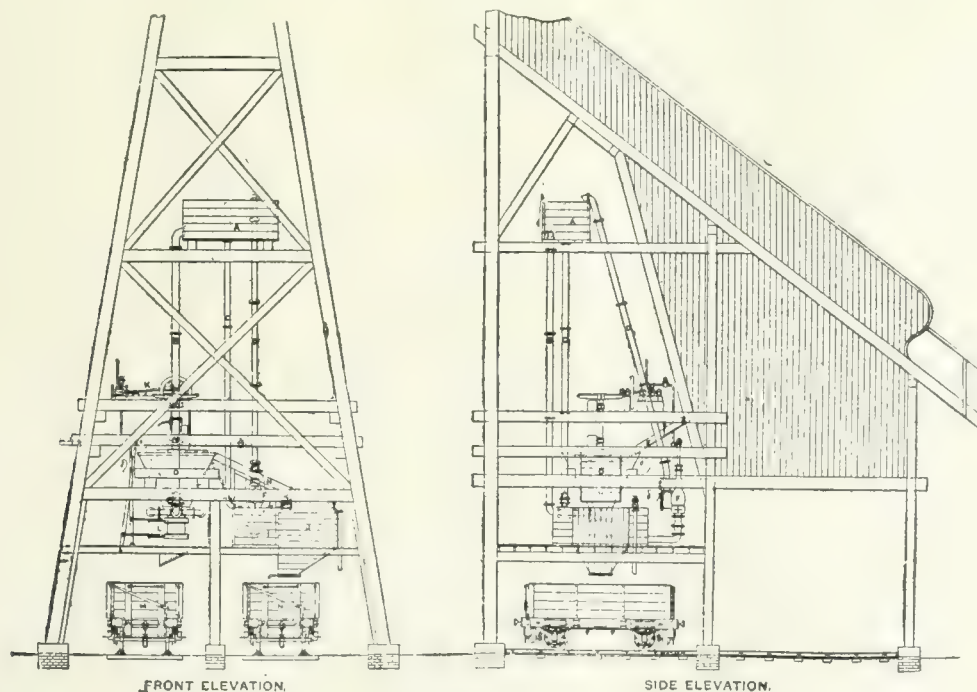


Fig. 7—Robinson Coal Washer.

very much less space than a corresponding belt would do.

Small coal is treated in various ways. Cobbles, say down to $1\frac{1}{2}$ ins. or sometimes 2 ins., are sometimes screened out in a second screen below the top one, and run on to another belt where they can be picked, but picking of anything under $1\frac{1}{2}$ ins. is practically impossible, and everything under that size has to be treated by the methods which I shall describe somewhat more fully.

Mechanical means have to be resorted to for cleaning this small coal, this being usually spoken of as the process of coal-washing. In practice the difference in the specific gravities of coal and dirt is utilized for their separation. The specific gravity of coal varies from about 1.25 up to about 1.6 for anthracite. Ordinary bituminous coal averages about 1.3. Dirt (namely shale, fireclay, sandstone, and so forth) will probably range from 2 to 2.7 and may be averaged at about 2.5. Iron pyrites ranges from 4.8 to 5.2 and averages about 5 in specific gravity, so that any methods that will separate the ordinary dirt from coal will still more easily separate the pyrites from it, and for that reason I shall only discuss here the methods for removing ordinary dirt.

A very simple laboratory test can be used for showing whether the coal is washable, and to what extent, by crushing the coal and throwing it into a solution of specific gravity intermediate between that of the clean coal and of the ash to be separated from the coal. I found recently that a solution of zinc iodide in water is very useful and convenient for this purpose; it can be obtained up to 1.9 specific gravity, and solutions ranging from 1.4 to 1.7 are very convenient for this method of separation. The ground mass is stirred in the solution, using a separating funnel, when the coal will float up and the dirt sink to the bottom, when each can be drawn off, washed, and examined. Such a method is, of course, impossible in practice on account of the expense and the difficulty of working with heavy solutions, so the power of gravity is utilized by setting either the fluid or the coal in motion and using the momentum set up in the moving particles to effect the separation. The nature of this action is easily intelligible: If a particle is let fall in vacuum, it goes on falling with a uniform accelerated motion, there being no resistance to retard its fall, but if it falls in a resisting medium, it has work to do in displacing the particles of the fluid and in overcoming its own friction against those particles. This work becomes greater the faster it is falling, and, as it increases, it must attain such a velocity that any further increase would so greatly increase the resistance that it cannot fall faster. In other words there is a limiting velocity to which all particles are subject when falling in a resisting medium, so that a particle falling in a resisting medium falls at first with gradually increasing velocity until it reaches this practically constant uniform velocity and continues to fall at that,

and this speed of falling is known as the "ultimate falling velocity" of the body.

For spheres it can be proved that a simple formula holds good. If two spheres having diameters D and D' and specific gravities S and S' respectively fall in a medium of specific gravity F , the relation holds that the ultimate falling velocity of these spheres will be the same when

$$D \quad S' - F.$$

— = ———. When the fluid is water, F is practically

$$D' \quad S - F$$

$$D \quad S' - 1$$

equal to 1, and then — = ———, if the particles are to

$$D' \quad S - 1$$

fall with the same ultimate falling velocity. When the particles are falling in air, F is practically zero, and

$$D \quad S'$$

then — = — for ultimate equal falling velocities. Com-

$$D' \quad S$$

paring coal and shale, and taking the specific gravity of

$$D$$

shale as 2.5 and of coal as 1.3, the value of — is 5 for

$$D'$$

spheres falling in water; that is to say, a sphere of coal will fall in water at the same ultimate falling velocity as a sphere of shale if the diameter of the sphere of coal is 5 times that of the sphere of shale, and in the same way when falling in air, if the diameter of the sphere of coal is twice that of the sphere of shale they will fall with the same ultimate falling velocity. This result helps very much in designing coal washing machinery. At the same time it is not strictly true, except for the particular case when particles are falling in very large masses of fluid; when, as is usually the case, there are a very large number of particles crowded together, taking up a considerable portion of the fluid, the conditions are usually spoken of as the conditions of hindered falling; the particles get into each other's way and jostle each other, and under the ordinary conditions of working, particles of coal and shale can be separated when the differences between their diameters are considerably greater than given by these equations, so that it is not necessary to screen as closely as these formulæ would indicate. If it is required to separate all the coal from all the shale in a given mass, which has been screened through two screens, of which the mesh of the larger is five times the diameter of the mesh of the smaller, it is obvious that the largest possible particle of coal that can be in the product must fall more slowly than the smallest possible particle of shale that can be in it, and therefore separation is possible. It is by working on such formulas as these, that it is possible to determine the ratio that the successive meshes of screens must have to each other, in order to enable us to separate the particles. Nevertheless, these relations are only true when the particles are of certain definite sizes; when the particles dealt with are so small that another factor comes into play, namely,

the internal friction of the particles of the fluid or the viscosity of the fluid, which may be neglected when dealing with particles of any considerable size, then these equations no longer hold; an entirely different set of laws come into play and such very fine particles are no longer capable of being treated by ordinary washing machinery.

The principal of separation by equal falling can be applied in several ways; one of them is to allow fragments of the coal that have to be washed to be carried

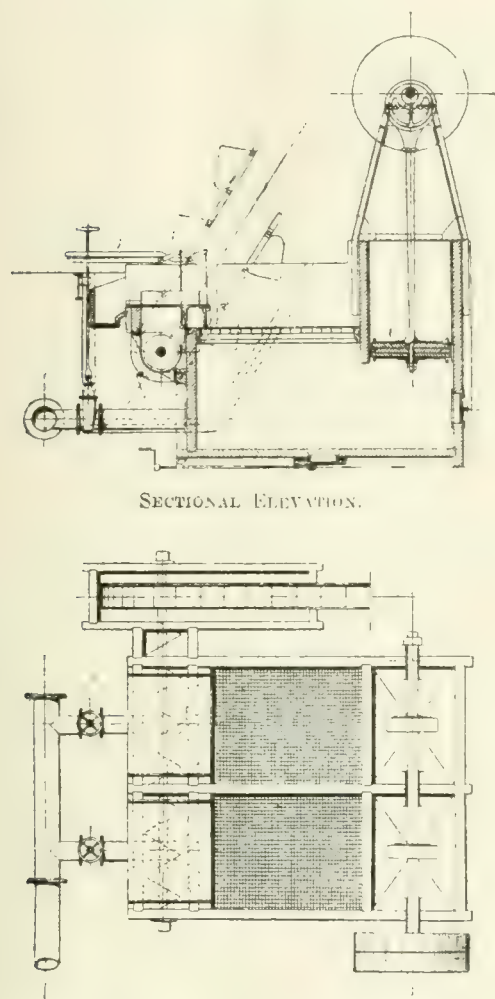


Fig. 8—German Type of Nut Washer.

along in a stream of water, when, obviously, the slower falling ones will be carried further and those that fall more rapidly will fall nearer the point at which the coal enters the water; this is the simplest of all the principles involved. As the corollary of the above equations, if a particle is placed in an ascending stream of water, it is found that if the velocity of the ascending stream is greater than the ultimate falling velocity of that particle, the particle will be carried upwards, but if the velocity of the ascending stream is less than the ultimate falling velocity of the particle, the particle will fall. Obviously, if a graded mixture of dirt and coal is introduced into a stream ascending with suitable velocity, the coal can be made to ascend whilst the shale falls, and that is an application of the prin-

ciple of simple equal falling, which I shall speak of as the first principle of coal washing.

If the particles are subjected to a descending instead of an ascending stream, all the particles will move downwards, but the descending water current will accentuate the falling velocity of the faster falling ones, so that if a set of particles is submitted to alternately ascending and descending water currents, the action of these reinforces each other and a sharp separation is obtained between the faster falling and the slower falling ones. Under these conditions separation takes place a good deal more easily than the equal-falling formula would indicate. This forms what I call the second principle of coal washing, *i. e.*, the principle of submitting dirty coal to a mixture of alternate ascending and descending currents. It is the principle which is perhaps most widely used in coal washing, and forms the principle of the appliances known as jigs or bashes. Yet another set of conditions obtain, when another form of appliance is used, which we meet with again quite frequently. If a stream, carrying particles of coal and particles of dirt mixed together, is allowed to flow down an inclined plane, it is obvious that the larger particles project further from the plane and are therefore exposed to faster water currents, whilst the smaller particles lying immediately upon the surface of the plane are exposed to the slower water currents; therefore a larger particle would be carried down the plane more rapidly than a smaller particle. With particles of the same size, the lighter particle would be moved the faster because it affords less resistance to motion along the plane. Therefore, under these conditions, even a roughly screened mixture of particles of coal and dirt can be washed, and it is possible to so adjust matters that the coal will be washed down the plane while the force of the stream of water will leave the particles of shale upon it; or can be arranged so that both shall be carried down, one more rapidly and the others more slowly. This may be called the third principle of coal washing.

Finally, yet another principle consists in submitting fragments of dirty coal (mixed coal and dirt), suspended in water, to a series of jerks or shocks whilst under the influence of a stream of water. This jerking motion separates and frees the particles from each other, and helps the stream of water to act in the way already indicated, and at the same time it imparts differential momentum to the particles and thus causes them to move at different rates over the surface of this jerking plane. In fact it is possible to cause the heavy particles to move in an opposite direction to the stream of water whilst the light particles are carried along by the stream. This I will call the fourth principle of coal washing.

I call the first principle that of "Differential rates of falling in a moving body of water"; the second principle "Exposure to alternately ascending and descending water currents"; the third principle "Streaming

particles over an inclined plane," and the fourth principle "Exposing particles to impulses whilst they are also subjected to a water current."

By the application of any of these principles we can separate a mixture of particles in accordance with their specific gravities, provided that the disparity between the sizes of the largest and smallest particles present be not excessive, in other words provided the particles have been suitably sized by screening. To illustrate these principles, I propose to quote an example or two of well-known coal washing appliances belonging to each class.

A good example of the first type is a machine very well known in the North, but not so much used as it was, known as the Robinson Coal Washer (Fig. 7). It consists of an inverted cone in which a number of arms are revolving; the coal is fed in through a shoot at one side; in the lower part of the cone are a number of perforations connected to a water pipe; the water ascends and carries the lighter coal upwards whilst the heavier particles of dirt fall and are collected in the bottom of the cone, and by that means a separation is easily attained. It is usually applied to coal screened to pass about $\frac{3}{4}$ -in. mesh or less. The capacity of such a machine is from 15 to 30 tons per hour, and one man and a couple of lads can attend to one of these machines. One of these washers in use at South Derwent Colliery could treat 100 tons of small coal a day and reduce the ash from 11 per cent. to 5 per cent., at the same time also largely reducing the sulphur.

The illustration (Fig. 7) shows the Robinson washer complete. The coal is contained in a bin and passes from a shoot into the washer. The clean coal is carried round in the washer and issues over a screen, to get rid of the water. The washed coal drops into a little hopper and is discharged into the railway wagon below it. The water draining off the coal runs into a tank, whence it is pumped up by a pulsometer into a tank from which a pipe goes down to the washer to supply the annulus from which the perforations open into the body of the machine. The dirt drops down into a box which can be closed by a top or bottom slide at will. In ordinary running the bottom slide is shut and the top slide open. The dirt collects here, and from time to time the top is momentarily closed and the bottom opened, and the dirt drops out into a wagon, when the previous arrangement is restored and so the work goes on.

The second principle, the principle of the jig or "bash," has been in use for a considerable time. I believe its first application was made by Bérard, in 1850, and ever since that time it has been used to a greater or less extent for coal washing. In all cases the coal rests upon a sieve through which the water currents pass. The sieve can either swing up and down in hutch of water, or it may be fixed and the water caused to pulsate through it by the motion of a piston. Both methods are applied, and, as far as

results go, there is nothing to choose between them, but in most instances the fixed sieve arrangement is more convenient, and the majority of machines are built in that form. There is a very important difference between the machines designed for washing very small coal and those for washing nut coal. In wash-

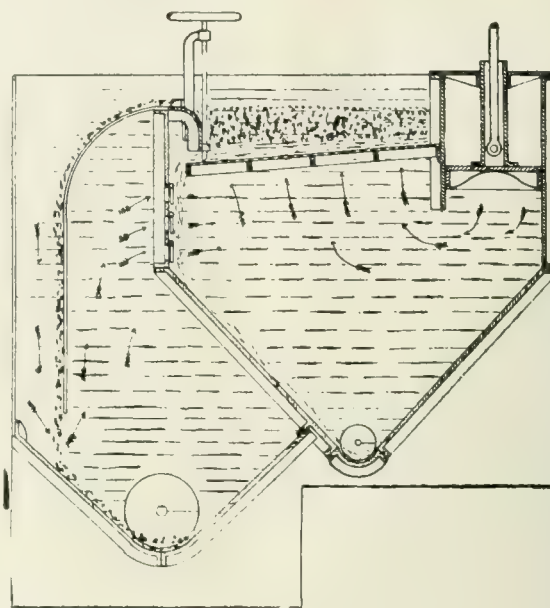


Fig. 9—Nut Washer Built by English Firm.

ing nut coal the mesh of the screen is always decidedly less than the size of the nuts to be washed. The nuts therefore rest upon the screen; they are fed in at one side and travel across the sieve under the influence of a stream of water; in the course of their travel they are subjected to these alternating water currents, and are thus separated into a lower layer of dirt close to the sieve, and an upper layer of clean coal above it; at the discharge end the coal is discharged at a higher level and the dirt at a lower level

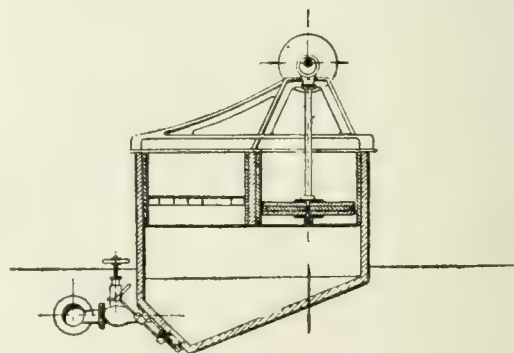


Fig. 10—Single Feldspar Washer.

immediately over the screen, by means of adjustable gates.

For washing fine coal, on the contrary, the mesh of the screen is larger than the largest particles to be washed, and, in order to prevent both coal and dirt passing through the screen, a bed of some material

has to be placed upon the screen. In practice a bed of felspar is generally used. Clean Norwegian felspar cleaves into fragments of rhombohedral shape, and when these rest on a screen and are subjected to the action of water currents they tend to revolve about an edge and thus form a sort of valve which helps in

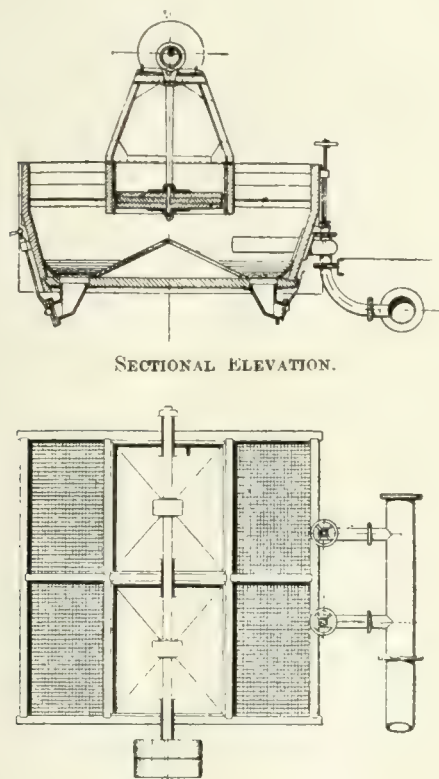


Fig. 11—Double Feldspar Washer.

the separation that I have been describing. At the same time, Norwegian felspar is a material of convenient specific gravity; it is fairly hard, and will stand tolerably well in the wearing action on the screen, and is, therefore, an all-round convenient substance. The fragments of felspar used are usually $\frac{1}{2}$ in. to $1\frac{1}{2}$ in. in size and their area usually about 50 per cent. greater than the mesh on which they rest. In the course of use the edges get rounded, and the pieces are then no longer fully efficient. When that happens, the pieces have to be broken down to the next smaller size and used on a smaller jig, and so on until they are rendered too small to be any longer of use. The average life of such a bed of felspar is six weeks.

Fig. 8 shows a German type of nut washer, which consists of a hutch, in a side compartment of which a piston moves up and down. The coal to be washed is fed in through a hopper and drops upon a screen. By the motion of the piston, water is caused to pulsate through the screen and the coal as it travels along from the hopper towards the discharge, is divided into two layers; the lower layer is discharged through the adjustable lower gate, the height of which can be regulated by a little handle. The washed coal forming the upper layer is discharged at an adjustable upper gate and drops onto a little jiggling screen which

is to drain it. The water drains back into a trough and is brought back to the machine again. The dirt washed out through the lower gates drops into a trough, and is carried away by an elevator. This forms a very neat and complete form of washer.

Fig. 9 shows a nut washer built by an English firm, Messrs. Sheppard and Sons, the principle of which is very much the same. At one side is the piston moving up and down in a separate compartment. The coal is washed over a gate into a tank whence it is carried away by a screw conveyor; the dirt drops out through an adjustable lower gate into a hutch where there is another screw conveyor for removing it, and by means of valves the water that comes over with the coal is brought back so that the amount of additional water required is very small. This machine is intended for washing nut and pea coal for sizes ranging between $1\frac{1}{2}$ in. and $\frac{3}{8}$ in., with a piston making 60 to 70 strokes from 5 to 7 ins. per minute. It will wash these coals at the rate of about 5 tons per hour.

Fig. 10 is a single felspar washer for fine coal, built by a well-known German firm. A bed of felspar is laid on the screen, and coal fed onto it. The dirt passes through the bed and the screen and drops into a hutch from which it is removed from time to time by a gate at the bottom, and the fine coal passes over the top and is conveyed away as required after being washed. This is the form used in small washeries. For large washeries double felspar washers of the same type are used (Fig. 11), the principle being ex-

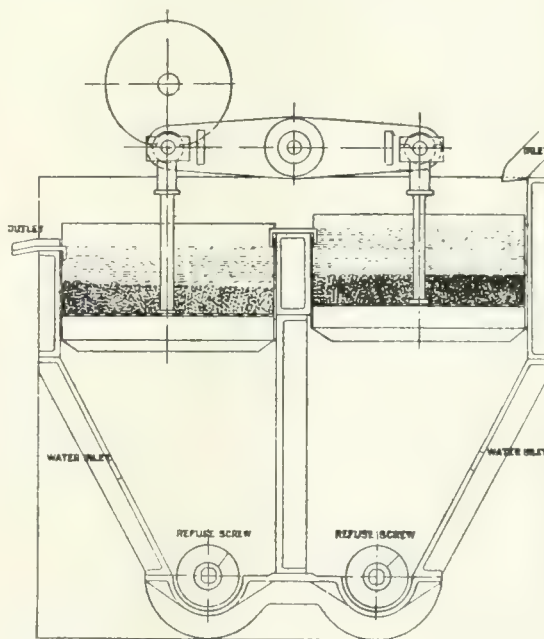


Fig. 12—Feldspar Washer Showing Screen Jigs.

actly the same, only there are two washers, with a large piston working between each pair of washers, thus saving space.

A more usual arrangement, instead of having sliding gates, is for the dirt to drop into a hutch from

which it is removed by a screw conveyor or a scraper conveyor, and the clean coal from all the washers is usually discharged into a trough along which it is carried by a stream of water or by a conveyor into a storage hopper. In order to illustrate the moving screen jigs, Fig. 12 shows a felspar washer, built by Messrs. Sheppard and Son, in which there are two screens suspended from a rocking lever moving up and down in a hutch. These are intended for successively washing the same coal, when it is required to get a very clean article. The coal enters at one end, is partially washed on the first screen, the partly washed coal passing over a gate onto the next screen, and the dirt dropping through into the hutch. The washing is finished on the second screen, the clean coal is discharged at an overflow, and the dirt drops down into the hutch, whence screw conveyors take it away. This machine is intended for washing coal less than $\frac{3}{4}$ in. in size. It works at about 180 quarter-inch

When coal and dirt flow down this trough, the heavier dirt settles down and collects against these dams, whilst the lighter coal is washed over the dams and discharged at the bottom end. The troughs are worked in pairs, and as soon as the dams in one trough are filled up with dirt, the neighboring trough is brought into play, the dams in the first trough are merely lifted up and the dirt scraped out or washed out from the trough, then the dams put back again and the trough is ready for use again. In order to make the trough continuous acting, when, of course, the troughs need not be in pairs, a number of devices have been employed. One of the simplest is the machine known as the "Elliott" washer (Fig. 13); this consists of a single trough set at an inclination of about 1 in 15; instead of there being fixed dams, there are a number of dams or scrapers, as they may be called—reaching down just to the bottom of the trough, and attached to two endless chains, so arranged that the

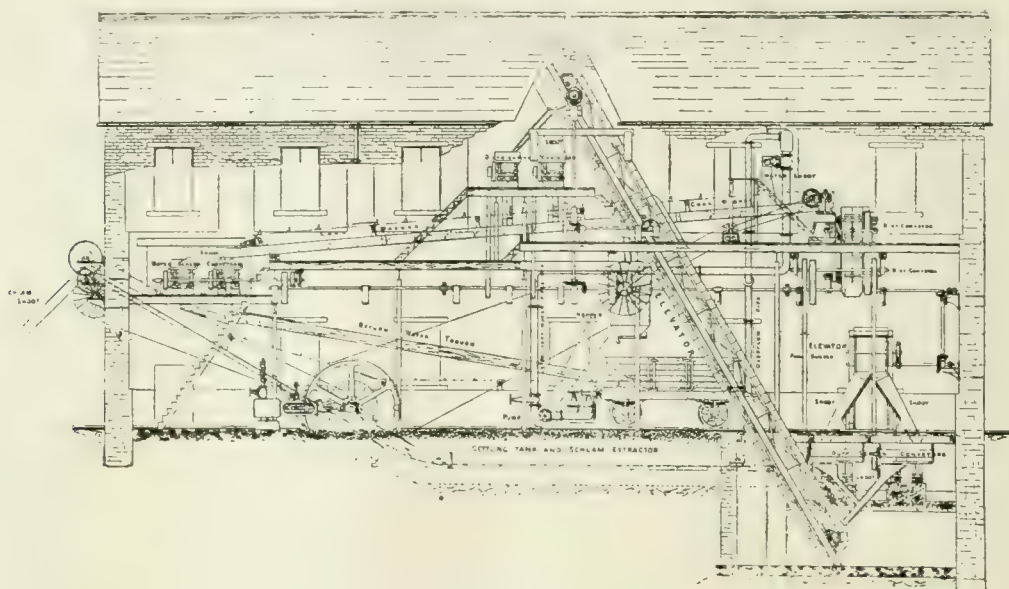


Fig. 13—Elliott Washer.

strokes per minute with a felspar bed 4 in. to 6 in. in depth, and will deal with something like $2\frac{1}{2}$ or 3 tons per hour. The Baum jig may be built either as a nut or fine coal jig, the difference of this make being the way in which the water pulsations are obtained. Instead of having a moving piston to actuate the water, a stream of compressed air is allowed to act upon the surface of the water, being regulated by means of a little valve, so that the air acts in alternate puffs on the surface of the water. Baum, in applying his washers, does not use a felspar bed, and claims to get practically the same result by using a rather deep layer of rough dirt on the surface of the screen.

The best illustration of the third principle is afforded by the "Trough washer," of which there are a number of different types. A simple example of a trough washer consists of an inclined trough across which there are a number of dams or riffles placed.

set of dams lying in the troughs are moving upwards. The coal is fed in about half way down the trough, whilst clean water is allowed to run in at the top of the trough, which helps to wash the coal over the tops of the scrapers; it is discharged at the lower end and runs over a screen to get rid of the entangled water and drops into hoppers, whilst the heavier dirt, settling in the bottom of the trough, is dragged up gradually by the scrapers as they travel slowly upwards, and is discharged at the upper end of the trough into a wagon, this being exactly the same principle as that of the plain trough washer, except that the Elliott washer is continuous acting and automatic. The scrapers are usually 3 in. or 4 in. deep and set about 6 ft. apart, and such a washer will deal with anything from 5 to 20 tons per hour. The illustration is from a colliery in Cheshire, where a washer is working to tons an hour. The dirty coal, *i. e.*, the

unwashed slack, contains 25 per cent. of ash, the washed coal 4.2 per cent., and the dirt discharged as much as 68.7 per cent. of ash.

These are the principal patterns of washing machines in use. There are one or two, but not many, on the shaking table principle, which I have called the four principle. One of these is the Craig table, which was tried for a while in the North here, and met with a fair measure of success, but never seems to have come into general use. It is a Y-shaped table with narrow discharge shoots at the bottom and broad shoots at the top, the whole being set on a slight incline; a two-armed cam drives the table sharply downwards against a bumping block, giving a sharp bump, whilst a strong spiral spring pushes it up the incline again to be met by the next stroke of the cam. The coal is fed on about the center of the table where the

Coal is fed in at one end, and, under the action of these jerks, it is caused to travel along and is discharged at the upper end onto a screen which carries it to a coal conveyor by which it is carried off to the required coal discharging point. The fine dirt settles under the action of the jerks in the lower part of the screen and is discharged there by means of a gate at the side which is kept closed by a spring or counterpoise until a sufficient weight of dirt has accumulated to overcome that counterpoise, when the gate opens and the dirt is discharged, and the washing goes on, so that the appliance works continuously. It is said that the capacity of one of these washers is about 15 tons per hour.

The above are examples of individual appliances, but the best results can only be obtained when a certain number of machines is combined into a complete

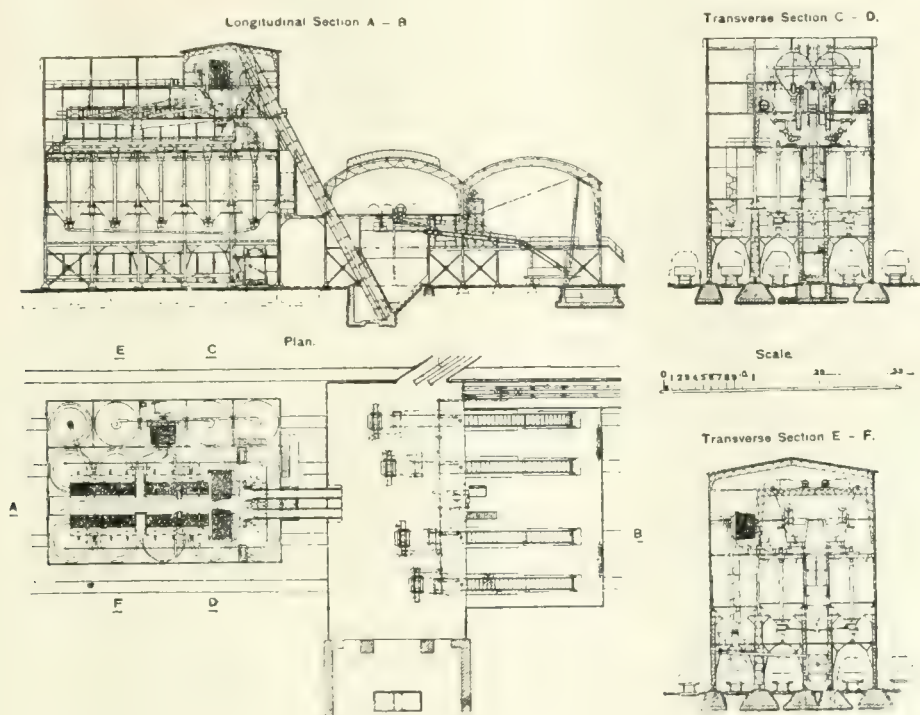


Fig. 14—Washery as Erected at German Colliery.

two arms of the Y meet, and the coal, as it rests on the table, receives a series of bumps as described, thus jerking the dirt towards the upper ends of the Y-shaped table, whilst the lighter coal flows down to the lower end and is discharged from the small spouts. It was tried at Coanwood Colliery, where it washed 8 tons per hour and reduced the ash from 11.5 to 4.8 per cent, but apparently it was found rather complicated and does not seem to have come into extensive use. A machine on a somewhat similar principle is quite a new machine, the Greaves Washer. So far as I know the first washer on that principle in the North is just now being erected in southeast Durham, but has not yet been tried in this district. It consists essentially of a shaking tray or screen. This screen is so suspended as to be able to swing, and is caused to swing to and fro under the action of a cam.

installation. In a complete washery all the small coal—anything, say, under $1\frac{1}{2}$ in.—is usually brought to the washery. It is then usually lifted directly into a screening appliance, the usual form of screen in these washeries being the revolving cylindrical screen or trommel, a number of which arranged successively being often used for screening the coal into a number of different sizes. Where space is an object, a spiral or concentric trommel may be used. Of course, all these screens have to be enclosed in iron housings to prevent the dust flying about. By these means a number of separate sizes are made. Usually the coal, as it comes from the colliery, includes everything under $1\frac{1}{2}$ in. or 2 in.; it is sized into various sizes, usually from $1\frac{1}{2}$ in. down to $\frac{3}{4}$ in. for nut coal, $\frac{3}{4}$ in. down to about $\frac{1}{4}$ in. or $\frac{1}{2}$ in. for bean, and peas down to about $\frac{3}{8}$ in. These three sizes are mostly washed on

nut washers, each size going to its separate nut washer; everything less than $\frac{3}{8}$ in. goes to the felspar washers for washing; of this duff usually two or three different sizes are made, each going to its individual felspar washer. The general arrangement then is that the cleaned coal is delivered into hoppers. Sometimes if required for coking, all the sizes may be mixed; if required for market, the different sizes are often kept separate. Often the dirt from the nut washers still contains a large proportion of coaly matter, and in a complete washery it is usually found worth while to take the dirt from the coarser washers to a crusher to be crushed fine, and wash the resulting fine stuff in felspar washers, a separate set of washers being usually dedicated to this purpose. There are several such washeries in this country with capacities ranging from 50 to 200 tons per hour.

For the sizing of the finer small coal, which is rather difficult to do by screens on account of their aptitude to clog, a set of V boxes is often used. The coal is run in at one end of a set of inverted pyramids or

tank at the bottom of the building, where a scraper moves it along into the boot of an elevator which lifts it in turn to the same hopper as the other clean coal. The capacity of such an entire plant, including the picking belts, is about 1500 tons of coal per day of 10 hours. Such a complete washery, as erected at a German colliery, is shown in Fig. 14. A complete picking belt heap-stead, without washery, built by the Maschinenbauanstalt Humboldt, is shown in Fig. 15.

In any system of coal washing, it is often found that the finest coal will form slimes which will only settle in water with great difficulty, and which are thus very often lost. Now it has been found in many cases, especially in dealing with coals for coking purposes, that this extremely fine coal is the most valuable part as regards its coking qualities, and its presence or absence often makes all the difference in getting good or bad coke. In Germany this difficulty has been got over by the very ingenious method of blowing out the very fine coal first before it gets to the water and has a chance of making slimes, and treating it dry.

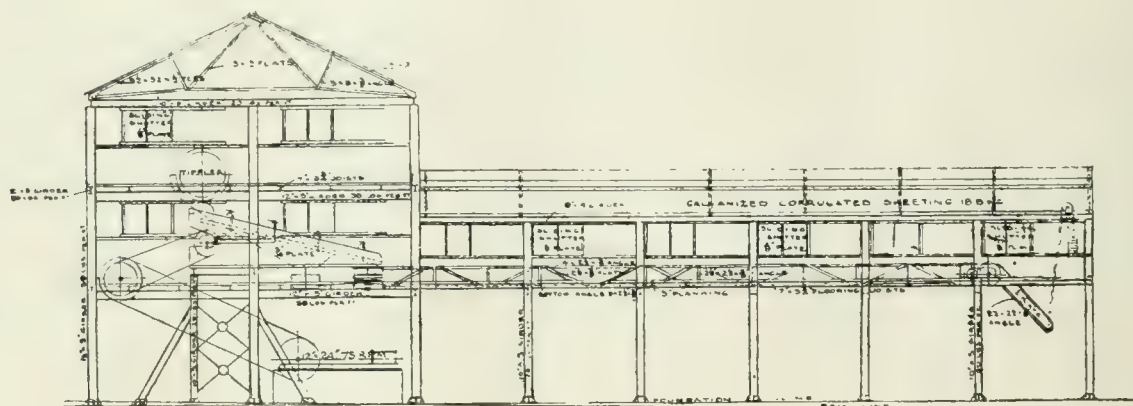


Fig. 15—Elevation of Coal Cleaning Plant Capable of Handling 500 Tons Per Day.

cones, and in each of these cones there is an upward stream of water. In accordance with what I have called the first principle of coal washing, it is evident that the coarser stuff will settle in the first cone, the next fine in the next, and so on up to the finest. The spigots from these different cones are then used to supply the individual felspar washers, this being perhaps the best method for sizing the finest sizes of coal.

In a complete washery the coal comes from the mine, and is tipped in tipplers on to several jiggling screens. The over-size or the coarse coal, drops onto the picking belts, the fine coal goes through underneath, and is carried by a transverse belt into storage hoppers. From these it is elevated to the washery proper, where it first passes through one of the spiral screens before mentioned. Here it is sized, the nut coals going to a set of nut coal washers, the finer material going to a set of pointed boxes, the spigot of each of which supplies its individual felspar washer. The finer coal is allowed to drop down into a large

A good example of a machine for that purpose is one which has been devised by Hochstrate. The coal is screened in a revolving screen, and the under-size from this is dry-blown before washing, the under-size being about $\frac{1}{4}$ -in. mesh. It drops from the screen housings, and in dropping down it is caught by a blast from a blower through a blast pipe, and the lighter stuff is blown up an inclined shoot, the heavier stuff being able to fall down direct; the lighter material when carried up strikes against an inclined plane, the heavier particles roll down, getting clear of the blast, and drop back into the shoot into which the heavier particles have already fallen. The lighter stuff is blown upwards into a big dust chamber. The stuff that drops through the shoot is met by a second stream of air which again blows out the finer stuff and carries it into the same dust chamber, whilst all the heavier particles drop down together through the shoot into a water trough which carries all the coal of about 0.3-in. mesh to the washeries where it is washed in the ordinary way.

THE WORK OF ENGINEERS IN THE GAS INDUSTRY.*

By FREDERICK P. ROYCE.†

The business of manufacturing and distributing gas is older than that of any other of the public utilities with the exception of that of distributing water.

Early in the nineteenth century companies were established and in active operation in England. Many of the companies still doing business in this country were organized and in operation prior to 1850. It was not until a much later date, however, that results due to the efforts of scientifically trained men were generally apparent.

The first dry distillation of coal producing gas for commercial purposes was accomplished in retorts made of iron placed in a horizontal position, the heat necessary being directly applied by furnaces placed underneath the retorts.

With the exception of the substitution in the retort of fire-brick material for cast iron, this type of retort with slight modification was used for many years and in some small plants is used today.

In the early days of the industry it was found that with little effort sufficient gas could be produced and sold at a high price for lighting purposes to furnish a satisfactory return on the capital invested.

At the time, little or no consideration had been given to the relations of the companies and the public, which are now properly regarded as of great importance.

It was not understood that to get the best results for all, these companies should be operated as regulated monopolies and the earnings of the companies were not sufficiently large to invite general competition.

These gas companies then furnished all illumination not produced by lamps or candles. Under such circumstances there was apparently little incentive to the engineer to enter the business and apply a scientific knowledge to the reduction of the cost of the plant or of operating it.

As the time went on, these conditions changed and in the early eighties a very strong competition was introduced through the development of the electric lighting companies which for a time threatened to destroy the business of the gas companies and in some places actually did reduce it materially.

It was then absolutely necessary that the owners of these properties should devise means of reducing their operating cost and to do a much larger business at a smaller rate of profit. This could only be done through the employment of trained engineers.

At that time coal gas was made almost exclusively and the largest element of cost in the production of the gas was in the carbonization of the coal. There has always been a saying to the effect that money was made or lost in the retort house and it has been

in the retort house that the greatest improvement has been effected.

As has been said, the retorts were then of the direct-fired type, the coal to be carbonized was placed in the retorts by hand and the coke was removed in the same way.

It was then considered good practice if 30,000 ft. of gas could be made in a day by each man employed in the retort house. This hand firing was slow, resulting in a serious waste of gas and loss of heat.

The first step toward improvement was the introduction of machinery to charge the retorts and remove the coke. Notwithstanding the extremely difficult conditions under which this apparatus must be operated, it has been on the whole very successful. Numerous types of machinery have been built applicable to the smallest or the largest works. At about the same time the regenerative setting of retort benches was designed and put in successful use.

In this generator is made a part of the bench setting and used to produce carbon monoxide gas through partial combustion, using a limited amount of primary air. This monoxide gas is distributed as desired through the bench and brought in contact with secondary air which has been preheated by passing through flues adjacent to other flues through which the products of combustion are carried to the chimney. The carbon monoxide being fired as it combines with the secondary air produces a high heat which can be satisfactorily controlled, assuring a uniform temperature throughout the various retorts of the setting.

This type of bench was followed by the inclined retort setting. In this bench the retorts are set at an angle of approximately 31° from the horizontal, the charge being introduced from the top and distributed through the retort by gravity, the coke falling from the bottom mouthpiece as desired.

The principal advantage of this type of setting is in the avoidance of the use of expensive charging and drawing machinery. The inclined setting was followed in a few years by the vertical type. It is interesting to note that although experiments were made with vertical retorts in the very early days of the gas industry, it was not until three or four years ago that engineers in Germany were able to build and operate them successfully on a commercial scale. The first successful installations of this sort were designed for intermittents charging and discharging. Little machinery was required to operate them and due to the fact that the retort was completely filled with coal, a marked improvement was noted in the character of the bi-products.

As the weight of the coal in the vertical retorts is supported principally by the lower mouthpiece rather than the retort itself, the mouthpiece being easily repaired as needed, there should be and apparently is an improvement in the life of the setting.

A more recent development in the vertical retort setting has been accomplished by the perfecting of

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charging and discharging apparatus, by means of which a continuous carbonization of the coal is effected.

A still further improvement in the carbonization of coal in large plants has been effected by the successful design and operation of the so-called coking chamber. This in reality is a development of the inclined retort, but instead of using a comparatively small retort which would contain a charge probably not to exceed 1000 lbs. in weight, a large chamber designed to contain and carbonize six or seven tons is used. The bottom of this chamber is built at an angle similar to that of the inclined retort so that the coke slides freely from it when the mouthpiece, which is really the entire lower end of the chamber, is removed.

Through these developments the cost of labor has been greatly reduced and whereas with the early benches there was a production of 30,000 cu. ft. per man per day, with the coking chambers a production of 150,000 cu. ft. per man has been made possible. At the same time the yield of gas from a pound of coal measured by the product of its volume and candle power has been increased 25 per cent.

The amount of fuel required to carbonize the coal, an important item of cost, has been very greatly reduced. The capacity of unit has been increased to such an extent that much less space is required to produce a given amount of gas, which results in a reduction of plant cost. The quantity of certain bi-products has been increased and the character of these bi-products much improved. This has been particularly noteworthy in connection with the vertical type of setting. The coke made in the vertical retorts and in the coking chambers is of much the same character as that made in the beehive oven. It is stronger and consequently a smaller percentage of it is in the comparatively worthless form of screenings or breeze. A larger percentage of ammonia is recovered. The tar produced is better and comparatively free from lamp black and other impurities.

It is regrettable that most of the improvements noted in retort house operation have been due to the efforts of foreign engineers. The charging and drawing machinery was first perfected in England. The inclined and vertical retort benches for intermittent carbonization were developed by German engineers and the continuous system of carbonization with vertical retorts is due principally to the efforts of Englishmen.

To American engineers, however, there may be fairly attributed the development of the water gas process. The Lowe type of apparatus developed in this country is now the standard throughout the world where water gas is made. It is estimated that more than one-half of the total production in this country is of water gas. This has been of great importance in connection with the production of coal gas.

The cost of the latter is directly dependent on the

price received for residuals and it is probable that if coal gas only were made in this country the increase supply of residuals would be in excess of the demand so that the average price received for them would be reduced and consequently the cost of producing the coal gas would be increased.

It is now common practice in America to make both water gas and coal gas in the same works. The water gas can be made of any desired candle power and mixed with the coal gas, making it possible to furnish a commercial product of almost any quality desired. The improvements in the art, however, have not been confined to the generating house.

The quality of the gas depends in great measure on the method used to clean and purify it. The importance of this may be seen when it is realized that the so-called impurities in the gas, which are harmful as a part of the gas, are themselves valuable bi-products when removed.

To obtain the best results the temperature of the gas must be made to pass through wide ranges during the different steps of cleansing and purification. Engineers have determined the correct temperature for each stage and have devised apparatus by which these conditions can be controlled.

This work of the engineers has produced a great decrease in the cost of gas. Twenty years ago it was possible to reduce the price of gas to a point where it would be made to compete successfully with coal and other fuels, thus greatly increasing the market for it and definitely establishing the stability of the industry.

Some companies which sold gas at as high a rate as \$4.00 per 1,000 cu. ft. in 1870 are today able to produce and sell it at less than \$1.00 per 1,000 cu. ft. and still make a fair return on the necessary investment. This great decrease is the result of the reduced cost of production and of the largely increased output that followed it.

Through the development of the Welsbach mantle it is now possible to get much more light with a given amount of gas than could ever be obtained with the open burner and at the same time to use gas of a low candle power, thus materially reducing its cost. This has helped to place the gas companies in a position where they can successfully compete with the electric companies for a large portion of the lighting business.

The field at the present time is a most excellent one for the engineer. There are still opportunities to improve the methods of carbonization of coal and the generation of water gas. The highest efficiencies in these departments have been by no means reached. The actual chemical reactions in the process of carbonization and the effect on them of various temperatures offers a subject for further profitable study.

There is opportunity to increase the quantity of residuals recovered to believe that the market for these residuals can be substantially increased.

A small percentage of saving in the cost of the product is quickly reflected in the year's earnings. A

company selling 500,000,000 cu. ft. per annum is one of medium size and yet a reduction of 1 ct. per 1,000 cu. ft. in the cost of manufacture or of distributing the gas means a saving to that company of \$5,000,000 per annum.

It has been demonstrated that gas can be manufactured by a large plant and distributed by means of a high pressure service to smaller communities many miles distant, where it can be sold at a price much less than that at which it could be manufactured and distributed locally. This is a development which is certain to be of future importance, but calls for more engineering skill than it has received.

Gas is becoming used more freely for industrial purposes and this field should have the most careful consideration.

The man who is best equipped to improve and develop the gas business should have the knowledge of chemical and mechanical engineering that can be obtained at the institute, and for such a man there should be a successful future in this industry.

A recent U. S. Consular report states that there is much interest at present in the possibility of important developments in New Zealand as regards its presumably large resources of petroleum and of iron ore. On the western coast of the North Island of New Zealand, in the vicinity of Mount Egmont and close by the sea at New Plymouth, there has been for many years more or less attention paid to promising indications of mineral oil, but until recently attempts to bore for the oil have not met with much success, the wells apparently not being sunk to a sufficient depth to get the oil. Lately, however, three bores sunk by one company to a depth of about 2,500 ft. have resulted in a fair flow of oil being obtained, and also a strong pressure of natural gas has been revealed. From these bores about 20 barrels of oil a day are now being obtained, this oil being shipped in a crude state to Auckland and Wellington, where it is used in the local gas works. The New Zealand Government some time ago offered a bonus of 6 cts. a gal. on the first 500,000 gals. of mineral oil produced in the Dominion, and the company operating at New Plymouth has just claimed and been allowed one-half of this bonus, amounting to \$15,000, for the first 250,000 gals. produced. The attention of English capitalists has been drawn to the district and it is now reported that a sale of the property owned by this company to an English syndicate has been effected and that a new company will shortly be formed, improved machinery purchased for boring operations, and a refinery erected. It is also proposed to utilize the natural gas for power purposes. The oil appears to be rich in materials valuable as by-products. Refined samples contain about 15 per cent. of petroleum spirit below 150° C. (302° F.), 42 per cent. of kerosene distilling between 150° C. and 300° C. (302° F. and 572° F.), about 20 per cent. of lubricating oil, 13 per cent. of paraffin, and 5 per cent. of coke.

THE ACTION OF OZONE ON AIR.*

By M. O. TROY.

Perhaps one of the most universal applications of ozone will be in the treatment of air for the destruction and removal of noxious odors, organisms and emanations. This subject has received some attention in America where the matter of ozone application is a new one, and much attention in Europe, particularly in France, where ozone has long been recognized as a valuable agent of sanitation.

The general consideration of ozone, per se, will not be dwelt on here, the subject having been extensively treated in various publications.

Necessarily what follows must partake somewhat of the nature of a disquisition on the immolation of the bacterial content of the atmosphere; for notwithstanding the arduous and exhaustive researches of such men as Erklentz and Flügge, Atwater, Heyman, Paul, Dr. Leonard, Hill and others, we are not in possession of a great deal of unquestionable information respecting anything noxious in the atmosphere, with the exception of moisture and bacteria.

Ozone acts on the air as a bactericide as well as a powerful agency of deodorization. For the purpose of studying the power of ozone to destroy noxious odors, Scoutettin chose a ward of the hospital at Metz, having a magnitude of about 1,100 cubic meters. In this hall he placed two piles of manure about 10 meters apart. These manure piles were permitted to remain for 48 hours, during which period the room became filled with a pernicious odor indicating an advanced stage of putrefaction, as shown by the evidence of the ammonia evolved.

When this had been accomplished, two vessels of 8 liters capacity were opened in the hall, permitting their contents of ozonized air to diffuse therein. The ammoniacal odor diminished considerably, though it did not disappear completely. The manure was then removed and the experiment repeated. This time the odor disappeared completely and rapidly, the noxious gases, hydrogen sulphide, carbon bisulphide and ammonia having been destroyed.

According to Schoenbein 60 liters of air containing 13 mgs. of ozone will disinfect 324 cubic meters of air. This figure is somewhat equivocal, however, as the quantity required in any given case will depend on the nature and condition of purity of the air.

It will not be out of place to describe briefly the fundamental routine of bacterial culture, the reader being referred to special books on the subject for further details and information.

The various germs differ among themselves in many ways, a most important difference being their growth and behavior on the organic culture media which are used for the purpose of studying them. It is necessary to choose culture media which are habitable by the various germs studied. Briefly the procedure consists

*From the General Electric Review.

in applying the bacteria, which are of microscopic dimensions and totally invisible to the unaided eye, to the surface of a medium on which they can thrive, and in the course of a given period of time to count the number of colonies which have appeared on this surface.

Bacteria proliferate by a process known as segmentation; i. e., they divide at the middle into two smaller cells which grow and in turn divide each into two others, etc.; the process repeating itself continually about every thirty minutes.

The plates consist of shallow glass dishes into which is poured the culture medium, which is permitted to jell in a horizontal position. These dishes, which are called "Petri dishes," have a diameter of from 10 to 20 centimeters, with straight sides.

The culture medium consists of gelatin, gelose or agar agar, specially prepared and rendered perfectly sterile before use.

From what has been said of the rapid proliferation of the germs, it will be seen that at the end of a comparatively short time the surface of such a culture plate which has had germs strewn upon it will be shot with colonies of bacteria, the number of these colonies representing the number of original germs, providing they are sufficiently sparse to fall singly. The planting of the germs on the sterile plate is technically called "inoculation."

The experiments of Chappuis on bacteria in air have demonstrated the bactericidal value of ozone. He prepared sterile cotton wadding plugs through which air was drawn, and then exposed one half of these plugs to the action of ozonized air, the other half being kept as checks. The plugs collected the dust in the air.

A culture medium of yeast bullion was prepared in some test tubes, and into these were introduced the cotton plugs. The yeast bullion exposed to the ozonized cotton remained clear, while that exposed to the untreated plugs became turbid, demonstrating the presence of numerous micro-organisms in the dust collected from the air. This shows that the germs of the air, or at least those capable of growing on yeast bullion, are destroyed by ozone.

Various experimenters have made tests with the more dangerous disease germs, and De La Coux reports experiments with *bacilli carbonis* (black-leg disease of sheep, cattle and goats) *typhoid bacillus*, *staphylococcus*, *spores of aspergillus niger* and *diphtheria bacillus*, in which these germs failed wholly to grow in air containing 1.5 to 2.0 mgs. per liter, i. e., from 0.6 per cent to 1 per cent of ozone by volume.

Some experiments have tended to show that ozone does not possess a germicidal action. Prof. Oudin has investigated this subject and has found that if the cultures of bacteria grow deep within the culture medium, the action of ozone seems not to be inhibitive. He has shown on the other hand, that where a sample of a culture colony, taken from such a deep seated growth

which had previously been exposed to ozone, is from the exposed layer at the surface, no inoculation of a second plate can be obtained.

If the ozone concentration of the air be high; e. g., 8 to 10 gms. per cubic meter, the germicidal action will be found to be quite powerful even in deep seated cultures.

Experiments with cultures of the *tubercular bacilli* have shown that these grow with only one fourth the rapidity of check cultures, when exposed to the action of ozonized air.

These results show that where ozonized air comes in contact with the living colonies their development is impeded; but that when the bacterial colony grows deep within the culture medium, the action of ozone applied to the surface only is less marked, if not altogether imperceptible.

This is what should be expected according to Ohlmüller, who has demonstrated that the bactericidal action of ozone is greatly interfered with in the case of colonies growing on organic matter; for the ozone oxidizes the organic medium, thus destroying itself, before it makes sensible its action on the bacteria. Ozone destroys itself in oxidizing organic matter and coagulates albuminous matter.

It may be deduced from the foregoing that any extraneous organic matter found in air which it is desired to sterilize, will diminish the action of ozone by combining with it; and in consequence, the air should be first filtered whenever practicable. Many failures to produce sterilization in researches on ozonizing air have resulted from the presence of a relatively large amount of organic matter in the air.

Ozone will find an application in the sterilization and deodorization of the air of hospitals, apartments, studios, schools, etc., wherever there is likely to be large crowds.

In stables, chicken coops, toilets and factories, where there are evolved noxious emanations, ozone will greatly ameliorate the conditions. In particular, the shops for assorting rags, manufacture of fertilizers and factories which work gelatin, glue, hides, hair, fat, bones, horn and other slaughterhouse by-products, and those which are a source of emanations dangerous to the public health, will find in ozone a powerful ally.

Wherever pure sterile air is of value in the factory either before, during, or after the completion of the product, e. g., distilleries, breweries, wine houses, etc., the use of ozone should be resorted to.

Experimental installations of ozone apparatus have been made by the engineers of the General Electric Co. for the purpose of studying the applicability of ozone to the purification and deodorization of air under various conditions.

The Art Theater on State St., Schenectady, N. Y., a moving picture show, had experienced difficulty with its ventilation. The theater consists of a hall about 30 by 100 ft., and the ventilation is provided by a suction blower capable of aspirating about 90,000 cu.

ft. per hour. The management were very desirous of providing the best ventilation possible, as is evidenced by the elaborate and expensive system cited. It was found, however, that notwithstanding the magnitude of the blower, "crowd odors" persisted in the room. The blower was as large as could be used, for anything larger would have produced obnoxious draughts.

As a solution to the trouble, an ozonator was installed above the front entrance to the theater, in such a way as to permit the ozonized air to diffuse into the current of ventilating air drawn toward the aspirator. The instantaneous effect of this was remarkable. The theater has been entirely deodorized and even during the hottest weather of the present summer the air within the theater has been fresh, cool and odorless, excepting for the faint and rather pleasant smell of the slight excess of ozone.

The next case which we may cite provides an even more remarkable instance of the efficacy of ozone in deodorizing obnoxious air, since this case relates to a factory in which, through the nature of the work carried on, emanations are evolved, which constitute a vehicle of certain volatilized dilutents and solvents of the varnishes and adhesives used. In a workshop some fifty feet by one hundred feet upwards of two hundred girls are employed in the preparation of vari-out articles of pasted mica. It is easy to realize that the problem of providing clean air under such conditions will always be a difficult one, and in the present instance a considerable expenditure of money and ingenuity was incurred before the correct solution was found. In order to obtain a sufficient supply of fresh air to counteract the effects of the noxious fumes it was necessary at all seasons of the year to have all the windows of the building open to their fullest extent. This plan was feasible during the summer months, but, during the winter, when the temperatures below zero were encountered outside, it was necessary to provide a very costly system of heating, in order that the temperature of the room might be sufficiently high to make the place endurable. Many thousands of dollars were also expended in providing draught pipes of large diameter for conducting to the outside atmosphere the fumes diffused by each machine. A point was reached at which it became imperative that some less costly and more efficient system of cleansing the air be adopted, and it was at this point that resort was made to the ozonator.

Two of these were installed and were placed on the center line of the room, each one being some 25 ft. from each of the two end walls. The effect of their action was that during the winter months, the windows, which formerly had to be opened to their fullest extent, required now to be opened only to the extent found necessary in ordinary living rooms. The costly draught pipe system for conducting away the noxious fumes from the machines was removed, and the atmospheric conditions, as far as the comfort of the em-

ployes was concerned, were every bit as good as are to be found in any other workshops.

Previously, there were occasions during which the atmosphere became so befouled that the whole staff were compelled to leave thir work through general malaise; while at the present time if through any abnormal condition such as failure of current supply, the ozonators are unable to operate, the reversion to the old state of affairs is instantaneously noticed.

A third instance which we may quote illustrates the use of the ozonator in purifying the air of buildings which have become burdened with smoke and fumes of combustion. Messrs. Charles Holtzmann and Son are the proprietors of a large store on State St., Schenectady, for the supply of clothing, furnishing goods, etc., and were recently put to considerable inconvenience as the result of a fire, which broke out in a stable located towards the rear of Messrs. Holtzmann's premises. Clouds of smoke from this fire invaded the clothes store to such an extent that not only were the rooms on various floors rendered untenable, but the goods with which these floors were stocked became, in many instances, impregnated with the odor of smoke. A delay of several days disastrous to the interests of the management might very easily have been occasioned; but at this juncture an ozonator was installed on each floor with very gratifying results. In the almost incredibly short space of twelve hours the atmosphere on each floor was purified and brought to its normal condition. At the same time the ozone seems to have acted upon the stock, which was carried on these floors, as a deodorant and fumigator, to the effect that all odor of smoke was immolated. This instance illustrates the utility of the ozonator in directions which up to the present have not perhaps been very generally recognized.

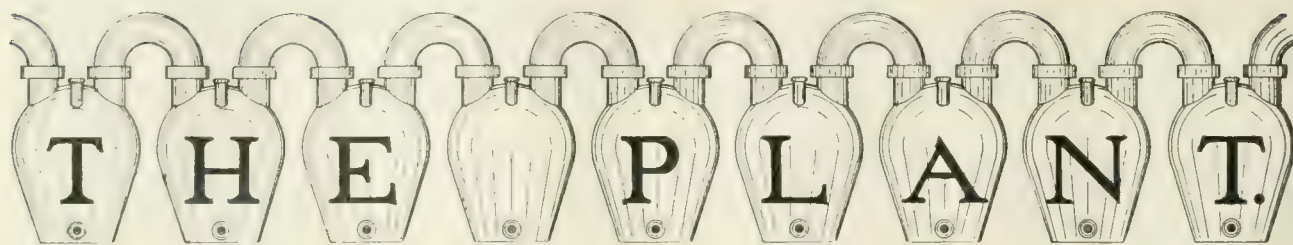
It would be an easy matter to multiply instances affording remarkable evidence of the beneficial uses to which the ozonator may be commercially applied, but the foregoing will probably suffice to give an indication of some of these uses.

In the sterilization of air, the ozone should be blown into the apartment, or the air should be drawn through a special chamber in which the ozone is mixed with it. It is important that the ozone come freely into contact with each individual particle which it is desired to destroy.

The machine for producing the ozone should not produce any *nitrous oxide* or any other gas having an untoward action on the human organism.

The generation of ozone should continue until the air, as determined by experimental test, is thoroughly sterilized, and the machine should produce this result without loading the atmosphere with ozone to an injurious concentration.

The Home & Foreign Cotton Co., Ltd., has recently completed a \$500,000 cotton spinning mill at Shanghai, China.



THE MANUFACTURE OF PORTLAND CEMENT FROM BLAST FURNACE SLAG.*

By RAY S. HUEY,† E. E.

Portland cement is now almost as familiar to the general public as wood or stone, and its uses have become so general and diversified that it has become an invaluable material for durable and fireproof articles in the arts and sciences. The fact that Portland cement is so easily worked, and that almost any one of average intelligence can do a creditable job with it, makes it a building material for which there is an ever increasing demand. It also has the advantage of being a product, the raw material for the manufacture of which can be found in almost any locality in the world, and if the demand is great enough can be made close to the locality in which it is to be used, thereby reducing the cost of transportation.

Portland cement is a combination of silica, iron, alumina, and lime, in proper proportions, the raw materials of which may be obtained from shale, clay or blast furnace slag and limestone or marl. The process of manufacture, using the blast furnace slag and limestone, at the Buffington plants (Nos. 3, 4 and 6) of the Universal Portland Cement Co., is the one which will here be described.

Many persons are under the impression that the slag from furnaces of any description, blast or open-hearth, can be utilized for making cement, but this idea is erroneous. Slag suitable for cement is very carefully made, and of suitable materials—for example, the slag from a blast furnace using dolomite (magnesium limestone) as a flux cannot be used because the percentage of magnesia in the cement would be too great to pass standard specifications.

Slag is a logical raw material for making Portland cement; it contains roughly 36 per cent silica, 14 per cent iron and alumina, and about 46 per cent lime, is readily fusible, and simply requires the addition of more lime in the proper proportions to make the mixture for first-class Portland cement. It is in a finely divided state, so does not require crushing, and consequently is readily handled. Blast furnace slag suitable for use in the manufacture of cement is made by allowing the molten stream of slag to run from the furnace to a tank where, as it falls off the trough, it is struck by a stream of high-pressure water which cools

and disintegrates it immediately. It is then loaded into hopper-bottom steel freight cars by a traveling crane, which digs the granulated slag out of the water by a clam-shell bucket, and shipped to the cement plant where it is dumped from a trestle into large bins. The slag is in small particles, slightly resembling sand in size and color, and contains a large percentage of water, varying between 20 per cent and 40 per cent—depending upon the physical character of the particles and the temperature at which it was granulated.

This slag is discharged from the raw material bins into elevators which carry it to the dryers, these being slowly revolving cylinders having compartments lined with flights which turn the material over and over and keep it traveling toward the discharge end, allowing hot gases to come in contact with the wet slag and drive off the moisture.

The slag is now elevated again in bucket elevators and spouted to various ball mill feed hoppers which are intended to keep a sufficient stock in them to keep a mill supplied for some time, should the preceding machine which supplies the hopper have to be shut down for repairs. The slag is preliminary ground in a ball mill, conveyed to an elevator by a belt conveyor, and again elevated to hoppers over the weighing machines or scales. The ball mill consists of a pair of steel discs mounted on a shaft about 5 ft. apart, having 20 heavy cast-steel wearing plates mounted near the circumference. Ten of these plates are solid and ten are perforated, and are mounted in such a way that when the 4,500 lbs. of 4 in. steel balls fall on the material to be crushed, some will be crushed and go through the perforated plate. Between the steel wearing plates and the periphery of the disc are two sets of screens, consisting of one set of heavy protecting screens of perforated metal which protects the lighter and finer wire-cloth screen on the outside from injury in case a steel plate breaks, and which also takes the wear off the lighter screen from material which could not possibly go through the fine screen. All the material which does not go through is returned to the plates and crushed until it will go through.

The limestone, which is the other ingredient of the raw material mixture, is quarried at the company's quarry in the Fairmount District, crushed to about a 6-in. cube or smaller, shipped in steel hopper cars and unloaded into the trestle bins in the same manner as the granulated slag. It is then crushed to about 1½ in. in a gyratory crusher, elevated to a dryer, dried and ground in the same manner as the slag.

*From The Armour Engineer.
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The dried slag and stone are in separate bins above a pair of tandem, automatic, electrically operated scales, and are conveyed to the scale hoppers by means of screw conveyors. The scales are arranged so that a contact is made with a dumping mechanism, which discharges both scale hoppers simultaneously after both are up to full weight. The material is then mixed and carried by a screw conveyor from a hopper below the scales to a bucket elevator which spouts the mixture to the tube mill hoppers.

The final raw material grinding is done in tube mills, each of which is a tube 5 ft. or 6 ft. in diameter and 22 ft. long, and lined with hard cast-iron plates bolted to the shell. The shell is supported by a head at each end on which is a hollow trunnion, the material being fed into the trunnion at one end and discharged through the one at the other end. This mill is about half-filled with flint pebbles and is revolved at about 25 revolutions per minute, thus causing the material to be crushed to a fine powder by the falling of the pebbles on it. The finished raw material mixture is taken on a belt conveyor to an elevator, and then across a bridge by a screw conveyor to the hoppers over the feed end of the rotary kilns in the burner building.

The kilns are steel shells lined with fire-brick about 7 ft. 6 in. in diameter and 120 ft. long, and are set on a pitch, which, when the kiln is revolved, tends to move the material toward the lower end. The kilns turn around once per minute and by the time the material is discharged at the lower end it has been fused by intense heat into balls called clinker. The rotaries and dryers are heated through the combustion of pulverized coal blown in by an air-blast, this producing a flame that appears very much like a big gas flame and a temperature of about 2350°F. at the hottest end of the kiln.

The pulverized coal is produced by crushing and drying coal screenings and then pulverizing them in Fuller mills, one of which consists of a vertical shaft having mounted on it a spider which pushes four 9-in. balls around a hard cast iron ring with sufficient velocity to furnish the necessary centrifugal force to crush the coal. By means of a fan attached to the main shaft above the balls the fine coal is blown through the screens.

The clinker is now elevated and spouted into a large open bin holding about 75,000 bbls., and is picked up and distributed by traveling cranes, to which are attached large clam-shell buckets. While on the pile the clinker is sprinkled with water until it contains about 1½ per cent moisture, this being to slack out the free lime which is always present in small amounts in fresh clinker. When properly seasoned, it is picked up again by the cranes and put into the hoppers over the clinker crushers in the finishing mill. In jaw-crushers, the large lumps are crushed to about 1 in. in diameter, and the clinker falls into the Maxecon mills, to be granulated.

The Maxecon mill is a machine composed essentially of a ring having a concave inner surface, and three rolls having convex outer surfaces made of hard material. The rolls are set 120° apart and pull out against the inside of the ring. The tension between the ring and the rolls is regulated by a spring, each one of which reacts between the frame of the machine and a yoke on which are mounted the bearings for the shaft holding each roll. The top roll is driven, and this through friction drives the ring which in turn drives the two bottom rolls. The material to be crushed is fed in between the ring and one roll and the centrifugal force carries it around inside of the ring to the next roll, this continuing until it drops off through the frame to the elevator. The elevator then lifts the crushed material to a screen in which the fine material is separated and taken to the gypsum scale. The coarse material remaining is returned to the Maxecon mills to be crushed again.

By means of automatic weighing machines about 2 per cent gypsum (calcium sulphate) is added to regulate the setting time of the cement. Cement without gypsum would set in about 3 or 4 minutes, so it can readily be seen that it would be useless to work with as a building material, for the most rapid mixer and workmen would be unable to mix and place the concrete before it would set.

From the gypsum scales the ground clinker is again elevated to the hoppers over the tube mills and given a final grinding. The cement in this condition is finished and is conveyed by a belt conveyor from the tube mills to the storage bins in which it is kept until required for shipments. When this is desired it is drawn out from the bottom of storage bins, through gates into the screw conveyors to elevators, thence to packing hopper, packed by automatic weighing machines into ¼-bbl. sacks, either in paper or cloth, and loaded into cars for shipment.

The total combined capacity of all the plants of the Universal Portland Cement Company is 40,000 bbls. per day, of which 27,000 are made at Buffington, in Plants 3, 4 and 6. This requires about 200 cars of raw material, and about 100 to 300 cars, depending on the season of the year, are required every day in which to ship the finished product.

The finished cement is sampled by an automatic sampler once every eight seconds, and this sample taken to the laboratory every hour where complete tests for fineness, setting time, and soundness are made. Every car is sampled before shipment, and the same tests made in order that there may be complete records for reference. About 96 per cent of the finished cement will pass through a 100-mesh sieve, which has 10,000 holes per square inch—the diameter of the wire forming the sieve being .0045 in. About 80 per cent will pass through a 200-mesh sieve, having 40,000 holes per square inch—the wire in this case being .0024 in. in diameter.

Chemical analyses are made continually to keep the

ingredients in the raw material constant, and strength tests and analyses are made daily to see that the quality is kept up well above the standard requirements.

All the machinery is electrically driven, and requires approximately 10,000 H. P. at the sub-station switchboard. The power required is generated by the waste gas from the blast furnaces, by either steam engines, steam turbines or gas engines, and transmitted ten miles at 22,000-volts, 3-phase, and 25-cycles, to the sub-station in which it is transformed to 440-volts and distributed to the different buildings by independent switches, so that in case of trouble in one building it can be disconnected and the trouble remedied. The machines are individually driven and as far as possible direct connected, thus eliminating many belts and geared speed-reducing devices. This makes it possible to shut down each machine for repairs, when necessary, without disturbing the rest of the mill, and so allows the mechanical department to keep the plant in a better state of repairs.

The motors are all of the squirrel-cage induction type and give very little trouble under the severe loads and dust conditions which are found in a cement plant. The progress made in the improvement of machinery for grinding cement has been remarkable in the last ten years and nearly all the machinery installed then is now out of date.

From a college man's standpoint a cement plant is an interesting place. To be a successful operating man at the head of a large cement plant, he should be a chemical, mechanical and electrical engineer combined, and the more thorough the ground-work the more capable man he will be. In detail, regarding the engineering qualifications of the head of such a plant, he should know the effect of certain variations of the raw material. It is also necessary to know the effect of the chemical constituents of the steel and other materials which are used in machinery, in order to get the most efficient and durable material to be used in new machinery, or in making repairs. As a mechanical engineer he is called upon to figure strength of parts, size of pulleys, capacity of machinery, and design new improvements.

Since my connection with the cement plant, my viewpoint regarding machinery has been entirely revolutionized. I formerly thought that when a machine was purchased it was ready to do the work for which it was designed without any changes. I have found, however, that with few exceptions there is hardly a machine on the market which does not need reconstruction to make it better and more adaptable to the work required of it and there is frequently a small detail, the lack of which will make it a failure. It is here that a thorough knowledge of mechanical engineering coupled with a practical experience gained under individual conditions is valuable. It is therefore necessary to scrutinize carefully the detail drawings of every machine and try to see in one's mind whether the design of a new machine cannot be al-

tered to adapt it to the work to be performed in your particular plant, as most machinery seems to be designed by persons who have had little practical operating experience, and consequently know but little of the difficulties which are encountered in the operation of their own machinery.

On account of the increasing use of motors and electrical apparatus in connection with the cement industry, if one is to be familiar with the details of the work under him he must be qualified to pass on the design of motors and transformers and all kinds of improved electrical apparatus which must be built to suit certain conditions. He must be able to design and calculate new installations and make specifications to meet his peculiar requirements. He must also know how to diagnose electrical troubles and prescribe the cure. When one is qualified to meet all the above specifications, he still has the most difficult problem before him, which is that of handling men. To have a good organization which will pull together and produce results at a minimum cost, is the goal of every man in an executive capacity, not only in the cement industry, but in every other kind of business.

According to an article by Mr. Walter Freudenberg of Bremen, the estimated planted acreages of rubber as given by three of the leading trade publications, are as follows: India Rubber Trades Diary, 776,000 acres; India Rubber Journal, 980,000 acres; Gummi-Zeitung (Berlin), 1,310,000 acres; Mr. Freudenberg takes the second estimate as the basis for future yields, and gives the future supply as follows: From Malaya (authority, Sir John Anderson), 70,000 tons; Ceylon (authority, Ceylon Observer), 19,000 tons; Dutch Indies, Borneo, South Mora, and Burma, 20,000 tons; total annual supply of plantation rubber in 1916-17, 109,000 tons. To this must be added the probable supply of wild rubber, which until recently has been about 70,000 tons annually. He considers, however, that the supply from this source may decrease, in view of the larger crop of plantation rubber, and estimates only 35,000 tons, making a gross total of 144,000 tons in 1916-17. As to present and probable future consumption, it would appear from available statistics that for the year ending June 30, 1910, a total of 76,000 tons was used, showing an increase of about 5 per cent annually during the past 10 years; and if this rate of increase continues, about 107,000 tons would be required by 1916-17. But this rate of increase does not, of course, take into consideration new uses to which rubber may be put.

A \$200,000 California corporation has started quarrying onyx in the Serro Blanco Peninsula, Lower California, about 200 miles south of the international boundary. Thirty men are now employed in the company's quarries, and monthly shipments will be made from the landing at San Jose. This Mexican onyx lays in flat ledges close to the surface.

METALLURGY.

TITANIUM AND ITS BENEFIT IN IRON.*

By CHARLES V. SLOCUM.

When the manufacture of titanium in the shape of an alloy was undertaken on a commercial scale in 1907, the results in primary tests were considered for a time as the basis upon which it would be necessary to place this material on the market. When a number of trials had been made, however, it was found that different results were apparently secured and it then became necessary to modify these original ideas in accordance with these developments and in fact since three years is not a long time to learn everything in regard to a new element, this modification is still going on.

When it is remembered that the output of titanium alloy exceeds that of all other alloys combined above the grade of manganese and silicon, and that this development has been made in such an unusually short period, it is safe to assume that as time goes on the results will be still more developed, percentages changed for certain purposes and the whole situation become more definitely systematized and made to conform more readily to the methods and ideas which time and experience can alone develop to the fullest extent.

It was the idea at first in using titanium in iron that since comparatively large percentages gave increased strength in tensile and transverse tests, that these larger percentages were essential to the process of securing a titanium reaction in the metal and therefore must be used. Gradually, however, it became evident that minute percentages of titanium were successful in producing many of the benefits of a larger quantity. As for instance the use of two-tenths of one per cent of alloy which contains only 0.02 of metallic titanium is frequently sufficient for the purpose desired, which consists principally of cleaning the iron of impurities remaining in the metal after the ordinary fluxes have accomplished all that they can in this direction. This cleansing is not one which is possible with limestone or with fluor-spar, for these fluxes have no effect whatever upon the oxides or nitrides contained in the molten iron.

Titanium, on the other hand, removes both of these objectionable elements partially in any event and wholly or practically so when sufficient alloy is used for the purpose. The effect then of the small fractions of a per cent in iron is, more practically speaking, to increase the fluidity of the iron and at the same time to make the metal more homogeneous, so that the

resulting castings are closer grained, are free from pin-holes and gas bubbles or blow-holes, the iron has a denser structure and at the same time is easily machined.

The shrinkage is less in the treated iron than in the plain or untreated casting. This is particularly true in hard or chilled iron, where heavy castings, such as car wheels, will shrink an eighth of an inch less or one whole tape number in circumference.

Much finer work may therefore be done with iron treated with the alloy, since the casting will conform more nearly to the original pattern and will have a more uniform shrinkage than the plain iron.

In a letter dated April 5, 1911, the writer was advised by Asa W. Whitney, metallurgist of the Enterprise Foundry & Machine Co., Bristol, Va., that he has made a number of careful trials of titanium in hard or chilling iron. He finds that 0.1 to 0.2 per cent of the alloy is usually all that is necessary to make otherwise viscous, high chilling mixtures come from the cupola close grained and pour from the ladle as freely as iron carrying half as much chill and of more open grade. The iron pours well to the last and gives clean solid castings. "I find," Mr. Whitney says, "that I am able to compensate for the cost of the titanium with less manganese and a trifle less silicon."

To foundrymen the fact that the metal remains hot longer than untreated iron is a matter of much importance for certain classes of work, since the greater fluidity means that the iron will settle more slowly in the mold, and thus give time for the gases to escape and more time for the iron to fill every smallest outline of the mold without pulling away from the main body of the casting. These features of good foundry practice are some of the ones which are often overlooked, and a high percentage of bad work results. Now in relation to the benefits to be expected from using larger proportions of alloy, I maintain that a great deal of undue importance is often attributed to an increase in transverse and tensile strength. This may be necessary for certain government work and for some few castings which must resist unusual stresses in service, and for such it is necessary to use at least 1 per cent of titanium alloy.

My contention is, however, that such a percentage is not only a serious increase in cost per ton of product for ordinary castings, but in all the practical uses to which most castings are subjected no special increase in strength is required, although an improvement in quality may be absolutely necessary. We have in mind a recent case where the castings in a certain foundry were stronger than necessary, were well made and were satisfactory to the foundry superin-

*From a paper presented at the Pittsburg Convention of the American Foundrymen's Association

tendent and to his customers, but when he tried the small quantity of alloy usually recommended for such cases merely for the purpose of improving the fluidity of the molten metal and the density, machining quality and durability of the casting, he found that the machine shop of one of the great railroad companies which turned up the casting reported that these treated specimens (piston rings) were closer grained, more like steel and were more desirable for their purposes. Yet of the four test bars cast, one of untreated iron was strongest in transverse strength.

On the other hand a large manufacturer of my acquaintance having a ten ton casting to make and being anxious to have no failure in so important a job, added 1 per cent of titanium alloy and secured a splendid casting which may have cost him \$25 extra for all the alloy used, but which made a sure thing of a single part of the machine worth at least \$400 even without including the machine work.

To summarize these statements then, it may be said that titanium alloy is of great benefit in small percentages for fluidity, for sharper castings, for remarkable uniformity and for durability, while the larger percentages are for strength and density of metal in addition.

I realize of course that certain classes of work do not justify an increase of cost to any extent worth mentioning, and yet even in cheap work the use of a pound or two of alloy per 1,000 lbs. of iron is a distinct benefit to the metal and costs but 25 to 50 cts. per ton of product. Other classes which will perhaps bear a little more expense may be increasingly benefited with three or more pounds per 1,000.

One important feature of titanium in iron should not be overlooked, viz.: the great durability of all metal into which it enters. In some tests made by Prof. M. Hokanson at the Carnegie Technical Schools some two years ago the following results were obtained:

Chilled test blocks machined down to approximately $\frac{1}{2}$ in. by $\frac{1}{2}$ in. by 1 in. were used.

First Test.—Under crushing strain, blocks from plain untreated iron developed 173,000 lbs. per square inch at failure. Metal from same tap of cupola, but treated with 1 per cent of titanium alloy, developed 298,000 lbs. per square inch at failure.

Second Test.—Chilled test blocks from a different foundry and made from plain iron, machined as were the others previously mentioned, developed 200,000 lbs. per square inch crushing strength at failure, while similar pieces from same tap of cupola, but treated with 1 per cent titanium alloy, developed 392,000 lbs. per square inch at failure. It seems to be this quality of burden bearing imparted by titanium which has made such a demand for its use in steel rails.

Iron castings which require great care in molding and which are difficult at all times to make may well be treated with titanium, for such castings cost much money to manufacture and a high percentage of loss

works havoc in the showing of profits for a year's business.

In cylinders, for instance, and for locomotives and stationary engine castings of various kinds, many manufacturers from the Atlantic to the Pacific are now using this cleanser and purifier as a means of not only improving the casting itself, but of reducing that deadly percentage of loss which often stares the foundry officials in the face from one month's end to another.

The feature of greater fluidity cannot be emphasized too strongly. When we seek for increased fluidity through the addition of ordinary fluxes the result is often disastrous, for much fluxing makes the metal boil and blow holes, pin holes and defective castings generally result, together with increased expense in the repairs or even in actual relining of the cupola, sometimes necessary from this method of procedure.

When titanium alloy is added in the cupola, it does not affect the lining unless large percentages be used, and when the larger percentages are necessary it is better to add the alloy in the ladle with the usual precaution of keeping the alloy from the slag so that the full benefit of the use of this cleanser may be secured. There are times when it is not practicable to use the alloy in the ladle and other times when it should be used there.

To illustrate, in adding 1 per cent of alloy for purposes already mentioned it should preferably be done in the ladle and, if practicable, in the receiving ladle, but this can only be done there when it is possible to keep the iron free from slag by skimming or by some other means. Titanium attacks slag as a bull dog attacks an enemy, and the former should therefore always be used to cleanse the iron from impurities remaining in the metal after the melting process is completed and never by any chance be allowed to reach the slag already on the surface of the iron or about to be removed from its surface.

These facts are presumably quite fully understood by the best foundrymen, but occasionally it is found that some one continues to try the experiment of throwing the alloy into the ladle after the iron is already there and necessarily secures very indifferent results, if any.

In a recent trial of titanium alloy in the cupola with a mixture of burnt iron, stove plate, etc., in a large foundry in the Pittsburg district the rather large percentage (for iron) of half of 1 per cent of alloy was used, together with one-half of 1 per cent of ferromanganese of the usual 80 per cent grade. This trial demonstrated the interesting fact that the poorest mix which can be imagined perhaps may be brought to good normal number one iron by this method. The average of all the test bars, of which there were ten, was 3,100 lbs. breaking strain and an average deflection of .146 in.

Under ordinary conditions, with the addition of the

manganese only, the strength would scarcely have reached 2,000 lbs. per square inch.

The name of the plant where these castings were made will be cheerfully given to any one desiring to make the same kind of an experiment. It should be remembered that burnt iron is usually high in sulphur and the iron in these castings contained .151 and .147 sulphur respectively by two different analyses. Such a remarkable strength from such remarkably poor iron is unobtainable at anything like the small cost by any other process, since the expense of making the metal homogeneous, of excellent wearing quality, etc., was about \$1.50 per ton of castings. The iron before treatment could scarcely have been given away for ordinary purposes and was possibly worth \$10 per ton. The complete analysis of the treated iron by two separate determinations was as follows:

Silicon	C C	G C	Phos	Sulph.	Mang
1.30	.70	2.50	.501	.151	.71
1.30	.70	2.42	.455	.147	.79

Average of ten test bars, 3,100 lbs.

Minimum, 2,900 lbs. Maximum, 3,265 lbs.

Not to be misunderstood, I desire to repeat in brief that high sulphur iron may be made strong and available for practically all purposes by an expenditure as above of \$1.50 or less per ton of metal melted.

In the works of one of the best known foundries in Columbus, O., they use for certain purposes a cheap mixture which analyzes about as follows:

Silicon	1.60
Sulphur	0.11
Phosphorus	0.55
Manganese	0.50

One-fourth of 1 per cent of titanium alloy was added in the cupola and the foundry foreman reported immediate benefit in the fluidity of the metal and in the distinct improvement to the castings. This rejuvenation of the iron, so to speak, involved an extra cost of 62½ cts. per net ton of metal treated, and was more than repaid in the reduction in bad work without regard to the benefit to the iron. Castings from this iron, with fine grain and good metal, were made which were only 5-16 in. by ¼ in. section.

As a substitute for charcoal iron, titanium is making much progress. In a letter dated Jan. 9, 1911, the secretary of the American Foundrymen's Association, Dr. R. Moldenke, states as follows:

"I am getting more and more confirmation that by the use of titanium as one of the most easily oxidized elements I know of, we will eventually make a common coke iron equally as good for our purposes as the best cold blast charcoal iron of the same composition."

In a letter dated Dec. 3, 1910, Prof. John J. Porter of the University of Cincinnati gives expression to the following idea of the merits of titanium as a means of securing better castings without regard to the mere question of strength, which as a rule takes care of itself in a well regulated foundry:

"I believe most thoroughly in the great value of titanium in cast iron. I know that it will remove

many of the causes of bad castings and feel sure that many foundries can save money by using it, in addition to obtaining a better quality of castings. I have several times seen this demonstrated."

Now there are a few suggestions which occur to the writer and which may have become familiar to others, but it will do no harm to repeat them, viz.:

In using any alloy do not govern the quantity or the method entirely by the mere untested statements of others, but be governed largely by the particular features and practices of the foundry in which the trial is to be made. For instance, one of our best known manufacturers reports a failure of titanium alloy to improve iron when used in a cupola which was not slagged and which therefore kept the titanium in almost constant contact with the very impurities which should have been kept away from it. Such a poor use of good gray iron should be placed in the dark corner and kept there for all time.

Another questionable procedure is to have a few of the castings looked after carefully by some good man like the foreman or even the superintendent, and then to have the other titanium additions for the same kind of iron and from which the same results are expected, looked after by some laborer or "hunkey" with as much knowledge of the necessities of the case as the city directory has of metallurgy.

The don'ts are perhaps more necessary in the use of this alloy than important rules of guidance. We put an alloy into iron or into steel for a distinct purpose, but unless we can at the same time place a little common sense and good foundry practice along with it, the results will be doubtful in the extreme. We can undertake to put a great deal of pressure upon the foreman, but unless he has sufficient foresight to realize, for instance, that silicon, although used as a softener, is also at times a great weakener of good iron and therefore should be used as sparingly as possible with titanium (which does all the softening necessary without doing any weakening at all), it would be useless to instruct him as to the merits of any alloy. A mere man is only one of the most obtuse of God's creatures unless he uses the good gray matter with which most men are endowed and which is so necessary in the manufacture of good gray iron.

In conclusion the uses of titanium in iron may be summarized as follows: Very small percentages, as low sometimes as one-tenth of 1 per cent (0.1) of alloy (only .01 Ti.), are sufficient to cleanse the iron of impurities not touched by limestone or fluo-spar, while larger percentages increase the improvement in other directions, making the iron more fluid (usually better), making the castings sharper, finer grained, free from blow holes and pin holes and easily machined, while at the same time the expense runs from 25 cents per ton of castings up to say \$3 as a maximum, all of which is often repaid in the reduction of bad work and in improving poor irons.

PURE IRONS FROM THE OPEN HEARTH FURNACE.*

By DR. ALLERTON S. CUSHMAN.

It is the object of this paper to present to your attention the important step in metallurgical development which has made possible a process for making irons of extraordinary purity on the large scale usual in steel-making operations in the open-hearth furnace. Purity in a commercial product is difficult, if not impossible, to define, and it is necessary in the beginning to state emphatically that although the words "pure irons" in the title have been selected advisedly, there is no intention to go beyond the actual facts.

That the electrolytic theory of corrosion is now accepted, by the large majority of authorities both in this country and in Europe as having left the realm of pure theory for that of fact, is shown by numerous very recent contributions to this subject. In the early days of these investigations experimenters found it extremely difficult to obtain samples or specimens of steel which were to any extent free from manganese, although, of course, many examples of charcoal and puddled wrought iron of a high degree of purity in this respect could be obtained.

About this time, however, a prominent American manufacturer, struck by the voluminous literature which had been appearing on both sides of the controversy, determined to make the effort to produce open-hearth metal of the highest possible commercial purity, which would be not only as free as possible from manganese but also from the other four chief impurities which have in the past been taken special note of in the manufacture of steel. This decision on the part of the manufacturer took the experimental work for the first time from a purely laboratory to a large mill scale of operation.

As this experimental work proceeded, the material at first produced was simply a low-manganese mild steel; and while the movement was decidedly in the right direction it was only a step forward and the product obtained was not very essentially different from mild steels that had been produced elsewhere from time to time. When, however, in following up this work the attempt was made to eliminate the manganese and carbon completely from the metal, the resulting ingots were of such an oxidized character as to be entirely unsatisfactory and to lead to most serious troubles in the attempt to roll and manufacture it into a final product. In fact, material made at this stage of the progress of the investigations was not pure iron and was of little utility.

It has been the usual practice in the past to apply the term iron when used in its specific sense only to the products of either the charcoal knobbling fire or the puddling furnace, although, the product of the busheling or fagoting method of production has also

yielded material which has managed to masquerade as iron. In conjunction with the results of chemical analysis, the microscope has recently been developed as a means of enabling the metallurgist to easily distinguish iron from steel. These methods of investigation have gradually led to considering iron to be a name applicable only to products which show slag inclusions under the microscope. Steel, from the very nature of its manufacture, does not exhibit this peculiarity, and it is not surprising, therefore, that the mere presence or absence of slag should gradually have come to be held as the distinguishing characteristic between steel and iron.

The practical production of a new form of the metal, by a process in which the product was finished in the molten condition and cast into molds, necessarily yielding a slagless body, has reopened the question in some quarters as to whether or not the claim is justified that these pure forms of metal should be called iron and not steel. From the purely scientific standpoint, when we use the word iron, we must inevitably refer to a product which in all of its characteristics essentially corresponds to the element itself. It was, of course, impossible to decide just where the border line between an iron and a steel should lie, until the results of metallographic investigations which have been made within the last few years had taught us the essential difference between the pure ferrite structure and the carbon-iron steel.

From the modern standpoint, neither the presence nor absence of slag could be held to settle the matter of nomenclature, except that the presence of slag inevitably points to the process by which the material was prepared. In the opinion of the writer, it is difficult to understand how a product which is in every way essentially metallic iron and exhibiting a true ferrite structure, could be defined as steel after the carbon had been eliminated to practical traces irrespective of whether the metal is found to contain inclusions of slag. In fact, a metal which contains 3 per cent or more of a foreign body such as slag falls just that far short of being pure iron, and from a perfectly reasonable standpoint it would appear that after the carbon-iron alloy field has been eliminated, the nearer the metal approaches to the theoretical constitution of the element itself, the more it justifies the name of iron.

This pure iron made in an open-hearth furnace has become generally known in this country and to a certain extent in Europe as "ingot iron." This term, as we are well aware, originated many years ago and was applied chiefly to the lower carbon product of the Bessemer converter and the open-hearth furnace. Of late years, however, it has fallen into disuse and from the standpoint of the steel maker it was undoubtedly a misnomer, so that gradually the term mild or soft steel took its place. Low-carbon Bessemer or open-hearth steel which usually carries from 0.3 to 0.5 per cent of manganese or over is not iron, for the simple

*From a paper presented at the annual meeting of the American Society for Testing Materials, June 27-July 1, 1911.

reason that it does not show the true ferrite structure and contains enough carbon to provide it with mild hardening qualities. Besides these facts, mild steels possess many qualities not possessed by iron and may be used for purposes for which iron would be unsuitable, although this applies more especially to the slag-bearing iron as made by the original processes. On the other hand, a pure slagless iron is probably superior for many other purposes to the general run of carbon-bearing metals. It would appear that the term "ingot iron" is especially suitable to a pure iron product made in an open-hearth furnace and cast while molten into ingot form.

Practically all the scrap available in this country carries copper varying from very small quantities to amounts in some cases exceeding one per cent. A great many conflicting statements in regard to the influence of copper upon iron have appeared in the literature, but in most cases it appears to have been considered an undesirable element, imparting the property of red shortness to iron or steel. When present in considerable quantities, this is unquestionably the case, and when accompanied at the same time with sulphur the effect is highly intensified. In very small percentages copper does not seem to appreciably affect the working or rolling of the product.

Quite early in the investigation with which the writer has been associated, it was found that certain quantities of copper when added to steel would very appreciably retard the solution of the steel in dilute mineral acids. This was also found to be true, though to a much less extent, with the carbon and manganese-free product of the open-hearth furnace, for the resistance to acid attack is to a large extent influenced by the purity of the metal. In fact, the presence of oxygen in the metal has been found to be more responsible than that of any other element as far as the much-discussed question of acid resistance is concerned. In the writer's experience and practice, the question of copper in iron has been given very careful attention; a wide range of heats have been made, which have purposely been given increasing quantities of copper up to as high as 1.5 per cent. These iron-copper alloys were of necessity made on the usual large scale of operation, but no particular difficulties were encountered in rolling or finishing them. It should not, however, be understood that the intention has been to have copper present in these purer forms of open-hearth irons, for, on the contrary, even though it should appear that the presence of copper within reasonable limits is no detriment to the product, the effort would still be strenuously made to reduce the percentage of copper to the lowest possible point.

If the selling price of the product was commensurate with the expense attendant on securing metal entirely free from copper, there is no reason why such irons could not be readily produced, but it would im-

mediately render useless for the purpose much of the pig iron and most of the scrap at present available.

The manufacture of pure iron in open-hearth furnaces in its entirety is undoubtedly a distinctly new development, in spite of the fact that the process has a great many points in common with the open-hearth production of mild steel. In the case of the iron the oxidizing operation under highly basic conditions is carried forward to an abnormal extent. It is not too much to say that the effort is made to do what has generally been conceded to be bad practice in an open-hearth furnace, that is to say, the metal is deliberately over-burned. This treatment yields a bath of metal and slag heavily charged with gas and in a highly super-oxidized condition. The elimination of the carbon and manganese to such low percentages that they amount to no more than traces makes this super-oxidation a necessary step in the process.

Having obtained the desired purity with respect to these elements the bath is given a heroic treatment, whereby a thorough deoxidation takes place, the gas being at the same time satisfactorily eliminated. No difficulties of any importance are encountered in the pouring of the metal into the ingot molds. It would naturally be expected that such treatment as described might yield a wild metal difficult to handle in the pouring, but this is not the case if the work has been satisfactorily performed. As a matter of fact, the resulting ingots of this process not only possess great purity but are as free from oxygen and gas as any ingots produced from well-made mild steel.

The process is necessarily a more expensive process than that of soft-steel manufacture, owing to the necessary prolongation of the heat and the higher temperature required during the latter stage of the operation. For these reasons, labor, furnace repairs, ladle linings, fuel charges, and the item of metallurgical waste must all be necessarily in excess of those encountered in the making of soft steel. The process, however, holds forth the same tonnage possibilities as in the making of mild steel, and it permits of the making of iron on a basis comparable to the high-pressure methods employed in modern steel manufacture. It also, in contradistinction to the previous methods of iron manufacture, permits of the making of iron in very large masses, and the subsequent rolling of the same in a manner similar to that of mild steel. It is a well-known fact that the greater the amount of work that is given to iron or steel, the more superior will be the resulting product. This permits the making of commercial products out of iron that have been customarily made of steel owing to the steel's superior workability when compared with iron made by earlier methods.

Up to the present time ingot iron has not been manufactured into all the products that are customarily made of steel, and of charcoal and puddled iron, but it has been successfully and easily rolled into billets;

slabs, plates, sheets, merchant iron, wire, nails, rivets, pipe and boiler tubes. A wide variety of forgings have also been produced.

TABLE I.—ANALYSES OF NORMAL INGOT IRON

	(1)	(2)	(3)	(4)	(5)	(6)
Silica	0.003	0.002	0.005	0.004	0.006	0.003
Sulphur	0.014	0.015	0.019	0.017	0.018	0.014
Phosphorus	0.002	0.001	0.005	0.004	0.003	0.003
Carbon	0.009	0.011	0.015	0.02	0.016	0.008
Manganese	0.012	0.015	Trace	0.02	0.015	0.025
Oxygen	0.024	0.020	0.021	0.016	0.022	0.019
Copper	0.006	0.007	0.005	0.008	0.004	0.002
Aluminum	0.005	0.011	0.012	0.010	0.013	Trace
Nitrogen	0.006	0.004	0.005	0.003	0.005	0.007
Hydrogen						

In Table I are a few analyses of normal ingot iron as it is being produced today, all of which serve to show the possibilities of this process in the production of pure iron. The analyses are extremely accurate, the carbon and the oxygen being determined by combustion with the special care necessary in dealing with the determination of such minute quantities.

In tensile strength it closely approximates high-grade well-rolled charcoal iron, and in elongation and reduction of area it not only equals but in many cases surpasses the finest soft steel. It is a well-known fact that iron is usually inferior to steel in elongation and reduction of area. Owing to its more steel-like qualities in this respect, therefore, pure open-hearth iron becomes available for deep drawing and stamping purposes where very often ordinary wrought iron is unavailable and steel must usually be employed.

A recent U. S. Consular report on the radium market states a radium bank was established in Paris a short time ago, which in 1910 disposed of 1.92 grams of radium of highest activity at \$80,000 per gram. Of that quantity \$15,000 worth has been acquired for industrial purposes and \$139,000 worth for use in therapeutics. According to the report radium is found in Luntwengule, Morogoro district, East Africa; Evje N. v. Cristiansund, Norway; Cornwall and South Devon; Joachimsthal in Bohemia; Gilpin County in Colorado; and Kolmlagerstaetten in Sweden. Of all uranium mines only the one at Joachimsthal, which is working regularly, may be depended upon; there the two mines to be taken into account have an estimated yearly output of 16 to 20 tons of uranium pitch ore, or pitchblende, containing 55 per cent. U_3O_8 . Radium-free uranium color is obtained from this ore, which increases the radium content of the residue. Six tons is the annual production of residue, from which it is reasonable to expect a production of 1.8 grams of radium salt of the highest activity. The yield in Austria up to the present—3.5 grams—is not derived from the regular production from pitchblende, but from numerous old stocks, which are probably exhausted now. The yearly yield of 1.8 grams of radium salt of the highest activity may, according to present prices, be valued at \$144,000.

STEEL CASTINGS FROM ELECTRIC FURNACE.*

By C. H. VOM BAUR.†

The electric induction furnace for making steel has been in regular commercial operation in Europe for more than 10 years, and more than 30 of these furnaces are in regular commercial operation there. It has now been brought to the United States in its improved form, which permits sizes of 8 to 16 tons' capacity, and 30-ton furnaces have been projected. These electric furnaces are of the Röchling-Rodenhäuser induction and resistance type, operating on single, two or three-phase alternating current, utilizing any convenient voltage of commercial circuits; 25-cycle is preferred, but 60-cycle current can be used in the smaller sizes.

The furnace action is similar to any ordinary transformer. The primary coil, receiving the incoming current, is subdivided into series of steps, usually five, which, by the aid of a switch, regulate the current and consequently the heat in the metal bath. The secondary current is mainly (70 per cent) induced directly in the bath, which is really one short-circuited turn of the secondary winding of the transformer. The remaining energy, 30 per cent, is induced in a large copper bar secondary winding, wound directly over the primary winding, which latter receives the electric current from the alternator. This auxiliary secondary current is carried to two steel pole plates, set beneath the lining, one at each end of the bath. The magnesite lining covering these plates is the hearth of the furnace and becomes the conductor of electricity when red hot, acting similarly to the filament of a Nernst lamp in this respect.

The metal can be kept at any practicable temperature between a dull red and, say, 4700° F. (which is the limiting temperature of the magnesite lining). As there is no involuntary change in the current, and no violent fluctuations take place, there is no strain placed on the prime mover. The furnace runs quietly and there are no carbon particles blowing through the air. The amount of steel in the furnace being known, the heat can, with a little practice, be accurately determined by observing the ammeter reading. No foreign ingredients can inadvertently get into the metal bath, like sulphur, phosphorus, carbon or substances from any flame, electrodes or lining, which fact makes the furnace most gratifying to operate.

Chemical composition and temperature can be regulated independently of each other. The composition of the metal can easily be changed and the temperature held at the desired point. The metal can be poured at the temperature to suit the conditions; for large castings, into crane ladles; for small castings, into bull ladles or small hand ladles. The furnace,

*From a paper read before the recent meeting of the American Foundrymen's Association.

†American Electric Furnace Co., New York.

being of the tilting variety, facilitates the pouring and the removing of the slag by the rear door.

When charging cold materials for making steel castings, the cheapest steel scrap can be used to advantage and refined to the desired degree. The following makes a cheap and effective mixture:

COST OF ELECTRIC STEEL PER TON, MATERIAL CHARGED COLD	
$\frac{1}{4}$ Bundled scrap at \$11.25 per ton	\$ 2.81
$\frac{1}{4}$ Machine shop and heavy turnings at \$9.75 per ton	2.31
$\frac{1}{2}$ Old steel rails at \$13.75	6.88
Total	\$12.00
Oxidation loss, 3 per cent.	.60
Total	\$12.60

Electric current is being produced today for 0.6 cent per kilowatt hour with internal combustion engines, running on blast furnace, producer, natural gas or crude oil. The electric induction furnace provides a high load factor, operating, as it should, at full load for the greater part of the day. This continued operation insures this low cost for electricity.

The conversion cost per ton is, therefore, as follows (2-ton furnace with 280 to 300 kw., operated at Völklingen):

	Per long ton.
700 kw.-hr. for melting at 0.6 ct. per kw.-hr.	\$ 4.20
200 kw.-hr. for refining at 0.6 ct. per kw.-hr.	1.20
	\$ 5.40
Fluxes, etc.—roll scale, 22 lbs., lime, 77 lbs., fluor-spar 11 lbs., sand 20 lbs., ferro-manganese 8.8 lbs.	.44
Loss of fluxes owing to $\frac{1}{4}$ of all metal remaining in the hearth	.16
Labor, American equivalent, 2 to 3 men	1.50
Tools, repairs and lining	.67
Depreciation 10 per cent, interest 5 per cent on \$11,300—300 days, 6 tons a day*= $\$1,695 \div 1,800$ tons.	.94
Auxiliary apparatus (cooling air for transformer)	.04
Total	\$ 9.15

Adding this, we get:

Raw material	\$12.60
Conversion cost	9.15

Cost of one ton electric steel ready to pour.....\$21.75

To this cost must be added a slight license fee per ton, depending on the output. The time of heat is about 4 to 4½ hours.

When charging hot metal for refining or merely for thorough deoxidation and segregation, it may come directly from the blast furnace, from a mixer, a cupola or some other furnace. At Dommeldingen very impure pig iron was charged into an induction furnace of the Röchling-Rodenhauser type and refined as shown in Table I. Such impure hot metal is rarely charged in the electric furnace, except under unusual conditions, as it can be partly refined by a gas-fired furnace cheaper than in any electric furnace.

TABLE I.—RESULTS OF REFINING PIG IRON IN ELECTRIC FURNACE

	Analysis of Charge Per Cent.	Analysis of Cast. Per Cent.
Carbon	4.0	0.5
Phosphorus	1.8	0.025
Sulphur	0.2	0.03
Manganese		0.76
Silicon	1.00	0.056

Duration of conversion, 5 hours.

*12 hour day.

The cost of refining hot metal taken from the mixer at Dommeldingen, allowing for American conditions, is as follows for a 5-ton furnace:

Raw material	\$12.00
Oxidation loss 3 per cent.	.36
	\$12.36
Current—280 kw.-hr. at 0.6 ct.	\$ 1.68
Fluxes, etc.	.60
Labor	.50
Tools, repairs and lining	.64
Depreciation 10 per cent, interest 5 per cent on \$17,000—300 days at 40 tons a day*= $\$2,550 \div 12,000$ tons.	.22
Auxiliary apparatus	.06
Total	\$16.06
Cost of preliminary refining, about	3.00

Total cost of one ton of electric steel ready to pour, \$19.06
Time of each heat about 8½ hours.

The estimated cost of refining hot metal melted in the cupola, and consisting mainly of steel scrap, having about 2 per cent carbon in the resultant mixture, is as follows:

Raw material	\$14.00
Total oxidation loss 8 per cent.	1.12
	\$15.12
Conversion cost similar to the above.	1.00
Cost of preliminary melt in cupola, about	3.00

Cost of one ton of electric steel ready to pour.....\$23.02
Time of each heat about 3½ hr.

Under ordinary conditions there is no very great difference in the total cost of hot metal ready to pour

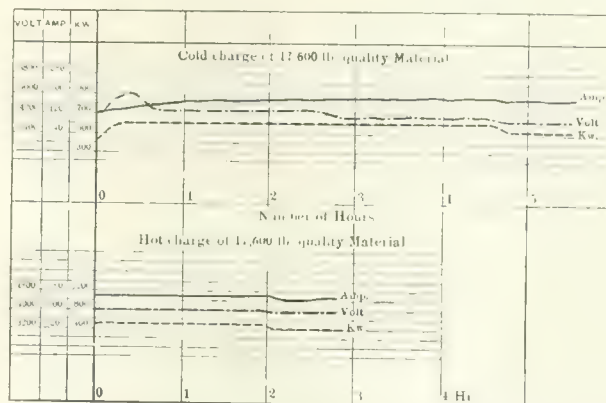


Diagram Showing Power Consumption in Röchling Rodenhauser Electric Furnace.

into the ladle, when charging either hot or cold material. The discrepancy is in the output. With a 5-ton furnace it is as follows:

	Per Month Single Turn	Per Month Double Turn
Cold charging	275	170
Hot charging	500	1,000

The electric current, and all the machinery it entails, does not enter largely in the total cost of a casting. It is usually not over $\frac{1}{4}$ cent a pound, or as little as 5 per cent or even 2 per cent of the selling cost of the finished product. It follows that steel can be melted in the electric furnace for \$4.20 in a 2-ton furnace, and for about \$3.50 in an 8-ton furnace with electricity at 0.6 per cent kw.-hr. At the same time, though, the output is decreased by about one-half, as

*24-hour day.

indicated above, for two reasons: 1. It takes longer to treat cold metal than hot. 2. We only pour three-quarters of the capacity of the furnace when charging cold material, instead of the total contents of the furnace when treating hot material. When charging cold metal, a little of the hot charge remains in the furnace to complete the electrical circuit. Cold scrap being in many pieces, does not meet the conditions, as the voltage in the bath, being as low as 2 or 3 volts, is not sufficient to overcome the contact resistance between these many pieces.

The metallurgical course of the process is very similar to that employed during the refining period in a basic open-hearth furnace. The limestone and roll scale are charged and the bath is then refined until the analysis shows that no phosphorus remains, or the process may be stopped before this, thus retaining some phosphorus and cheapening the process. This refining usually lasts an hour or so, varying somewhat with the initial percentage of phosphorus present. During this dephosphorizing period the carbon is greatly reduced and the silicon entirely or almost entirely eliminated. This first slag is then thoroughly removed by tilting the furnace backward at a slight angle and rabbling off the slag. Then, according whether mild or hard steel is required, a quantity of carbon is added to meet the requirements. At this time sufficient ferro-manganese is charged to meet the specifications. The desulphurizing slag is then added. As soon as this is melted, the bath and slag are deoxidized, the slag becomes white, and, owing to the now increased temperature, absorbs the departing sulphur rapidly, the refining being dependent upon the temperature at the point of contact between the metal and the slag. The gases are now expelled and likewise the small particles of slag which are brought about by deoxidation; in the same manner the sulphides are absorbed by the white lime slag while a small part of the sulphides volatilizes; otherwise the composition of the bath remains the same. Carbon is then analyzed for and the necessary modifications made.

In the case of alloy steels, the alloys are only added after the deoxidation period, so that any loss which might be caused through the formation of slags is avoided. This deoxidation period usually lasts an hour. It is this deoxidation period, so thorough and effective, which has no parallel in any gas-fired furnace or converter. The sulphur has meanwhile almost entirely disappeared, getting as low as 0.01 per cent to 0.005 per cent. After a last analysis shows that the deoxidation of the metal is complete, the furnace is tapped without any additions of any kind. Castings of any intricacy from $\frac{1}{2}$ lb. to the capacity of one or more furnaces are regularly made. In one of the works of Luxemburg, from which the following figures are taken, four grades of steel for castings have a call:

FOUR GRADES OF STEEL MADE AT LUXEMBURG.

Grade.	Tensile Strength.	Elongation
	Lbs. per sq. in.	Per Cent.
a.....	56,900 to 64,000.....	25
b.....	64,000 to 71,000.....	20
c.....	71,000 to 85,400.....	15 to 18
d.....	85,400 to 99,600.....	8 to 10

For the electrical industry, a particularly soft steel of the following analysis has a market: C, 0.05 to 0.06; Si, traces; Mn, 0.20; P, 0.005; S, 0.003.

Three pieces of equal weight, brightly polished, one of electric steel, one of open-hearth steel and one of Bessemer steel, of similar chemical analysis, were placed in an acid solution. Electric steel may lay claim to incontestable preference wherever materials have to be furnished to meet the strictest specifications in respect to resistance to acids and weather. It showed much less loss than the other kinds of steel.

COMMERCIAL POSITION OF GLYCERINE.

The following abstract of an article from the London Times should be of interest, particularly as it explains the reasons for the present increased price of glycerine.

Glycerine is a by-product in soap manufacture; when animal and vegetable oils are treated with alkali, soap and glycerine are formed. Except for its use in pharmaceutical preparations, glycerine was of small commercial value until nitro-glycerine was invented, and nowadays owing to the restricted circle of explosive makers little is heard of it as an ordinary market commodity. The "spent lye" left behind in the manufacture of the best soaps contains about 9 per cent of glycerine. The relative amount of glycerine and soap produced by the reaction varies widely according to the nature of the raw material and the kind of soap produced. Most of the soap makers boil the lye down to produce a crude glycerine containing 80 per cent, though some of the producers of poor quality soap throw it away. Only a few of the large houses make a refined product; the others either sell the crude to the bigger firms, to the limited number of explosive manufacturers who undertake refining. Nobels being the chief, or to merchants who buy for America. The British Government cordite works do not refine, but buy mostly from Nobels. At the beginning of 1907, crude glycerine was worth £28, 10s per ton, and refined £50 per ton. By the end of the year crude had advanced to £68 and refined to £105 per ton. The cause is due chiefly to the advance in the price of oils and fats. The soap maker has found it impossible, or at least injudicious, to recoup himself by increasing the price of soap, owing to its being a national commodity necessary to the poorest classes, so he has had to turn his attention to the other outlet of his products. It is interesting to note that the chief reason for the advance in the price of fats and oils is their increasing use in the manufacture of margarine. Some of the largest soap makers have found it necessary to protect themselves by becoming their own producers of vegetable oils.



DETECTION AND DETERMINATION OF SMALL QUANTITIES OF ETHYL AND METHYL ALCOHOL AND OF FORMIC ACID.

The detection and determination of small percentages of ethyl and methyl alcohol and of formic acid are often of considerable importance in food chemistry. Certain methods which appear to be of special value for this purpose are presented in a recently issued circular of the Bureau of Chemistry, U. S. Department of Agriculture. This circular, which is reprinted below in full, was prepared by Mr. Raymond F. Bacon, Assistant Chemist, Division of Foods, Bureau of Chemistry:

ETHYL ALCOHOL.

Ethyl alcohol, formed as a result of the action of the yeasts on sugars, is commonly found as a product of spontaneous fermentative changes in food products. Its presence in sufficient quantities in a fruit, a vegetable, or other food product, indicates decomposition at some time in the history of that substance. The mere detection of this alcohol, however, does not give much information as to the soundness of a food product, since traces of ethyl alcohol alone might not indicate a sufficient degree of chemical change in the fruit or vegetable to render it decomposed in the ordinary sense of the word. Further, mere traces of alcohol might have their origin in esters which give the aroma and taste to the fruit, and not be the result of a fermentation of the sugars present. It also appears to be a debatable proposition whether alcohol is not formed in saccharine solutions by many types of plants when kept under anaerobic conditions.¹

It is evident, therefore, that not only the detection, but also the quantitative determination of alcohol is necessary to prove decomposition of the type represented by alcoholic fermentation. The amount of alcohol which may be present in such a decomposed food will probably vary from 0.1 to 1 per cent. An amount less than 0.1 per cent would, in the writer's opinion, call for further investigation to determine its significance; an amount greater than 1 per cent would probably represent a degree of decomposition such as to render the product of such an appearance as to be unsalable. The method used to estimate amounts of alcohol within these limits must be of such a nature as to leave no possible doubt that the substance determined is ethyl alcohol. Most of the present methods

for detecting or determining small quantities of ethyl alcohol are weak at this point, being based on the quantity of potassium bichromate or other oxidizing agents which the alcohol will reduce. Such are the methods proposed by Bourcart², Nicloux and Baudreux³, Benedict and Norris⁴, and others. It is evident from a careful study of these methods that slight changes in temperature or concentration materially affect the accuracy of the results; indeed, Pozzi-Escot⁵ claims that even when uniform conditions are rigidly maintained it is impossible to obtain accurate results for alcohol determinations by methods involving oxidation with potassium bichromate and sulphuric acid.

As very many substances reduce potassium bichromate, it is evident that in determination based on that process there is no surety whatever that the substance so determined is ethyl alcohol. The method of Verley and Bölsing⁶, in which the acetic ester is prepared by the action on alcohols of acetic anhydride in the presence of pyridin, is not accurate enough to be applied to very small quantities of alcohol, and it is evident that nearly any alcohol will give the same reaction.

As none of the proposed methods for determining alcohol when present in small amounts answers the requirements for accuracy and reliability of the work under the food and drugs act, studies to improve them were begun. The possibility of concentration of the alcohol from solutions containing only a small percentage of this substance was first considered. Nicloux and Baudreux⁷ found that in distilling 600 c.c. solutions containing 0.2, 0.1, and 0.033 per cent of alcohol, respectively, from a liter distilling flask, there was obtained in the first 30 c.c. of distillate 50, 52, and 55 per cent of the total alcohol present. In repeating this work, using a Glinsky distilling head, the writer found that by these distillations it is only possible to obtain from 50 to 60 per cent of the total alcohol, when attempting to concentrate 0.1 per cent solutions into such a volume as to obtain solutions containing from 2 to 4 per cent of alcohol.

It was also found that if a considerable quantity of calcium or sodium chlorid was dissolved in the dilute alcohol solution before distilling it, using an ordinary distilling flask, from 94 to 100 per cent of the total alcohol could be easily concentrated in a relatively small volume of distillate, as is shown from the fol-

¹Stollas, Ber. 1. Chem. Ges., 1903, 36:622. Chem. Centrbl., 1904, 1:961. Takahashi, Chem. Centrbl., 1902, 2:1330. Conheim, Zts. physiol. Chem., 1903, 39:348. Battelli, Compt. rend., 1902, 137:1029. Margnan, Compt. rend., 1905, 140:1124.

²Zts. anal. Chem., 1890, 29:609.

³Bull. soc. chim., 1891, (3) 17:424.

⁴J. Amer. Chem. Soc., 1898, 20:293.

⁵Ann. chim. anal., appl., 1907:112.

⁶Ber. d. chem. Ges., 1901, 34:3351.

⁷Loc. cit.

lowing experiments selected from a number made to test this point:

Experiment 1.—To 500 c.c. of water containing 1 gram of absolute alcohol (0.2 per cent by weight) there were added 200 grams of calcium chlorid and this solution was distilled from a 1 liter distilling flask, 150 c.c. of distillate being collected. To this distillate there were added 50 grams of calcium chlorid and a 50 c.c. solution was distilled from it. This distillate gave at 17.5° C., with the Zeiss immersion refractometer a reading of 18.2, equivalent to 2 per cent of alcohol, a 100 per cent recovery.

Experiment 2.—Five hundred cubic centimeters of 0.19 per cent alcohol were treated as in experiment 1, save that of the first distillate but 100 c.c. were collected. The refractometer reading at 17.5° C. was 17.9, equivalent to 1.84 per cent alcohol, a 96.8 per cent recovery.

Experiment 3.—This experiment was conducted exactly as in the case of No. 2, except that instead of using calcium chlorid, the solutions were about three-fourths saturated with common salt. The refractometer reading was 17.9 at 17.5° C., equivalent to 1.84 per cent alcohol, a 96.8 per cent recovery.

Experiment 4.—From the distillates of experiments 2 and 3 (100 c.c. of 1.84 per cent alcohol) after three-quarters saturation with salt, there were distilled 50 c.c. The refractometer reading at 17.5° C. was 21.0, equivalent to 3.71 per cent alcohol (theory 3.68 per cent), a 100.8 per cent recovery.

Experiment 5.—Five hundred cubic centimeters of 1.6 per cent alcohol were treated as in experiment 3. The refractometer reading at 17.5° C. was 42.7, equivalent to 15.02 per cent alcohol, a 94 per cent recovery.

Experiment 6.—One thousand cubic centimeters of 0.1 per cent alcohol were three-quarters saturated with salt, 150 c.c. were distilled off, again three-fourths saturated with salt, and from this solution 25 c.c. were distilled which at 17.5° C. had a refractometer reading of 21.3, equivalent to 3.84 per cent alcohol; a 96 per cent recovery.

These experiments prove that very small percentages of alcohol may easily be concentrated from salt solutions to a strength that can be used for the accurate determination of alcohol by known methods. The method here proposed for detecting and determining small quantities of alcohol is as follows:

Using the methods outlined in the preceding experiments concentrate the sample until a solution containing from 2 to 4 per cent of alcohol is obtained. If acids are present in the original solution exactly neutralize before distilling, using a trace of solid phenolphthalein as an indicator. Treat the final distillate as follows: Determine its specific gravity and refractometer reading at a suitable temperature, obtaining from tables⁸ the corresponding percentages of alcohol. If the refractometer reading and the specific gravity

both correspond to the same percentage of alcohol, it is almost a certainty that it is ethyl alcohol that is present, and that the substance under examination contains the percentage of ethyl alcohol equivalent to these constants. To render this an absolute certainty, the ethyl ester of paranitrobenzoic acid is prepared from this distillate with paranitrobenzoyl chlorid by the Baumann-Schotten method⁹. To the solution containing the alcohol is added a slight excess of acid chlorid and an equivalent amount of sodium hydroxid. Shake until the odor of the acid chlorid disappears. This ester is a beautifully crystalline solid which may be readily identified by its melting point of 57° C.¹⁰ This ester may be weighed, and while the yield obtained is not quite quantitative (70 to 90 per cent of the theory when working with small quantities of alcohol), its weight serves as another check on the percentage of alcohol present in the solution. If paranitrobenzoyl chlorid is not available the benzoic acid ethyl ester may be prepared as just described, using benzoyl chlorid. The benzoic acid ethyl ester may be weighed (yield practically quantitative), and identified by its odor and its boiling point, 212° C., using for the latter purpose the convenient method for determining the boiling point of very small quantities of liquids recently proposed by Smith and Menzies.¹¹

As to the qualitative detection of ethyl alcohol, two tests used together, the formation of iodoform and the formation of ethyl benzoate would render the presence of ethyl alcohol probable, for the only benzoic ester whose odor is similar to that of the ethyl ester is the methyl ester, and methyl alcohol does not give the iodoform reaction.¹² A little careful consideration of the question, however, will make it evident that for an absolutely unassailable result, when ethyl alcohol is present in small amounts, some such method as that just outlined for the determination of this substance must be used. After the alcohol is concentrated as described, other confirmatory tests, such as the determination of the freezing point, the boiling point, the capillarity constant, the electrical conductivity, or the preparation of other esters, may be applied to the liquid.

It has been attempted to describe in a general way such tests as can be easily made and which prove absolutely that an aqueous liquid contains a definite quantity of ethyl alcohol. It is realized, however, that there are many mixtures from which the alcohol can not be separated by the general method here given, and such special cases must be treated individually. After the alcohol is separated by some method applicable to the mixture in hand the general methods herein described may be applied. They have been used in the laboratory of the Bureau of Chemistry by H. C. Gore in following the decomposition changes in bananas, with quite satisfactory results.

⁸Ber. d. chem. Ges., 1884, 17:2445; 19:3219.

⁹Ber. d. chem. Ges., 1905, 38:620.

¹⁰J. Amer. Chem. Soc., 1910, 32:897.

¹²The limit of these reactions is about 0.1 per cent of ethyl alcohol.

⁸Tables of Ackermann and Steinmann, Zts. gesam. Brauwesen, 1905, 28:259; Doroszewski and Sworzanczyk, J. Russ. Phys. Chem. Soc., 1907, 40:101-125.

METHYL ALCOHOL

A convenient and reliable method of detecting methyl alcohol is that of Mulliken and Scudder¹³, in which the alcohol is oxidized to formaldehyde by means of a spiral of oxidized copper wire. A careful study of this method showed that to obtain a positive test it is necessary to have a solution containing over 0.1 per cent of methyl alcohol. It seemed that greater delicacy would be obtained in such a test by conducting the oxidation with certain substances in solution instead of using the solid copper oxid in wire form as proposed by Mulliken¹⁴. A. Vorisek¹⁵ proposed such a method, but as he did not determine its delicacy and the apparatus used by him seemed cumbersome, the subject was further studied. The method proposed by the writer is as follows:

To 100 c.c. of aqueous methyl alcohol in a 200 c.c. distilling flask add from 5 to 8 grams of chromic acid; collect 10 c.c. of distillate and test for formaldehyde by the Leach or Hehner methods¹⁶. If other oxidizable substances, such as acids, sugars, starches, proteids, etc., are present the solution under examination is exactly neutralized with sodium hydroxid and one-third of it is distilled, the distillate being oxidized and redistilled as has just been described. The use of the relatively large quantity of chromic acid indicated is found to increase the delicacy of the reaction. By the proposed method it is very easy to detect one part of methyl alcohol in 100,000 parts of water, and by making one preliminary distillation to concentrate the methyl alcohol, collecting the first 100 c.c. from a liter and then proceeding as before described, a very strong test was obtained from a solution containing only one part of methyl alcohol per million.

When testing for methyl alcohol in the presence of strong ethyl alcohol it is advisable to dilute the alcohol with water to obtain approximately a 20 per cent solution, as otherwise the action of the chromic acid on the ethyl alcohol may become explosively violent. It is easy by this method to detect 0.1 per cent of methyl alcohol in 80 per cent alcohol, and by a little fractionation it was found possible to detect 0.01 per cent in 80 per cent ethyl alcohol. Potassium permanganate may be substituted for chromic acid in this oxidation, but the sensitiveness of the reaction for aqueous solutions when using this reagent is then only about 1 part in 4,000. For the determination of methyl alcohol the same principles and method given under ethyl alcohol apply.

FORMIC ACID.

The detection and determination of formic acid in food products is becoming of increasing importance, not only because of the proposed use of this acid as a preservative but also because it has been found to be one of the products of the fermentative action of bacteria and yeasts on carbohydrates. Fenton¹⁷ has shown

that carbonic and formic acids in aqueous solution may be reduced to formaldehyde by the action of magnesium. The reduction of formic acid has been studied in this laboratory to determine its sensitiveness, and certain other metals have been found to give the same reduction. The formation of formaldehyde from carbon dioxide by these metals is of considerable theoretical importance; but, in the practical method of testing for formic acid, as proposed here, there is no danger of confusing this acid with carbonic acid, as to obtain detectable quantities of formaldehyde from carbon dioxide it is necessary to cause this gas to act on the metal for several hours.

If a suspected solution is acidified with phosphoric acid and then distilled the amount of carbon dioxide found in the distillate is not sufficient in amount to give detectable quantities of formaldehyde when reduced by metals for any reasonable time. In the following table the action of various metals on 1 per cent and 0.1 per cent formic acid solutions is given, the reduction to formaldehyde being indicated as positive and negative:

THE REDUCTION OF FORMIC ACID BY THE ACTION OF VARIOUS METALS.

Metals.	1 per cent formic acid.	0.1 per cent formic acid.
Magnesium filings.....	Positive.	Positive.
Calcium metal.....	Do.	Do.
Coppered metal filings.....	Do.	Do.
Coppered zinc dust.....	Do.	Do.
Zinc amalgam.....	Negative.	Negative.
Coppered aluminum.....	Do.	Negative.
1 per cent sodium amalgam with magnesium chloride solution.....	Do.	Do.
Aluminum.....	Do.	Do.
Aluminum amalgam.....	Faintly positive.	Do.
Zinc dust.....	Do.	Do.
Zinc filings.....	Do.	Do.

As magnesium filings seem to give the best results the reaction is carried out as follows:

Strongly acidify the solution with phosphoric acid and distill about one-third of it. To the distillate add dilute sulphuric acid and magnesium filings in sufficient quantities to cause a vigorous but not a violent evolution of hydrogen. (In case quite a large quantity of acid is present in the distillate it is not necessary to add any sulphuric acid.) If the amount of formic acid is small (about 0.1 per cent) continue the action for one hour. If larger quantities are present, the test for formaldehyde will usually be obtained in from two to five minutes. Of course the absence of formaldehyde in the original solution must be determined.

The common methods for the determination of formic acid are based on the following general reactions: (1) The reduction of a mercuric salt to a mercurous salt, the insoluble mercurous salt being weighed; (2) the reduction of potassium permanganate in alkaline solution; (3) the action of concentrated sulphuric acid, the evolved carbon monoxide being measured. With regard to methods 1 and 2, aldehydes, certain alcohols, and many other volatile substances which may be present in fruits and vegetables effect the same reduction as formic acid. With

¹³J. Amer. Chem. Soc., 1899, 21:266.

¹⁴Loc. cit.

¹⁵J. Soc. Chem. Ind., 1909, 28:823.

¹⁶U. S. Dept. Agr., Bureau of Chemistry Bul. 107, Revised, p.

185; Analyst, 1895, 20:155.

¹⁷J. Chem. Soc. (Lond.), 1907, 91:687.

regard to method 3, a very large number of organic acids evolve carbon monoxid when treated with concentrated sulphuric acid.

While platinic salts are readily reduced to metal by many aldehydes and other reducing agents in alkaline solution, it was found that of a great number of substances tried only formic acid accomplished this reduction in neutral or weak acid solutions. The following method is therefore proposed: Distill the solution containing the formic acid with a small quantity of phosphoric acid until the distillate is no longer acid. If the volume of this distillate is too large to be conveniently handled, neutralize it with sodium hydroxid and evaporate to a convenient volume. Add an excess of platinic chlorid and sufficient acetic acid to make the solution strongly acid (usually about 1 or 2 c.c. of glacial acetic acid for less than 1 gram of formic acid), and boil the solution for one hour, using a reflux condenser. Collect the reduced platinum in the usual manner and weigh. The weight of the platinum multiplied by 0.472 equals the formic acid present.

The following are examples of results obtained by this method:

(1) A 100 c.c. solution containing 0.0782 gram of formic acid; platinum reduced, 0.160 gram = formic acid 0.0755 gram; 97 per cent of theoretical amount.

(2) A 100 c.c. solution containing 0.243 gram of formic acid; platinum reduced, 0.504 gram = 0.238 gram of formic acid, 98 per cent recovery.

(3) A 100 c.c. solution containing 0.243 gram of formic acid; platinum reduced, 0.517 gram = 0.244 gram of formic acid, 100.4 per cent recovery.

(4) A 100 c.c. solution containing 0.1215 gram of formic acid; platinum reduced, 0.255 gram = 0.120 gram of formic acid, 98 per cent recovery.

The electric conductivity of formic acid is so much greater than that of other similar organic acids that it seems possible to base a method for its determination on that property. A further study of this problem is in progress.

The shipments from the Cobalt field in Canada for the first half of 1911 were considerably lower than for the same period of 1910, when the total shipments were 14,437.97 tons; this year the 6 months' record is 12,333.33 tons. This falling off, however, may easily be attributed to the smaller production due to lack of power earlier in the year, owing to trouble with floating ice. Another cause for the decrease is that there are 15 concentrators now in operation in Cobalt and the concentrates are shipped from these plants instead of the crude ore.

The glucose and starch industry is the subject of a bulletin just issued by the Census Bureau at Washington. It shows that there were 675,938,000 lbs. of starch of all kinds, valued at \$17,360,000, manufactured in the United States in 1909.

THE TESTING OF CREOSOTE.*

By C. EDWARD SAGE, F. I. C.

The analysis of impure substances for commercial purposes must always yield unsatisfactory results if no uniform method of testing is adopted. With creosote analysis it is continually found that different chemists report results which are not in agreement, and such variations are nearly always explained when the methods of analysis are investigated.

In the United States the Department of Agriculture has done a great deal of work with the object of finding satisfactory and uniform tests for a product which is of so great importance in several industries, but in Great Britain each manufacturer or works chemist has his own particular method of testing, and every buyer, contractor or engineer has a specification which meets his own requirements.

The creosote produced in different parts of Great Britain varies in composition for three reasons—(1) the methods of gas production or coal distillation differ; (2) the nature of the coal employed varies in each district; (3) the method of tar distillation adopted may vary in many details.

Great Britain is the chief producer and shipper of creosote, and it seems anomalous that we have no universal standard methods of analysis such as are in use in so many other chemical industries. We work on uniform lines in the testing of chemical manures, cement, tanning materials, and some oils, and at the present time there is an opportunity for tar chemists in Great Britain to unify the methods of analysis of such an important product as coal tar creosote.

The very name "creosote" needs careful definition, for the word may be applied to a product made from gas tar, wood tar, oil gas tar, producer gas tar, and coke oven tar, and although many years ago creosote was always a wood tar product, now the name "creosote" outside of medical circles, always means the coal tar product unless it is qualified in some way.

The complete analysis and series of tests for any one sample of creosote may not all be required for the information of one of the parties who may handle or use it, but at some time the results of a complete examination will be necessary. The chief considerations in any examination are the physical conditions of the oil, the approximate composition, and the freedom from admixture with other oil. The information is needed for the guidance of the tar distiller, the seller, the buyer, the shipping or transporting company, the insurance company, the timber creosoter, the railway engineer, or the manufacturer of disinfectants or other products. The analysis has to settle the questions whether the oil is of suitable quality or up to standard requirements and how best it can be handled for transport, and no information for the future guidance of timber preservers is available unless the

*Paper read before the Society of Chemical Industry and printed in the Journal of the Society.

relative proportions of the constituents of the creosote are stated.

I venture therefore to offer the following suggestions as the basis for a discussion of the most suitable standard to adopt.

A specification for creosote may be drawn up to suit the requirements of any particular use to which the oil is to be put, but it would be useless to try to make one uniform specification for all creosotes. The oil in some parts of Great Britain differs entirely from London creosote, so I should only attempt to standardize methods of testing and not the product, which is necessarily a mixture of impure products.

Any specification for creosote should be prefaced by the words, "The creosote must be the product of the destructive distillation of bituminous coal and free from sulphur and free carbon.

Sampling.—Too much importance cannot be attached to this preliminary operation, for all the subsequent results depend upon its proper performance. To insure an approximately uniform sample the bulk must be liquefied, and the sample should then be drawn from the different parts of a tank. Not less than 6 half gallon quantities should be removed and bulked, and then at least three separate cans should be filled and sealed. Creosote from a number of barrels can only be sampled with an iron, and the difficulties of attaining uniformity are greatly increased by such a method.

Buyers of creosote in any quantity should specify the method of sampling and always keep more than one sealed sample in reserve, for the repeated warming of a can always causes some variation in its water contents.

PHYSICAL TESTS.

Specific Gravity.—To insure uniformity this can only be taken at some higher temperature than the mean average.

Samples are often quite solid at 15° C., and I would suggest 60° C. as the most suitable. The method of ascertaining the relative density should be by weighing 100 c. c. and comparing with water at 60° C., or else by means of a Westphal balance.

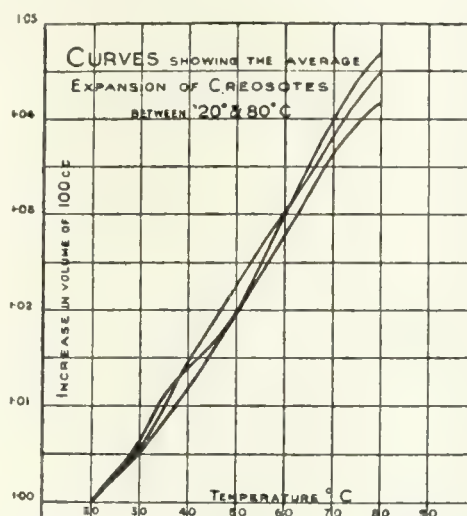
Expansion Coefficient.—100 c. c. of the oil should be measured into a flask marked at 100 c. c. on the neck and graduated in one-tenths of a c. c. up to 106 c. c. The volume should be first adjusted to 100 c. c. at 40° C. and the flask then warmed in a water bath to 80° C., and the volume at that temperature measured and recorded.

The factor most commonly employed is an expansion of 1 per cent for a rise of every 22½° F., but this is not true for all creosotes or for high temperatures as well as low ones.

I have plotted the curves obtained when heating samples from 20° C. to 80° C. and the range falls within very narrow limits, and my results show that the increase in volume is 1 per cent for each 13.3° C., or approximately 1 per cent for every 24° F.

The American Railway Engineering and Maintenance of Way Association, in a report on wood preservation, has recommended the measurement of creosote by weight and in dealing with barrels or small lots this is undoubtedly the more accurate method, but in dealing with tank cars the measurement of dimensions is so easy that a volume statement seems more convenient than one of weight. In either case, however, the specific gravity enables us easily to convert one into the other system.

Flash Point.—Some specifications mention the Abel test, but I would point out the unsuitability of the instrument for any temperatures above 150° F. Its water jacket prevents its use above 100° C., and for that reason the Pensky-Marten apparatus is more suited to creosote. In no test for creosote is the importance of a fresh representative sample more im-



portant. A sample which has been heated up several times may give entirely different results to that obtained when it is first opened. The information is usually required by insurance companies, and it is highly important that the test be made uniformly. The oils which I have tested flashed between 186° C. and 190° C.

Fluidity or Condition.—For shipping requirements it is necessary to know the condition of the creosote. It is not easy or in some cases possible to determine the lowest temperature at which a sample remains fluid; the creosote may not deposit any "salts" for days and then begin to solidify, or it may solidify almost as soon as it is cooled.

The requirements of the case are usually met by determining whether the whole of the oil is fluid at any selected temperature, say, for instance, 40° C., and by keeping 100 c. c. of the oil at 15° C. for 6 hours and stirring occasionally after adding a crystal of naphthalene. The result can then be specified as "fluid," "quite solid," "some salts deposited" or any other condition which may present itself.

Solubility in Benzol.—A limit should be set to the amount of insoluble matter. The test is easily made with creosote containing no water, but, in the pres-

ence of water, filtration is impeded. 100 grms. should be filtered, while hot, through a tarred filter paper and the filter afterwards washed with benzol in a Soxhlet tube until the benzol runs away clean. The filter is then dried and weighed. Many samples contain only traces of insoluble carbon, but in no case should more than 0.25 per cent be passed without comment.

Viscosity.—I have obtained no useful information from the determination of this factor, and the results are entirely dependent on the clean condition of the oil and its freedom from drops of water. When timber with a fine grain is to be creosoted, the presence of traces of insoluble carbon make far more difference than any slight difference in viscosity possibly could.

Distillation.—No test of creosote is complete without a fractional distillation of the oil, and no item of a test gives rise to more differences between buyers and sellers, because of the lack of uniformity in the method of distillation. In 1902 and during subsequent years I was engaged with Mr. T. A. Ellwood in the fractionation of both coal tar and wood tar creosote, for purposes which required far greater precision than the commercial tests. At that time we found a long necked retort the best vessel to use for coal tar creosote in the preliminary distillation. For the distillation of wood tar creosote I prefer a Wurtz flask, and for blast furnace creosote the same flask with a side tube low down in the neck.

For general purposes to meet the requirements of all grades of coal tar creosote, a retort is the most suitable vessel for the distillation, but to obtain uniform results the retorts must be of uniform size and the position of the thermometer in the liquid must be specified.

The retorts must be of hard glass and should have a capacity of 8 fluid ounces (up to the bend), and the shape should be such as to allow a free escape of the vapors. Messrs. J. J. Griffin & Sons, Ltd., make these retorts in Rhenish glass, which stand the high temperatures required. Dean and Bateman¹ suggest the use of a distilling flask of the Hempel type, with a long neck and a column of beads, for fractionating the creosote, and in an earlier paper a three bulb distilling flask such as is used in England for the distillation of essential oils, and known as a Schimmel three bulb flask. The objection to both of these is better observed in practice than it can be explained, but the chief fault is that with oils containing a very high proportion of naphthalene and varying quantities of water, the distillation must be pushed very rapidly to prevent the water carrying over naphthalene in the first fractions and to prevent the naphthalene from solidifying in the side tube.

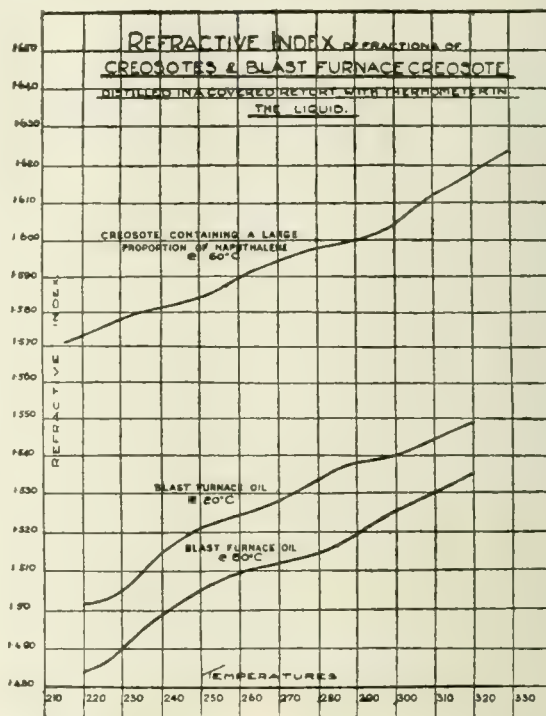
Shrenk, Fulks and Kanmer² advise the use of an 8 ounce retort and cover it up completely with as-

bestos card. They, however, put the thermometer above the liquid in the retort.

My own experience is that results agree to within 2 per cent when the thermometer is placed in the liquid, and the results are not always so reliable with the thermometer out of the liquid, because of variations in the vapor cooled by draughts.

There is no direct relationship between the figures obtained with any sample in the two ways, with the thermometer inside and outside of the boiling liquid. One result cannot be compared with another, for no two samples agree in any particular difference between the two methods.

I therefore favor the use of an asbestos covered retort (although a tin can with its side cut out is just



as accurate) and the position of the bulb of the thermometer is to be in the liquid less than half an inch from the bottom of the retort.

To obviate corrections of temperature it is well for all workers to use thermometers of the same pattern, then we may agree to eliminate the correction needed for the emergent column or we can all use the same factor. The thermometer should have a range of 190° C. to 360° C.

The fractionation of creosote is made with a view of ascertaining the relative proportions of water, light oils, creosote oils, heavy oils and undistilled residue. The presence of any considerable quantity of water causes irregular distillation of the early fractions, and for this reason it should be specified that if a creosote contains more than 1 per cent of water it should be first distilled to remove water and any oil which distills separated from the water and returned to the dry oil before proceeding with the distillation. The fractions collected in any particular test can be as conveniently measured as weighed, provided the meas-

¹U. S. Department of Agriculture: Forest Service Circular, No. 112.

²American Railway Engineering and Maintenance of Way Association Bulletin No. 65.

urement is made at a specified temperature. In order that all fractions shall be fluid the oil should be measured at 60° C. I have seen so many specifications for the distillation of creosote that I feel it would be useless to suggest one which would please everybody, but the most useful points are:

Distillate up to.....	210° C.
Distillate between.....	210°-235° C.
Distillate between.....	235°-270° C.
Distillate between.....	270°-315° C.
Residue not distilling at.....	315° C.

Water.—This is determined by measuring the volume in the first fraction distilled. Any difficulty in reading the dividing line between oil and water can be remedied by the addition of 5 c. cm. of solvent naphtha to the distillate.

Naphthalene.—This should be estimated in the fractions distilling below 270° C. and not in the entire oil, by filtering the cooled distillate through filter cloth and pressing between bibulous paper. By this means a fairly accurate result is easily obtained. Mann² has suggested a method of determination by means of what he designates the "latent heat point." His method is useful when examining creosote of fairly uniform quality, but an admixture of other oil would upset the curve which he has plotted.

Tar Acids.—These should be determined by shaking out with a 15 per cent caustic soda solution and subsequent acidification and measurement at 60° C. The creosote itself will not yield satisfactory results and it is usually sufficient to shake out the fractions distilling up to 315° C. Phenols are present in the higher boiling fractions, but they are frequently sticky semi-solid bodies and not easily measured.

Some very precise method of stating the amount of "tar acids" is needed, for the figures given are often very misleading.

Pyridine bases are a most important factor for consideration by manufacturers of sheep dips and disinfectants. I usually determine the bases in the fractions distilling below 315°.

Residue Undistilled.—This fraction of creosote may or may not have a particular interest to a purchaser. Many timber creosoting works ask for oil with a high proportion of high boiling fractions, and then it is of importance that the residue should be examined. In order that creosote may easily enter timber it is needful that it shall not contain any high proportion of pitchy residue, and so that softness of this residue may be assured it is often necessary to use a blend of creosotes.

Sulphonation.—In the United States petroleum fractions are sometimes found present in creosote; English creosote cannot naturally contain any high proportions and many contracts limit the sulphonation residue to 0.25 per cent of the creosote.

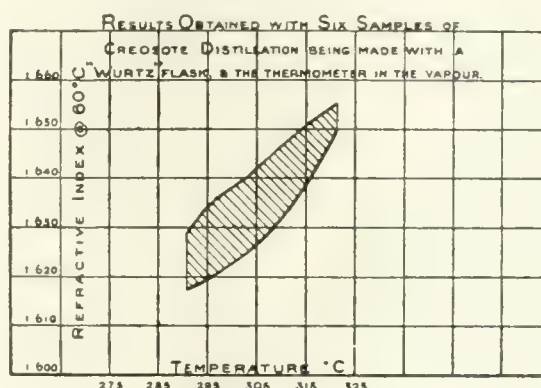
Paraffins exist naturally in the high boiling fractions of coal tar and of anthracene oil, but I have only found the following proportions in creosotes:

London Creosote	A 0.15 per cent paraffin
London Creosote	B 0.14 per cent paraffin
London Creosote	C 0.2 per cent paraffin
North of England Creosote	0.3 to 5.0 per cent
Blast Furnace Creosote	0.3 to 0.5 per cent
High-boiling Blast Furnace Creosote	4.6 per cent
High-boiling Special Creosote	6.0 per cent

The identification of paraffin is made easy by determination of the refractive index of the sulphonation residues, and the differentiation of petroleum distillates from blast furnace creosote and coal tar creosote is rendered possible also by measurement of the refractive index of the fractions of the oils.

In common with most oils, the paraffins have a refractive index verging on 1.45 at the lower limit and 1.5 at the higher one, whereas blast furnace creosote distillates obtained during testing have a range from 1.48 to 1.54, and coal tar distillates from 1.57 to 1.63 or even higher.

The method of distillation makes a great difference in the figure for refractive index at any particular



temperature, but I have included in the accompanying graphs the refractive index of the fractions of six samples of creosote distilled in a Wurtz flask with the bulb of the thermometer in the vapor and those of blast furnace creosote and coal tar creosote distilled in a retort with the thermometer bulb in the liquid. The refractive index of paraffins can so often be taken at 20° C. that I have included the figures for blast furnace distillates at that temperature.

I feel convinced that it is safe to recommend that all creosotes which are pure coal tar distillates should conform to the following requirements: That when distilled from an 8 ounce retort, covered during the distillation, with the bulb of the thermometer in the liquid, the fractions distilling above 300° C. should have a refractive index about 1.6.

Creosote is so largely used for the preparation of sheep dips and disinfectants that its germicidal properties are often of first importance to purchasers. The average germicidal value of London creosotes for *B. typhosus* is nearly twice that of crystallized carbolic acid, and the tar acids contained in creosote vary within wide limits. In the course of some hundreds of tests I have been led to form the opinion that it is often the impurities in a sample which give it its specific bactericidal properties, and not so much the percentage of "tar acids." As a fungicide it is diffi-

²Journal of the Society of Chemical Industry, 1910, 29, 732

cult to measure comparatively the value of creosote, but I am only anticipating matters slightly when I state that a biological test of creosote will be needed in the near future.

In conclusion I have to thank Mr. Wallace C. Nickels for much valuable information and to express my indebtedness to the governors of Battersea Polytechnic for many facilities which they have afforded me during the preparation of my paper.

DISCUSSION.

Mr. V. Blagden asked Mr. Sage to explain whether the last method which he described would detect the presence of blast furnace creosote mixed with coal tar creosote. That matter would interest the trade very materially, and if Mr. Sage had succeeded in discovering a reliable and easy test on that point they would all be deeply grateful to him.

Dr. J. Lewkowitsch said he was not quite sure whether the author put these methods forward as methods that ought to be adopted. He gathered Mr. Sage laid rather more stress upon the determination of naphthalene than one would as a rule. The determination would, he thought, be a faulty one, and would range within very wide limits; he would therefore have expected Mr. Sage would have laid more stress upon the determination of the tar acids, because, rightly or wrongly, some of the value of the oil was ascribed to them, and they could be fairly easily extracted. As they all knew emulsions were obtained by shaking out with caustic soda. The method adopted 30 or 40 years ago was to dilute with low boiling tar fractions and use caustic soda solutions of low specific gravity. Then a clear separation was obtained, all the tar acids were in the alkaline solutions, and no further difficulty was encountered. The pyridine bases would of course offer no difficulty. With regard to the refractive index, Mr. Blagden had already touched on the crucial point—how were they going to differentiate between mixtures? Too much reliance was still placed upon the refractometer or rather upon the deductions from its indications, which were accurate enough; but when mixtures were obtained, one was entirely at sea and did not know how to interpret the refractive readings. In his opinion it was not worth all the trouble spent upon it. Lastly he could not agree with the method of fixing the thermometer in the liquid and relying upon the boiling point thus obtained. The matter had been threshed out 20 or 30 years ago. The results obtained were hopelessly wrong, and in his opinion the best and only reliable way was to put the thermometer into the vapors as they left the vessel. By the employment of such a method he was afraid the discrepancies of which they heard were only too easily explained.

Mr. W. J. A. Butterfield expressed his agreement with what Dr. Lewkowitsch had said with regard to the position of the thermometer. In distilling these products he had frequently observed that a thermometer placed in the liquid would suddenly run up even

200° C., while a thermometer in the vapor would rise only some 20 or 30 degrees—that was with the higher fractions of such products. With regard to the distinction between blast furnace creosote and coal tar creosote he thought that a careful fractionation and a comparison of the specific gravities of the fractions was on the whole the best criterion of the mixture. Mr. Sage had spoken of the determination of specific gravity, and proposed to take it uniformly at 60° C. That depended upon the creosote. If the creosote were described as being free from solid crystals of naphthalene at a particular temperature, it seemed only reasonable to determine the specific gravity at that temperature. There was no object in determining the specific gravity at a much higher temperature than that at which the oil was fully liquid. The point named by the author was very convenient undoubtedly for most creosotes for pickling timber. But there were many uses for creosote oil in which it was requisite that it should be free from naphthalene crystals at a much lower temperature. He need only specify the use of the oil for washing gas for the extraction of naphthalene, and its use for thinning pitch for preparing pitch macadam for roads. A recent specification issued by a government department mentioned a temperature of 20° C. for the determination of the specific gravity, and it prescribed that the oil should be free from solid matter, and remain so for a definite period—in that case half an hour was named—at the particular temperature. He agreed fully with the author that it was highly desirable that there should be a certain degree of uniformity in the methods of testing creosote oil, but the points on which uniformity rested must depend upon the specification, and he thought where the specification, for instance, said that the oil should be free from solid matter at 20° C. it was absurd to take the specific gravity at 60° C. So much must depend, in the testing, on the conditions of the specification with which the oil had to comply.

The chairman said that in discussing the question he proposed more or less systematically to take the points as they had been introduced by the reader of the paper. In the first place, however, they would all agree that there were two broad classes of creosote. First, creosote which was used for the creosoting of wood, and, secondly, creosote which was intended either for such special purposes as had been alluded to by Mr. Butterfield or for the manufacture of sheep dips and disinfectants. He did not propose to say anything about the latter class. Such creosote stood alone, and the character should depend, as Mr. Butterfield had said, upon the particular specification to which it was supplied, and that again depended upon the use to which the creosote was to be put. But if he was right in concluding that the most important use of creosote was undoubtedly that of creosoting timber, then he thought they might largely confine their attention that evening to a consideration of what were the desirable features in creosote intended for that

purpose; and he was inclined to think the time had nearly come when they should take a step forward from the old position in which the matter had been very largely left up to the present time, and which was based on the specification originally drawn up by the late Dr. Tidy. It had been generally accepted that the creosote should contain not less than 25 per cent of matter non-volatile at 600° F., which is roughly the same temperature as the 300° C. taken by Mr. Sage. But he would ask, ought not the proportion of naphthalene always to be taken into account when one is considering the suitability of a particular sample? He would urge that if there were an appreciable amount of naphthalene that ought to be added to the amount of relatively solid matter, which is represented by the non-volatile at 316° C. or 600° F., and be viewed as the total solid matter. If this were accepted considerable importance then attached to the determination of the naphthalene. The creosoter aimed at saturating his wood as nearly as possible with something that would not wash out. It was an old theory, but he thought he might venture to say that it was an exploded idea, that the tar acids were the valuable constituents in creosote intended for application to wood. It had been shown, some 30 years ago, in a paper read before that society by Mr. Boulton, that within a comparatively brief time after creosoted wood had been embedded in the earth the whole of the tar acids had gone. But the theory upon which the appreciation of the old tar acids was based rested, he thought, upon a misconception. The idea, in Dr. Tidy's phrase, if he remembered rightly, was that the tar acids coagulated the albuminoids of the wood. He did not think the albuminoids had now quite the same importance from that point of view, because in the light of the biological knowledge of the present day, if you excluded micro-organisms, then protein matter was no more liable to decay than carbohydrate substances. He would lay it down that what was needed in a creosote was a maximum amount of solid matter, made up, in the first place, of the constituents non-volatile at 316° C. and of naphthalene. With these there was required just enough tar oil to act as a solvent so as to permit of the solid constituents being carried efficiently into the wood in the process of creosoting. If he was wrong on that point, he would like some practical man to correct him. If they agreed upon that point then the testing of creosote resolved itself into a determination of the two solids, as he had called them, with something like an accurate determination of the amount of water and the amount of what he might call tar oils, though of course he did not say that the determination of the tar acids should be neglected.

Now taking Mr. Sage's order, he would first say that as regards his definition he entirely agreed, and he also agreed generally about the sampling and as to proportion insoluble in benzol, that is to say, this last should be an exceedingly small amount. As to stand-

ard methods he felt that if they were to make progress they must not lay down hard and fast or so called standard methods. The chemists concerned should confer and with a system of tender samples there ought to be no difficulty in each side deciding whether tender sample and bulk were alike or differed. On the question of specific gravity he felt there was no more convenient temperature for the determination than 15.5° C., or 60° F. It was a temperature which was most general and which was easily maintained in the laboratories of the United Kingdom. If one remarked, "Yes, but the creosote very often includes a considerable amount of solid matter," he would point out that this solid matter was readily dissolved on the sample being raised to a temperature of say 100° F., and when this was done a considerable time elapsed before the naphthalene again separated out. There was therefore not the slightest difficulty in determining the specific gravity of creosote at 60° F., or 15.5° C., with a specific gravity bottle. At all events, he had never found any difficulty, and he would urge all those who were interested in creosote not to be led into determining the specific gravity at a higher temperature than 60° F. merely because a sample of creosote as received contained solid naphthalene. Then as to a description of the condition of the sample as received. He had never found any difficulty about that. Almost invariably a sample stood at least overnight in the laboratory before it was touched. He suggested that all they needed to do was to register the condition of the sample when they started the examination. If, however, there was any point in this matter being systematized he thought that the sample might be just completely liquefied on receipt and then be allowed to stand all night, the temperature and condition being noted in the morning. Then it would be registered as "solid, temperature so and so," or "half solid," or "almost wholly liquid with traces of solid matter." No doubt from the point of view of handling, the actual physical condition of the creosote was an important matter. He had never had to handle creosote on a large scale, and therefore had no experience, but he thought there must be a very radical disadvantage in having to handle drums or casks of solid matter which required to be heated before the creosote could be got into the creosoting tank. Therefore he quite agreed that the physical condition of the sample ought to be registered, though it did not convey any reliable information as to the condition to be expected under other circumstances such as standing for comparatively long periods at relatively low temperatures. With regard to the expansion coefficient, though from the laboratory point of view he saw no advantage in the determination, it was possibly useful from the point of view of transit. How far that was a serious difficulty he had no means of knowing, and he had no reason to quarrel with the suggestion which the reader of the paper had placed before them. With regard

to flash point he thought there was a certain amount of loose talk about "flash point by the Abel test." As regards the Pensky-Marten apparatus it was practically the Abel apparatus adapted merely for higher temperatures. Within his experience, no difficulty had ever arisen, because there was no question of its flashing at a dangerous temperature. Undoubtedly, however, if a flash point determination were called for the Pensky-Martin apparatus was the one that should be used. Mr. Sage had suggested with reference to the question of fluidity a standing period of six hours. He (the chairman) had already said something on that point, and he would only add that from the point of view of use it was only necessary that the creosote should be fluid at the temperature at which it was employed. It was usual to specify that the creosote should be completely fluid at 90° F. or 95° F., and within his experience all ordinary commercial creosotes satisfied this requirement. The time within which naphthalene would separate varied with the composition of the creosote without affecting the question of suitability, while, as he had already said, he thought the condition at ordinary temperatures was best observed and recorded at the time of examination. With reference to the question of distillation he was pretty closely in agreement with Mr. Sage, except on the one point which had already been dealt with by previous speakers, viz., the position of the thermometer. There was, however, one suggestion he might make in connection with this part of the examination. Using a Wurtz flask of the kind described by Mr. Sage, but of 250 c. c. capacity, and with a delivery tube 1½ inches above the base of the neck and the top of the thermometer bulb level with the side tube, he suggested the advisability of weighing both the flask and the quantity of creosote taken for distillation, the latter being approximately 100 c. c. To the neck of the flask a small piece of wire was attached, and there was then no difficulty in weighing the flask, first empty and then after the creosote was added. On the distillation commencing, they at once encountered the question of water. If the volume of water was large (and within his experience that happened only very occasionally), the only way to obtain an average sample was to determine the volume of creosote and water and make the analysis on the separated creosote. Where such a separation of water was made, it of course did not represent the whole of the water—for once an emulsion of creosote and water was formed it was most persistent and was not to be got out by even long standing. But, again, he did not think that very often arose. Within his experience from 1 to 4 per cent of water was about the maximum that was normally met with, and he thought it well to conduct the distillation in such a manner that the first fraction up to a temperature of about 160°, and until the distillate was clear was always collected in a separate receiver. There was then no difficulty in arriving at the actual proportion of water. Some little care was

required in getting the water out without serious bumping, but the difficulty could be got over. When the distilling vapor reached the temperature of 316° C. or 600° F. the flask with the residue was weighed after cooling, so that the amount of non-volatile matter was correctly ascertained, and with a regulation of the time occupied in the distillation there was no difficulty in obtaining concordant results. He thought one should always note and record the character of the non-volatile residue, as in his opinion it to a very large extent indicated the origin of the creosote. A normal creosote from a gas works tar was a soft solid with semi-crystalline structure. If, on the other hand, it was blast furnace tar it was always more or less fluid, and if it included a certain amount of pitchy matter, then it was also fluid, but very thick and black.

That brought him to another point, viz., the detection of paraffin substances and certain soft pitchy bodies. He did not know whether the process he suggested was novel or not, though he had never seen any reference to it. He had observed that the kind of oil to which he had just referred—one containing soft pitch-like or paraffin substances non-volatile at 316° C., and not of the crystalline character of normal creosote—was easily distinguished by its degree of insolubility in alcohol. A normal creosote was almost wholly soluble in from three to nine volumes of methylated spirit. The proportion of spirit beyond three volumes did not make any very appreciable difference, but he preferred to use nine volumes of alcohol. If the creosote contained this pitchy or solid paraffin matter it was also usually indicated in the original sample by the way in which the creosote clung to a glass vessel, while a normal creosote oil with less viscosity and color did not cling, and at once left the glass clean and practically colorless. Normal creosote yielded an insoluble in alcohol of less than 1 per cent—frequently it was only two or three-tenths—but others gave 2 per cent, occasionally going up to 4 per cent and in rare cases reaching as high as 16 per cent. One of the most valuable points dealt with by Mr. Sage was the determination of the proportion of naphthalene. He fully agreed that the determination must be made on the distillate and not on the original creosote. He thought, however, that the time had come when a more accurate determination of the naphthalene was necessary than had been hitherto general. The method usually practiced was to merely separate without any special attention the solid which crystallized out from the distillate. But by cooling to —10° C. a very much more complete separation was obtained. The method to be recommended was to filter through linen on a Buchner funnel with the aid of the pump. If the distillate indicated that the proportion of naphthalene was large, it was filtered at once without special cooling. The filtrate was then cooled to bring out a further quantity which was once more filtered off, and finally the filtrate was cooled to —10° C. and again filtered. When the proportion of

naphthalene was large if one proceeded to cool the distillate at once, the liquid became so thick that there was considerable difficulty in effecting any filtration. As each portion of naphthalene was got on the filter it was pressed by a glass stopper which just fitted the Buchner funnel, and the comparatively dry naphthalene was removed from the linen to a beaker in which it was weighed, this being done before the next filtration was commenced. In the later filtrations the Buchner funnel was itself surrounded by the cooling mixture used to reduce the temperature of the successive filtrates. In this way a filtrate which remained clear at -10° C. was obtained and the separated naphthalene was so firm and dry that it did not liquefy at the room temperature of say 60° — 65° F.

It might well be asked, how much of the total naphthalene does the quantity so separated represent? Well, that obviously varied with the proportion of what might be called pure tar oil, for of course this remained saturated with naphthalene at -10° C. If the tar oil taken by difference from the other constituents was high, a considerable proportion of the total naphthalene might remain dissolved, but where the proportion of tar oil was low then the naphthalene retained in solution was about balanced by the tar oil retained by the pressed naphthalene which had been separated. There was therefore here a good practical determination, such a proportion in fact as ought to be taken into consideration and reported. The chemist, however, he thought, might well go further and determine, by means of picric acid, the total naphthalene and other substances precipitated by picric acid in the distillate after the removal of the tar acids. This he had himself done for a long time past, and he thought that the practice ought to be adopted more generally. There was no difficulty in this determination by means of picric acid, but he had already taken up so much time that he would not further go into the matter.

With reference to the determination of the tar acids and pyridene bases, he did not think any difficulty now arose. With reference to a point made by Mr. Blagden, he would suggest there were means of distinguishing a blast furnace creosote from a normal coal tar creosote. First, the tar acids or phenoloids of the blast furnace creosote were as a rule of a much higher boiling point than those obtained from ordinary gas tar; and, secondly, as far as his experience went, he had found that the residue non-volatile at 316° C. from a blast furnace oil was very small, and was liquid. Naphthalene of course might be added, and its presence therefore was not indicative; but if one got a small and soft residue non-volatile at 300° C. with a high proportion of high boiling phenoloids, the origin of the creosote was pretty clearly shown. With reference to sulphonation and the determination of the refractive index, he had never found any occasion actually to apply these, but he agreed with Dr. Lewkowitsch as to the difficulty of drawing conclusions from a refractive index when there was a mixture of sub-

stances. Probably sulphonation was more applicable in countries like the United States than in the United Kingdom.

Mr. Sage, in reply, said most distinctly and emphatically that in the majority of cases it was possible to determine the presence of blast furnace creosote in coal tar creosote. Dr. Lewkowitsch had pointed out the difficulty they all met with in dealing with a mixture of substances, but if they took the whole of the distilled fractions and determined their refractive index, they would find that if there was as much as 7 to 10 per cent of blast furnace creosote present, it would bring the refractive index outside the range shown in his (Mr. Sage's) diagrams. He thought members would find that if they worked upon these lines they would be amply repaid. Mr. Butterfield had remarked that it was possible to determine the presence of blast furnace oil in creosote by taking the specific gravity of the fractions. If the sample was sufficiently large to admit of this, it was perfectly correct, and what the chairman had said as to the amount of "tar acids" was also true. All those items helped to define the presence of blast furnace creosote in the oil. He would suggest, however, that there was another way, and that was by making the distillates alkaline and shaking with air, then noticing the color produced as absorption of oxygen proceeded. It was a long method of procedure, but the results usually left little room for doubt. He did not lay as much stress on the determination of naphthalene as Dr. Lewkowitsch appeared to think, although he was fully cognizant of the value of naphthalene as a timber preservative, and he thought that of the solids in creosote it was one of the most valuable. He had on several occasions dissolved naphthalene in "tar acids" and used it for timber. It preserved the wood, but did not give the nice brown color which was expected of creosoted timber. With regard to Dr. Lewkowitsch's remarks as to the position of the bulb of the thermometer, he wished he could agree with him. In fractional distillations of anything else he always used the thermometer with its bulb in the vapor, but many years' practical experience suggested to him that he always got better results the other way—namely; with the thermometer bulb in the boiling creosote. It was a matter which, above all others, needed definite settlement, and personally he was agreeable to either method if only it could be specified and adopted generally.

As to Mr. Butterfield's other query, he was in general agreement as to the point raised by him.

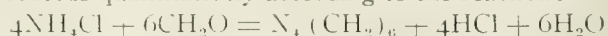
Dr. Tidy's specification, mentioned by the chairman, was one of those antiquities which they all turned up reverently when they wanted to read what other chemists had to say about creosote. He fully agreed with what the chairman had said with regard to the volatility and wash-out-ability of tar acids in creosote, which was used for pickling timber, but he would suggest that it was not only these tar acids and naphthalene in creosote which acted as preservatives. The

high boiling fractions contained some phenoloids, as well as anthracene, and these being of a thick, sticky nature ought to have some beneficial preservative effect. Mr. Hooper mentioned the temperature at which some of these things boiled. He had separated the tar acids from blast furnace oil on many occasions and he could assure Mr. Hooper that, with his thermometer well above 320°C ., he still had a lot of residue left in the flask. (The chairman: Was the thermometer in the liquid?) Mr. Sage explained that in his fractionation of tar acids it was not in the liquid but the bulb was opposite the outlet tube in a Wurtz flask. Mr. Hooper had mentioned that there were no particular Board of Trade regulations with regard to the flash point of creosote. The insurance companies, however, held the whip hand in these matters, and it would also be found that timber used in the work of the underground railways must not be creosoted; that if it was preserved in any way it must not be by means of creosote. He believed that many engineers present would confirm his remarks on that point. Recently he had to determine the solubility of some of the products of creosote in alcohol, and he could confirm what Mr. Hooper had said—namely: that it was a valuable method of distinguishing one variety of creosote from another. Preferably he added a little acetic acid to the alcohol so that any basic substances which were present would form more soluble compounds. His standard solution of picric acid was seldom used except for determining very small amounts of naphthalene in naphthas, and he was not favorably impressed with the idea of a volumetric determination in creosote by means of picric acid. The determination of naphthalene was of great importance to buyers and it was ridiculous to attempt to determine it on the undistilled sample of creosote when it was known how soluble the naphthalene was in the high boiling fractions.

Statistics compiled by the United States Geological Survey show that the production of spelter or metallic zinc from ore for the first six months of 1911 was 140,196 short tons, a gain of more than 5,000 tons over half the record output of 1910. Of this production, 5,135 tons was made from foreign ore. Spelter stocks were reduced from 23,232 tons to 17,788 tons. Imports remained about the same, but exports were nearly double those of half the preceding year. The apparent consumption of spelter was 133,497 tons, an increase of more than 12,000 tons over the half of 1910, but about the same as in half of 1909. The average price of spelter at St. Louis for the period of 5.36 cts. per pound, the London average being .2 cts. less per pound. During the latter part of May and the first part of June the average London price was about .1 ct. higher than the corresponding St. Louis prices. Under this stimulus the May exports of spelter, zinc ore and dross were largely increased over those of the preceding months.

A RAPID METHOD FOR THE ESTIMATION OF SULPHUR IN COAL GAS, OR OF AMMONIUM SULPHATE.*

Ammonia can be determined with rapidity and accuracy by making use of the well-known reaction of Malfatti—the formation of hexamethylenetetramine and liberation of acid by treating a solution of an ammonium compound with formaldehyde; the reaction proceeds quantitatively according to the reaction:



and the acid liberated can be titrated with standard alkali, using phenolphthalein as an indicator.

Determination of Sulphur in Coal Gas.—The liquor obtained in the "Referee's test" is made up to 500 c.c. and 50 c.c. boiled for two or three minutes to get rid of carbon dioxide; to the hot solution 10 c.c. of a neutral formaldehyde solution (of about 30 per cent strength) are added, and the solution is at once titrated with N/10 caustic potash solution. One c.c. of this solution corresponds to 0.0247 grain of sulphur.

It is well to standardize the potash solution by means of a standard solution of ammonium sulphate containing a little ammonium carbonate.

The following figures show that the method gives concordant results, and although the determinations by barium sulphate are a little lower, the discrepancy is too small to be of any importance:

SULPHUR IN GRAINS PER 100 CU. FT.		
Sample.	By barium sulphate.	By rapid method.
I	4.53	4.923
		5.211
		5.058
		Mean 5.064
II	10.19	10.89
		10.89
		10.665
		11.115
		Mean 10.89
III	32.81	33.97
		33.97
		Mean 33.97
V } Redhough Works {	27.46	27.72
VI } {	21.02	22.12
VII } {	21.12	21.63
VIII Elswick	1.80	5.30

As the volume of potash used is considerable the method can be utilized for the determination of sulphur in a smaller volume (1 or 2 c. ft.) of gas and the whole operation can be completed in an hour. It is only necessary to burn a measured volume of the gas under a glass adapter and by means of a water pump draw the products of combustion through a solution of ammonium carbonate, which is then treated with formaldehyde as already described.

Determination of Ammonium Sulphate.—The ammonium sulphate is dissolved in water and, as it is always acid, neutralized with standard caustic potash. An aliquot portion is then boiled for a few minutes, 10 c.c. of neutral formaldehyde solution added, and the liquid allowed to boil for a few seconds. Phe-

*From the Journal Society of Chemical Industry.

nolphthalein is now added, and the hot solution titrated with N/10 caustic potash (of which .1 c.c. = 0.0049 H_2SO_4 , 0.017 NH_3 , 0.0006 $(\text{NH}_4)_2\text{SO}_4$).

Rapid method Ammonia per cent	Barium sulphate method. Ammonia per cent.
1. 25.65	25.64
2. 25.14	25.16
3. 24.95	24.92
4. 24.78	24.77
5. 25.18	25.16

A NEW METHOD FOR DETERMINING OIL ADULTERATION BY MINERAL OR RESIN OIL.*

By ALEXANDER E. OUTERBRIDGE, JR.

When examined by reflected light, hydrocarbon oils (improperly named "mineral" oils), whether crude or partially refined, show a peculiar greenish tinge commonly called "bloom." When examined by transmitted light the bloom disappears and the true color of the oil is seen. This color ranges from dark red or mahogany tint through various shades of orange and yellow up to "water white," according to the degree of refinement. Resin oil possesses the same peculiar characteristics, except that the color of the bloom is pure blue. Its chemical composition is so nearly like that of a hydrocarbon oil that these resemblances appear to me to be more than accidental coincidences and suggest the possibility of a common origin between so-called mineral oil and resin oil. This speculation, however, is not germane to our topic, which has to do strictly with a new practical application of that property commonly called bloom to the instantaneous detection of adulteration of vegetable or animal oils with hydrocarbon oils.

Fluorescence of Certain Oils.—Doubtless everyone has noticed the bloom in mineral oils and wondered perhaps as to the cause of this singular greenish appearance, which is especially noticeable in crude oil and in heavy lubricating oils. Bloom is merely a popular name for a remarkable property possessed by a number of substances, the scientific name for which is "fluorescence." In simple non-technical words, fluorescence is a property inherent in some substances of becoming self-luminous while exposed to certain rays of light known as "ultra-violet" or "actinic" rays. These rays are always found in sunlight and in some forms of electric light.

In the course of my investigations I found that the greenish bloom or fluorescence of mineral oil and the blue bloom of resin oil noticeable in daylight can be enormously intensified or magnified, perhaps a thousand fold, so that, if a single drop of mineral oil be placed in a vessel containing a hundred or even a thousand drops of pure linseed oil, or any other non-fluorescent oil, its presence may be instantly detected by the greenish fluorescence which it imparts to the whole of the oil. The same is true of resin oil, which gives blue fluorescence. The utilization of this obser-

vation for practical purposes of detecting adulterations is the gist of this paper.

By preparing standard samples of any non-fluorescent oil containing one-tenth, one, two, three per cent, and upward, of mineral or resin oil, in clear glass test tubes placed in a suitable frame against a dark background, each showing readily and unmistakably the increasing proportions of the adulterant under a light giving ultra-violet rays, a "fluorescent scale" has been established, somewhat similar to the well-known carbon color scale used in steel foundry laboratories for quickly determining, by color comparison, the proportion of carbon in an acid solution of steel.

I am prepared to make unhesitatingly the broad assertion without fear of contradiction that there is no method known by which the presence of either mineral or resin-oil in any non-fluorescent oil in small or large amount can be so disguised as to be undetectable instantly by this method, and this certainly is an interesting and important fact of considerable practical value to consumers of costly oils.

Test Applicable to De-Bloomed Oils.—Methods of chemical treatment of mineral oil have been discovered to "de-bloom" mineral oil so that it can be used with impunity, so far as the bloom is concerned, as an adulterant for expensive vegetable and animal oils, and I learned that there is a very large trade in de-bloomed oils for this purpose. Samples of de-bloomed oils of different grades and colors were obtained. They are free from bloom in bright sunlight or ordinary diffused daylight, or in the light from an ordinary electric arc, but when subjected to the kind of light which I shall presently describe *they all became highly fluorescent*, and even the proportion of adulteration with de-bloomed mineral oil in any specimen of non-fluorescent oil mixed with such de-bloomed mineral oil may be stated. I anticipate that this positive statement now made for the first time will cause some consternation among makers of de-bloomed mineral oil.

If both mineral oil and resin oil be used in combination as adulterants, it becomes more difficult to make quantitative determinations instantly by the fluorescent method; hence the qualification implied in the word often. But "practice makes perfect" in many operations, and this is no exception to the rule.

How Fluorescence is Obtained.—It is the *ordinary inclosed arc*, so commonly used in industrial works by reason of its relative economy, that happens to give out rays of the exact wave length needed to enormously increase the fluorescence of these oils. If the plain glass cover of this light fits properly, so that air does not enter as rapidly as it is consumed, the arc burns in a partial vacuum or, at least, the air is rarified and, under these normal conditions, this light shows continuously, after burning a minute, a faint rosy light in addition to the powerful white light. If now a vessel containing any mineral oil, crude or refined, or any resin oil, be placed in the path of these rays the most intense fluorescence appears, even in daylight, green-

*From The Iron Age.

ish in the case of mineral oil, blue in the case of resin oil, thin films glowing in the same manner. So strong is this fluorescence that I have even detected 1 c. c. of crude mineral oil in 999 c. c. of non-fluorescent oil.

I have examined a large number of vegetable oils, such as cotton-seed oil, corn oil, China bean oil, China wood oil, etc., and have not found a trace of fluorescence in any of them. It is stated in some text-books that "oleic acid," which is found in lard oil, is fluorescent. On examination I find that pure white strained lard oil is entirely free from fluorescence under the ultra-violet ray, but all of the samples of so-called No. 1 or No. 2 lard oil (sold for use in machine shops) examined possess some fluorescence, and this may prove to be a novel means of rapidly determining the proportion of oleic acid in lard oil, though I only suggest it tentatively, not having studied the matter carefully from this point of view. The slight fluorescence of ordinary lard oil is different in appearance from that of mineral oil or resin oil, and does not materially interfere with the application of the fluorescent test for its adulteration with mineral or resin oil.

Simple Scheme of Testing.—In daily practice I have found it convenient to put the standards in narrow tubular oil-test bottles holding about 50 c. c. each; these are corked, labeled and placed side by side in small wooden racks (like test-tube holders) on a shelf in proximity to an inclosed arc light, beginning with pure oil at the left-hand side. Then a similar sample containing 0.1 per cent of mineral or resin oil, as the case may be, then 1 per cent, and so on, increasing by single percentages up to 10 per cent. It is advisable to prepare several different series of standards with fluorescent oils of different grades. Crude mineral or resin oils are much darker in color than refined oils, and the color by transmitted light is a guide to the kind of oil that has been used for adulteration and is consequently an indication of the proper standard series to be used for comparison in making a quantitative fluorescent analysis.

It is not necessary to prepare standards for each kind of vegetable or animal oil; thus, the standard series prepared with linseed oil serves for examination of cotton-seed oil, corn oil, China wood oil, China bean oil or any other non-fluorescent vegetable oil. It is necessary, however, to prepare special standards with lard oil for testing adulterated lard oils.

The compounding of core oils has become a large business and nearly all samples that have come to my notice contain mineral or resin oil or both. Neither of these oils impart any valuable properties to core oils, but are used simply to dilute more costly oils; and, in point of fact, they are positively deleterious, being of a non-drying nature, impairing the good oil binder and requiring more fuel and a longer time for baking the cores in the ovens.

I have found that "Soya" oil expressed from beans grown in enormous quantities in China and elsewhere is an excellent substitute for linseed oil for making

cores if used in its natural state, without having been compounded or adulterated by core-oil makers. It costs about 60 cents per gallon for finer grades. The very best substitute for linseed oil as a binder for oil cores that I have discovered is crude whale oil, costing about the same as Soya oil, the only objection to its use being an unpleasant fishy smell which escapes from the core ovens during the baking of the cores. It makes a splendid binder. Cotton-seed oil is used for the same purpose, but so much larger proportion of oil to sand is required that there is little economy in its use as compared with the other vegetable oils.

A simple and practical test of the value of core oils is to make a dozen companion test cores 1x1x15 in. from batches of pure linseed oil and sharp sand, and also from the same proportions of any other oil and sharp sand. These are placed side by side on an iron plate and baked under precisely the same heat conditions. When cold they are broken on a transverse testing machine with supports 12 in. apart. The relation between the average strength of the two sets of test cores is a measure of the binding qualities of the oils.

The principal countries exhibiting at the International Rubber Exhibition held in London, June 24 to July 14 last, were Brazil, Ceylon, Federated Malay States, Belgium, Holland, Germany, South India, Uganda, and the West Indies. The special medal for the best exhibition of rubber was awarded to the State of Amazonas, Brazil. This exhibition consisted of 20 tons of rubber and 5 tons of caucho ball, representing approximately the one-thousandth part of the rubber passing annually from Manaos. In 1901-2, 19,989 tons of rubber and caucho were received in Manaos from the State, inclusive of that in transit from Peru, Bolivia, and Venezuela. In 1909-10 this total had risen to 30,065 tons. This was natural or "wild" rubber, from vines and trees indigenous to the soil.

The statistics collected by the Association of German Iron and Steel Producers are returned so rapidly by the interested makers that it is already possible to ascertain the production of pig iron in Germany in the first half of the present year. Although the output of pig iron of 1,262,000 tons in June was less than in each of the three preceding months, it was yet in excess of the corresponding month in 1910. The total make in the first six months was 7,682,000 tons, as compared with 7,202,000 tons in the equivalent period, being an increase of 480,000 tons. The half year's output constitutes a record and corresponds with an annual production of 15,360,000 tons, while the quantity of pig iron made in 1910 only reached 14,793,000 tons.

In 1910 New Zealand exported wool to the value of \$40,338,873, as against \$30,646,616 in 1909.

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NOTES AND COMMENTS.

Need of Research in Chemical Engineering.

Every branch of engineering, with the exception of chemical engineering, possesses its hand-book to which the engineer can turn for information which is desired upon almost any subject that is under consideration. The chemical engineer has access to no such fund of information; primarily because there is no actual data available, on the fundamental operations of chemical engineering. Take for example, as common an operation as that of filtering in a press a liquid containing a suspended solid: there is absolutely no data covering the quantity of filtering area required to filter a definite quantity of liquid in unit time. There are no works which even contain any definite statements as to the best kind of filtering material to use for certain classes of substances, no data is available on the wearing qualities or life of the various filtering materials, or on the most effective working pressure.

Anyone looking for information on the subject of

evaporation quickly finds that there is a great dearth of reliable data on the transmission of heat through metal pipes or tubes of various sizes and thicknesses. It is soon found that much of the information available on this subject is misinformation. Practically all of the data on this subject is in the hands of the manufacturers of various types of evaporators, and is applicable only to their own product. There is even no reliable information on such a simple process as evaporation of water from a simple steam-jacketed kettle, with certain thickness of metal wall, and at a specified steam pressure within the jacket. In fact, the whole subject of the mechanical appliances, used in the chemical industries, with mathematical information as to their efficiency, is rather hazy.

It is the writer's opinion that the department of chemical engineering in our colleges and universities could be of real service to the profession of chemical engineering by securing and publishing some information along the lines indicated. They could very well obtain such information by properly directed experiments on the part of certain students, while at the same time imparting to the students valuable information as to the methods to be used in determining the efficiency of a certain process or appliance. It is suggested that the American Institute of Chemical Engineers might well inaugurate some such project and apportion certain lines of investigation to certain schools best prepared to handle the problem. A few years would be required to obtain sufficient data to be of much value, but the necessary information can probably be obtained in no other way, and the value of the results would undoubtedly repay the time and labor spent in obtaining them. Such work would be of infinitely more value than most of the present theses prepared by students in chemical engineering.

Factory or Laboratory.

The recent graduates from our courses in chemistry and chemical engineering have probably either spent the past two months in working under new surroundings, or in searching for an opportunity to show someone the extent of their scholastic training.

It seems to the writer that, in this period of the development of the young chemist and chemical engineer, there is no question the solution of which will be of so much importance to the young man, as the one of securing proper connections and surroundings for expansion in his work. The graduate in chemical engineering has two quite different fields open to him; he may secure employment in the laboratory connected with some industry, and start as an analyst, his aim being, we suppose, to familiarize himself with the conditions essential to the chemical control of the operations applied in this particular industry, and to use the knowledge thus obtained in securing a position of responsibility in a plant where he will have something to do with the actual manufacturing processes.

The other line of endeavor open to him is to accept

a position as a laborer in a plant, depending on his brains and training to raise him from this position through the grades of sub-foreman, foreman, assistant superintendent and superintendent, to a point where he will not only be receiving adequate compensation for his services, but also to where he will be actually responsible for certain manufacturing processes.

The writer has, during the past years, noted the progress of a considerable number of young men trained in chemical engineering, and the observation has been that the men who accept laboratory positions in the great majority of cases never succeed in leaving the laboratory and connecting up with the factory operations. If this occurred only in a few instances, the failure to make the necessary connections might be ascribed to something lacking in the man himself, but the large number of cases in which this occurs, leads one to the conclusion that there is something fundamentally wrong in the relation which the average laboratory bears to the departments carrying on the actual manufacturing operations. This is sometimes due to the prejudice entertained by old-time foremen and superintendents against technically trained men, sometimes it is due to the indifference of the laboratory chief to the part which his force should sustain in the composite relations of the plant processes as an entirety.

It is undoubtedly difficult for the average college graduate to reconcile himself to the idea of hard manual labor, and the associations which he will necessarily have with the men customarily employed for such duties in our manufacturing plants. The writer believes, however, that it is infinitely easier for a young man to obtain a position of responsibility where he will be in actual control of the manufacturing processes by starting as a laborer, than it is for him to obtain such a position by way of the laboratory. It is a test of the nerve and grit of any young man that he be willing to accept a laboring position in any manufacturing industry, but it is just this nerve and grit that is essential to his later success. It is the lack of it in many instances which leads him to go into the laboratory, and which keeps him there once he has entered it.

Valuable deposits of asbestos have been found at Kuantien, 45 miles from Antung, China.

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C. L. PARKER

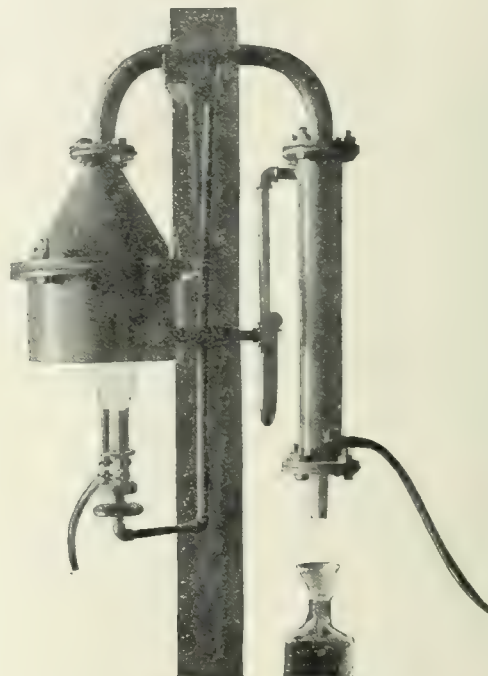
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THE PRACTICAL VALUE OF THE DETERMINATION OF BRITISH THERMAL UNITS IN TESTING ANTHRACITE COAL.

By S. F. PECKHAM.

This paper and the conclusions sought to be established, are applicable only to anthracite coal.

The City of Greater New York embraces an area that may be roughly outlined as half a circle, having a diameter of 35 miles, extending from Tottenville, S. I., to the boundary line between the Borough of the Bronx and Mt. Vernon. A large percentage of this area is covered by the water of the East River and Upper Bay. While means of communication are admirable, they embrace tunnels, bridges, and ferries, that consume a great amount of time in their use.

The deliveries of anthracite for the use of the City of New York, including the municipal ferries, bridges, public buildings and water supply, are made at about 1,200 stations. The quantities delivered at these stations vary from one ton to the contents of a barge of several hundred tons.

To test the quality of the coal delivered at these stations, requires that samples be made and gathered at one or several centers, where the tests of quality may be made.

It is assumed for the purpose of illustrating this paper that the samples are properly taken, as the samples used were all taken by one person. As an economic question the gathering of samples by competent persons over such a vast area is a serious problem. The samples gathered were sorted by screens as to sizes and the coal that passed each screen was carefully weighed and the percentages determined. These masses were again mixed and sorted by hand and the amount of bone and slate present weighed separately. The percentages of bone and slate are very important determinations. They indicate in a measure, when considered together with the results of analysis, whether the coal is poor in quality or whether a good coal, either through carelessness or purposely, contains an excessive amount of slate.

The whole of the sample was then crushed and pulverized in a ball mill, not to a powder, but to a finely granular condition, the grains resembling in size, grains of rifle powder.

This is an important consideration, for the combustion produced in the Mahler bomb is to some

extent explosive and immediate combustion should proceed throughout the entire mass in a manner precisely similar to the combustion of gun powder. If the coal is pulverized to a powder it is liable to be thrown out of the disc into the bomb and escape combustion. The grains should, however, not be coarse, otherwise the combustion may be more or less incomplete. It is very important that a complete combustion be secured.

This granular coal, which should contain very little powdered coal, is sampled and placed in a small bottle from which all of the material for analysis is to be taken. The first determination is made upon one gram, weighed into a platinum crucible, holding about 30 C. C. with a tightly fitted lid.

All of the samples tested for this paper were dried in an electric oven at a temperature of 180° F. The amount of water determined under these conditions can be duplicated any number of times on the same specimen to almost identity. The loss of weight gives the percentage of water directly.

The covered crucible was then placed in the flame of a Bunsen's lamp and heated to bright redness for seven minutes, cooled and weighed. The loss of weight gives the percentage direct of volatile matter. This percentage is sometimes termed volatile combustible matter, but it is not necessarily all combustible matter in samples of hard anthracite.

The action of these samples indicates very conclusively that a part of this loss consists of occluded gases of varying composition. The sample in the open crucible was then burned to ashes and the ashes weighed. The combustion of white ash coals is quite rapid, but red ash coals and coals containing pyrite require a longer time for complete combustion. The ash is weighed and its weight noted. The percentage of ash and the difference in weight between the ash and coal deprived of its volatile matter is computed as fixed carbon. These determinations divide the coal into moisture, volatile matter, fixed carbon, and ash. They may be almost indefinitely repeated to within a few thousandths of one per cent.

The determination is easy, rapid and exact, consequently while giving only proximate results, it is a very satisfactory method of analysis and I think it can be shown to constitute an indication of the quality of the coal that is both certain and exact.

If it is desired that a determination of the British Thermal units shall be had in addition to these

results, resort is had to the Mahler bomb or some of its later modifications. Experience has shown that the Mahler bomb gives results under certain conditions that corroborate and emphasize the results obtained by combustion.

The results furnished by the Mahler bomb, however, cannot be exactly duplicated, in consequence of the fact that the operation of the instrument is more delicate than our observations, that is to say, we cannot read the thermometers to a degree of exactness, corresponding to the variation in the efficiency of the instrument; for this reason, under the most favorable conditions, results will vary by a certain percentage. The action of the instrument is also more efficient at a given temperature, which if exceeded in consequence of very hot weather, or diminished in consequence of a low temperature, has a very marked influence in modifying the results obtained. Another cause of variation is observed to depend upon the amount of moisture contained in the specimen of coal. If the coal is at all wet or even damp, it is possible that the water may be decomposed by the carbon, the oxygen disassociated from the hydrogen, and the hydrogen burned as free hydrogen, thus increasing the apparent thermal units and destroying the values by a considerable percentage. I have repeatedly experienced an excess of thermal units, which I could explain in no other way. Another cause of error lies in carelessness in reference to the amount of coal taken for the assay. If much more than 350 milligrams are taken, the combustion is liable to be incomplete. The amount of assay was found on experience to be proportioned to the volume of oxygen and any excess over 400 milligrams is liable to produce an incomplete combustion, particularly where there is a residuum of slate or bone.

The use of the Mahler bomb is thus shown to be subject to variation, while theoretically indicating the exact value of the coal for heating purposes. Variation in the conditions under which the determination is made, subjects the results to variation, which can only be controlled with great difficulty and may result in proving the coal to be more efficient than it really is, or less efficient than it really is. The results being by no means as exact and constant as an ordinary laboratory determination, as for example those obtained by the proximate analysis which has before been described.

This subject has been agitated for several years by the City Government of New York. Formerly each department of the city purchased its own coal on such terms as it saw fit. The result was often found to be entirely satisfactory to the commissioners of one department, but were very unsatisfactory to those of another. One source of difficulty was found to lie in the varying and often antiquated, and sometimes wornout, plants in which the coal was burned. The varying results obtained, when estimated in dollars and cents, under these varying conditions, led to comparisons between the efficiency of city plants

and those of privately owned plants, often greatly to the disadvantage of the city.

In consequence, a movement was made in the last months of 1909 to standardize the specifications under which coal was purchased for the city, with an idea that more satisfactory results would follow. In other words, that more heat could be obtained for a dollar expended.

A committee was appointed by Mayor McClellan which embraced in its membership, representatives of all parties interested. The action of the committee, however, was delegated, by the chairman, to a sub-committee, in which practical experience was entirely overshadowed by academic and theoretical knowledge. It is not necessary to state the lack of fitness possessed by the chairman of this sub-committee, who was an accomplished physician, but no chemist. His influence over the committee, however, was very great.

A new administration took up this matter after January 1, 1910, but the influence of the theoretical advocates of paying money for heat actually obtained still held sway. The result was the adoption of specifications under which payments were to be made upon the contracts, based almost wholly upon the determination of British Thermal units, developed by the combustion of the coal.

The result has been practically disastrous to the interests of the city.

Under the old specifications, the coal was required in most departments to contain, not to exceed 16 per cent of ash, nor less than 80 per cent of fixed carbon, a theoretical consideration that would demand a very fine quality of coal, but which can be easily shown to be wholly impracticable. Analyses of several hundred specimens of coal show no coal on the market that yielded 80 per cent of fixed carbon, contains 16 per cent of ash. Sixteen per cent plus 80 per cent is 96 per cent, leaving only 4 per cent to include water and volatile matter. The water will average, from our laboratory experiments, $2\frac{1}{2}$ per cent in air dried coal. The volatile matter averages about 6 per cent. The volatile matter is a constant constituent of every variety and size of coal. The water may be removed by special care in drying, as I have analyzed samples that would not lose any water at a temperature of 212 degrees F., but 99 per cent of all the coal that I have examined of all sizes, contained an average of $2\frac{1}{2}$ per cent of water when air dried. In consequence of these facts, it is impossible that the fixed carbon and ash remaining after the water and volatile matter had been removed should be equal to 96 per cent. The sum of the fixed carbon and the ash should never exceed $91\frac{1}{2}$ per cent, if a maximum of 16 per cent of ash is allowed. The minimum of fixed carbon would then not exceed 75 per cent, which would indicate a barely good sample of coal. It is impossible to pick from coal as small as buckwheat, all of the slate; it is also impossible practically, to prepare an average of a dozen samples of buckwheat coal ob-

tained from any but the very finest mines that contains 80 per cent of fixed carbon. As a general rule, it is impossible to comply with these requirements as stated. The maximum, namely of 75 per cent, is too high for pea and buckwheat coal and 70 per cent would be more reasonable in the light of experience and could be more strictly enforced. The carbon contained in bone (which is only a very poor coal) and the mineral matter contained in the slate are included in the requirements for fixed carbon and ash and therefore, take care of themselves. Bone, whether containing much or little carbon is always included in the determination of the ash and fixed carbon made for the whole sample; if the bone is picked out and analyzed separately, it would leave the remainder of the coal richer in carbon and better than the whole sample.

The standardized specifications required that the coal delivered should conform to the following requirements.

Standard Analyses—Anthracite Coal.—The quality of coal of the different sizes contracted for shall conform to, or be superior to the coal described in the following standard analyses applying to such sizes:

Names of sizes of coal	Maximum allowable moisture, per cent as delivered	Maximum allowable ash, per cent dry coal	Maximum allowable volatile combustible matter, per cent dry coal	Maximum allowable sulphur, per cent dry coal	Minimum allowable B. T. U.'s, per pound dry coal
Broken	4	11	8	1½	13,200
Egg	4	11	8	1½	13,200
Stove	4	12	8	1½	13,000
Nut	4	12	8	1½	13,000
Pea	5	16	8	1½	12,300
Buckwheat—					
No. 1.	6	17	8	1½	12,200
No. 2.	6	18	8	1½	12,100
No. 3.	6	18	8	1½	12,000

These requirements are very high and their efficiency is entirely nullified by a provision for an aggregate deduction in gross tonnage as follows: "After the corrections for excess moisture shall have been made as herein specified, all percentage in deductions as above described to be made on account of a deficient number of British thermal units per pound, and an excess percentage of ash, volatile sulphur, and volatile combustible matter will be aggregated and totalized, and will be deducted as a whole, and payment will be made only for the balance of the gross tonnage at the price bid per gross ton furnished, delivered and stored and trimmed." That is to say, that after specifying the requirements of a very high grade of coal it is made possible to deliver any kind of coal that could be purchased.

I propose to demonstrate the truth of this statement by a series of tables which have been compiled from the results of the analyses of 100 samples of coal analyzed at the laboratory of the Department of Finance. These results are tabulated first, in extenso in Table I.

TABLE I.—ANALYSES OF 100 SPECIMENS OF ANTHRACITE COAL

No.	Size	Bone.	Slate.	Water.	matter, carbon	Ash.	B.T.U.	
1	Broken	2.8	1.8	1.15	6.70	82.58	9.57	13,435
2	Broken	3.8	1.5	2.87	5.73	80.98	10.42	13,372
3	Broken	7.7	5.9	2.13	6.35	76.73	14.79	12,578
4	Broken	4.0	1.6	2.45	5.35	81.25	10.95	12,501
5	Egg	3.1	1.2	2.15	5.94	83.68	8.23	13,692
6	Egg	2.6	1.9	2.82	4.98	83.05	9.25	13,512
7	Egg	3.7	1.8	1.59	6.68	81.08	10.65	13,478
8	Egg	4.7	4.8	2.86	5.72	80.72	10.70	13,289
9	Egg	5.0	3.0	2.10	5.34	80.50	12.06	13,276
10	Egg	1.9	.8	1.85	7.05	80.03	11.07	13,114
11	Egg	2.5	5.1	1.90	6.90	80.55	10.65	12,740
12	Egg	6.8	4.0	2.52	6.00	78.58	12.90	12,690
13	Egg	6.6	4.6	2.95	6.12	77.18	13.75	12,620
14	Egg	6.3	5.1	2.55	5.00	77.40	15.05	12,085
15	Egg	5.4	5.3	3.35	3.90	76.66	16.09	11,435
16	Stove	6.0	2.5	1.70	6.86	80.24	11.20	12,792
17	Nut	4.7	3.2	2.38	5.62	73.77	18.23	11,094
18	Pea	3.5	2.2	1.87	6.07	80.83	11.23	13,978
19	Pea	2.2	2.0	2.55	5.33	80.75	11.37	13,294
20	Pea	2.0	3.0	1.38	4.45	82.70	11.47	13,194
21	Pea	1.2	2.4	1.22	5.53	82.80	10.45	13,120
22	Pea	3.8	4.5	2.12	6.43	79.21	12.24	12,990
23	Pea	3.3	3.0	2.00	5.76	79.22	13.02	12,990
24	Pea	2.5	2.9	2.05	5.10	79.75	13.10	12,895
25	Pea	1.3	2.6	2.40	5.22	78.70	13.68	12,870
26	Pea	2.0	3.2	2.00	4.90	80.13	12.97	12,791
27	Pea	3.2	1.3	2.80	5.20	79.50	12.50	12,754
28	Pea	2.5	2.0	1.29	6.13	80.00	12.58	12,715
29	Pea	1.6	5.8	2.27	5.78	78.75	13.20	12,647
30	Pea	6.2	4.6	2.10	7.71	75.83	14.36	12,634
31	Pea	6.0	5.0	2.21	5.59	77.75	14.45	12,627
32	Pea	2.0	2.4	1.95	5.65	79.70	12.70	12,616
33	Pea	2.6	3.5	2.90	5.85	76.88	14.37	12,602
34	Pea	1.9	2.9	2.70	5.90	77.70	13.70	12,578
35	Pea	1.5	2.9	2.45	5.55	79.00	13.00	12,567
36	Pea	7.0	7.9	2.20	6.18	75.13	16.49	12,565
37	Pea	2.3	4.1	1.60	7.64	74.76	16.00	12,524
38	Pea	1.4	2.8	1.83	5.26	80.10	12.81	12,501
39	Pea	4.0	3.2	2.68	5.65	78.80	12.87	12,488
40	Pea	3.4	6.2	2.47	7.16	74.32	16.05	12,466
41	Pea	4.8	3.1	1.80	4.95	80.11	13.14	12,445
42	Pea	2.8	4.7	1.75	6.30	75.70	16.25	12,386
43	Pea	3.1	2.8	1.27	7.03	76.23	15.47	12,378
44	Pea	3.2	1.8	1.74	6.26	78.60	13.40	12,321
45	Pea	2.5	2.9	1.40	6.26	76.87	15.47	12,317
46	Pea	3.2	6.4	1.32	6.23	77.49	14.96	12,281
47	Pea	1.9	3.3	1.75	5.20	80.02	13.03	12,272
48	Pea	4.0	5.5	1.70	5.60	77.60	15.10	12,258
49	Pea	2.8	4.0	2.00	5.90	77.40	14.70	12,250
50	Pea	2.8	1.8	1.58	5.62	79.18	13.62	12,166
51	Pea	5.5	2.8	2.43	4.34	78.87	14.38	12,160
52	Pea	5.2	4.4	1.80	6.02	76.25	15.93	12,159
53	Pea	5.0	3.0	2.67	4.33	75.62	17.38	12,146
54	Pea	3.5	4.5	2.70	5.25	76.92	15.13	12,105
55	Pea	4.0	9.8	2.00	6.88	75.74	15.38	12,060
56	Pea	4.5	2.4	2.42	3.86	79.72	14.00	12,047
57	Pea	2.1	4.0	1.70	5.35	78.72	14.23	11,995
58	Pea	5.2	6.5	2.22	5.58	73.70	18.50	11,989
59	Pea	3.0	4.0	2.57	4.83	77.25	15.35	11,946
60	Pea	4.5	5.1	2.02	6.20	73.03	18.75	11,946
61	Pea	2.9	5.8	1.05	7.04	74.11	17.80	11,894
62	Pea	3.1	2.6	1.30	5.75	75.79	16.48	11,892
63	Pea	5.3	4.7	1.30	5.75	75.70	17.25	11,867
64	Pea	3.3	3.8	2.94	3.86	77.33	15.87	11,840
65	Pea	4.2	5.3	2.00	5.75	74.88	17.37	11,811
66	Pea	3.3	6.4	2.55	4.66	75.47	17.32	11,718
67	Pea	4.2	6.0	1.77	5.51	75.82	16.90	11,714
68	Pea	4.0	5.5	2.15	4.62	76.55	16.68	11,692
69	Pea	3.4	2.5	1.43	5.66	73.39	18.92	11,577
70	Pea	5.6	8.8	2.15	6.18	72.34	19.33	11,541
71	Pea	5.4	3.4	2.50	4.00	76.00	17.50	11,435
72	Pea	5.1	7.0	1.65	6.15	70.55	21.65	11,383
73	Pea	7.1	2.9	2.17	4.38	77.45	16.00	11,320
74	Pea	3.0	10.5	1.45	4.89	76.31	17.35	11,264
75	Pea	3.8	7.8	2.50	4.85	75.40	17.25	11,181
76	Pea	3.0	4.5	1.50	6.05	73.70	18.75	11,059
77	Pea	3.6	6.4	2.42	4.26	72.67	20.65	10,992
78	Pea	2.2	6.5	3.86	6.74	70.98	18.42	10,919
79	Pea	3.5	5.3	1.00	5.97	75.68	17.35	10,892
80	Pea	3.1	4.8	1.64	5.00	70.81	22.55	10,539
81	Pea	7.7	6.1	2.59	5.23	71.26	20.92	10,446
82	Buckwheat	1.6	4.0	1.32	6.33	76.25	16.10	12,427
83	Buckwheat	1.7	2.7	1.95	6.53	75.52	16.00	12,380
84	Buckwheat	1.8	4.1	1.90	5.57	78.93	13.60	12,180

85 Buckwheat.1.3	4.7	1.79	7.26	75.67	15.28	12.026
86 Buckwheat.2.0	6.0	2.51	5.94	75.87	15.68	11.916
87 Buckwheat.2.2	8.9	1.71	5.78	75.94	16.57	11.860
88 Buckwheat.3.4	2.7	3.50	5.94	74.44	16.12	11.838
89 Buckwheat.2.1	1.5	1.62	6.71	74.52	17.15	11.829
90 Buckwheat.2.0	4.5	1.45	6.22	75.15	17.18	11.666
91 Buckwheat.2.1	5.2	3.67	4.91	74.00	17.42	11.553
92 Buckwheat.3.00	4.5	2.78	3.44	77.70	16.08	11.495
93 Buckwheat.3.7	6.0	3.52	4.14	75.12	17.22	11.490
94 Buckwheat.2.1	5.0	3.00	4.10	75.52	17.38	11.382
95 Buckwheat.1.8	5.1	2.90	4.70	74.30	18.10	11.363
96 Buckwheat.1.5	3.5	2.40	4.10	78.42	15.08	11.356
97 Buckwheat.3.0	5.0	2.85	3.88	74.32	18.95	11.183
98 Buckwheat.2.9	3.7	2.23	6.42	68.92	22.43	11.064
99 Buckwheat.1.4	4.2	2.50	4.67	73.70	19.13	10.879
100 Buckwheat.2.5	6.0	3.35	4.78	69.37	22.50	10.202
101 Buckwheat.1.8	3.6	2.15	1.76	78.64	14.45	11.792

From these results as thus tabulated Table II has been computed:

TABLE II.

Broken ..	Yielding B. t. u. under 13,500 and above 13,000 were	2
Broken ..	" " " 13,000 " " 12,500 "	2
Egg	" " " 13,500 " " 13,000 "	5
Egg	" " " 13,000 " " 12,500 "	4
Egg	" " " 12,500 " " 12,000 "	1
Egg	" " " 12,000 " " 11,500 "	0
Egg	" " " 11,500 " " 11,000 "	1
Stove	" " " 13,000 " " 12,500 "	1
Nut	" " " 11,500 " " 11,000 "	1
Pea	" " " 14,000 " " 13,500 "	1
Pea	" " " 13,500 " " 13,000 "	3
Pea	" " " 13,000 " " 12,500 "	17
Pea	" " " 12,500 " " 12,000 "	18
Pea	" " " 12,000 " " 11,500 "	14
Pea	" " " 11,500 " " 11,000 "	6
Pea	" " " 11,000 " " 10,500 "	5
Buck	" " " 12,500 " " 12,000 "	4
Buck	" " " 12,000 " " 11,500 "	6
Buck	" " " 11,500 " " 11,000 "	7
Buck	" " " 11,000 " " 10,500 "	2

Table III has been computed as follows:

TABLE III.

	F. C.	Yielded	B. T. U.
	Per cent.	Ash. Per cent.	Per cent.
9 samples very good egg.....	80.59	11.03	13.388
1 sample fair egg.....	77.40	15.05	12.085
1 sample very poor egg.....	76.66	16.09	11.435
1 sample stove coal.....	80.24	11.20	12.792
1 sample poor nut.....	73.27	18.23	11.094
4 samples very good pea.....	81.77	11.13	13.399
17 samples good pea.....	78.33	13.59	12.697

The average of 4 samples of broken coal gives F. C. 80.38 per cent, ash 11.43 per cent, B. T. U., 12.971.

The average of the whole 21 samples of very good and good pea was F. C. 80.03 per cent, ash 12.35 per cent, B. T. U. 13.046.

Eighteen samples of good pea yielded: F. C. 77.52 per cent; ash, 14.96 per cent; B. t. u., 12.262 per cent.

The average composition of the 39 samples of good and very good yielded F. C. 78.77 per cent, ash 13.66 per cent, B. T. U. 12.604.

	F. C.	Yielded	B. T. U.
	Per cent.	Ash. Per cent.	Per cent.
14 samples of poor pea.....	75.29	17.19	11.744
11 samples very poor pea.....	73.98	18.94	11.039
4 samples buck size, very good pea.....	76.59	15.25	12.253
13 samples buck size, good pea	73.37	17.33	11.548
2 samples buck size, poor pea	71.53	20.81	10.540

Of these 100 specimens 29 were delivered at the municipal ferry houses at St. George, Staten Island, and at 39th street, South Brooklyn. The deliveries were made by one firm under the old specifications

during about one-half of the year and the other half of the year they were made by new contractors under the new specifications which admit of corrections which nullify the force of the specified requirements. Under the old specifications, the contractors were held to the specified requirements by frequent tests made by different parties, and hence confirming each other.

In Jan. four deliveries averaged F. C. 80.71 per cent, ash 12.32 per cent, B. T. U. 13.032.

In Feb. six deliveries averaged F. C. 79.72 per cent, ash 12.85 per cent, B. T. U. 12.786.

In March, two deliveries averaged F. C. 76.28 per cent, ash 16.36, B. T. U. 12.092.

In April, two deliveries averaged F. C. 78.00 per cent, ash 14.05 per cent, B. T. U. 12.285.

In May, four deliveries averaged F. C. 79.36 per cent, ash 12.66 per cent, B. T. U. 12.788.

In June, three deliveries averaged F. C. 79.48 per cent, ash 12.61 per cent, B. T. U. 12.879.

Average for six months, F. C. 78.93 per cent, ash 13.47 per cent, B. T. U. 12.642.

In August the deliveries under the new specifications began with coal averaging F. C. 80.11 per cent, ash 13.14 per cent and B. T. U. 12.445 and ended with a delivery that averaged F. C. 75.68 per cent, ash 17.35 per cent and B. T. U. 10.892.

The delivery of poor coal at these ferries produced two very marked results. First, it nearly doubled the labor required of the firemen. Second, it nearly doubled the labor and expense of removing the ashes produced in such large quantities.

I am told that it was only with great effort on the part of the firemen, that steam could be maintained at the proper pressure to give the ferry boats the required speed.

A more graphic use of these averages can be made by averaging them.

For the first six months the average delivery gave F. C. 78.93 per cent, ash 13.47 per cent, B. T. U. 12.642. If these requirements were used in the specifications drawn from such a large number of samples there would be no necessity for deduction in accordance with fine spun theories without practical value, having for their object, on the part of the city officials, to get the heat that is paid for, and on the part of the contractors the practical avoidance of the required obligations and the deliveries of coal containing so much slate that thousands of tons of slate are being sold to the city.

Anyone who will take the trouble to examine these results of the analyses of 100 specimens can easily discover that a constant inverse ratio exists between the amount of ash obtained on analysis and the B. T. U. determined by the calorimeter. At the same time a direct ratio exists between the amount of fixed carbon obtained by the analysis of anthracite coal and the B. T. U. as obtained by the calorimeter. For practical purposes the determination of the B. T. U. offers no additional information as to the value of the sample of anthracite.

The results obtained by the calorimeter simply confirm the results obtained by the determination of the fixed carbon and ash, adding nothing to these results. In the table of 100 specimens there are a total of 64 samples of pea coal, among these 64 samples every grade of pea coal on the market, from the best to one so poor that it will scarcely answer for steam purposes, may be found. They were delivered all over Greater New York and the testimony of the engineers, as to the efficiency of these samples was repeatedly confirmed by the results of the analyses.

The difference between good and poor coal does not correspond in efficiency to the direct difference obtained by deductions and corrections under these new specifications. The difference between a coal which is nearly worthless and which really will not pay for carting about the city, yielding on combustion in the calorimeter between 10,000 and 11,000 B. T. U.'s, and a coal containing so little impurity, that it will yield in the calorimeter from 13,000 to 14,000 B. T. U.'s, does not represent, in efficiency, the difference between 10/14 and 14/14, but represents more nearly 75 per cent of the total efficiency obtained on the combustion of the pure coal.

So far as I know, no attempt has been made to determine exactly what these deductions should be, but they are in practical use proved to be more nearly represented by geometrical than arithmetical ratio; hence the disastrous effects witnessed in the practical use of the poor for good coal and proved by the practical experience on the municipal ferries.

A still further striking illustration of the lack of practical value in the determination of B. T. U. is offered by the specimen numbered in the table, 101. My assistant one day was called in haste to the American Museum of Natural History. He came back with a sample that was called coal, which represented a delivery to the Museum. When sifted for sizes, this sample yielded the following:

	Per cent.
Retained on $\frac{1}{4}$ in. mesh sieve (No. 1 buckwheat).....	27.8
Retained on $\frac{1}{8}$ in. mesh sieve.....	58.9
Retained on $\frac{1}{16}$ in. mesh sieve.....	1
Passed $\frac{1}{16}$ in. mesh sieve.....	9.3

The material smaller than No. 2 buckwheat was 66.6 per cent of the whole and this consisted largely of pulverized coal, while it yielded to the calorimeter 11,792 of B. T. U. and it contained only 14.45 per cent of ash and yielded 78.64 per cent of fixed carbon, it was still so finely divided that it would not burn and steam could not be maintained with it. These figures representing the approximate analysis of the coal, indicate a good coal. The material was really the sweepings of a coal shed or a barge and was not No. 2 buckwheat coal at all. Yet some one, who claimed to represent the firm who delivered the coal, came to my office and declared that inasmuch as the coal yielded 11,792 B. T. U.'s, the city had got to accept it and use it. In this case neither the proximate analysis nor the calorimeter, or both combined, ensured that the city got what it paid for.

SYNTHETIC CAOUTCHOUC: A REVIEW COMPILED FROM THE LITERATURE.*

By FRANK E. BARROWS.†

It may be well at the outset to define what is meant by "Synthetic Rubber" or "Synthetic Caoutchouc." The India-Rubber Journal, Vol. 34 (197), p. 519, defines it as a substance "built up by chemical means, * * * and possessing all the physical and chemical properties of natural rubber." If we consider, however, the molecule of the natural rubber hydrocarbon as having an empirical formula $C_{10}H_{16}$, it will be necessary to modify this definition somewhat to include other products having all the physical properties of natural rubber but whose chemical properties, owing to variations in empirical or structural formulae, may be either identical with, or analogous to, those of the natural caoutchouc. The existence of the so-called homologous caoutchoucs, probably differing in empirical formulae from that given above, will be hereinafter referred to more at length.

Synthetic rubber, then, is the product of a chemical process as distinguished from the natural product which is obtained from the latex of rubber-producing plants. In composition and properties, however, the synthetic product may be considered the same as, or equivalent to, the pure india-rubber hydrocarbon.

The distinction between the real synthetic rubber, and the so-called artificial rubbers and rubber substitutes should be kept clearly in mind. These latter, which are sometimes improperly called synthetic rubbers, generally possess some of the physical characteristics of natural rubber, but may not be even remotely related to it chemically. They may consist either of a greater or less percentage of natural rubber together with various fillers and diluents, or they may consist entirely of vulcanized oils or gums.

It is the purpose of the present article to review, more or less completely, the literature which has up to the present time appeared on the subject of synthetic rubber, its formation and constitution.

It has long been known that an intimate relationship exists between isoprene and caoutchouc, isoprene being one of the products of the destructive distillation of caoutchouc, and being itself capable, under suitable conditions, of being again converted into caoutchouc by polymerisation. Many experimenters have observed these phenomena. It is therefore natural that the term "synthetic rubber" should first suggest the product made from isoprene. A discussion of this hydrocarbon, and of the caoutchouc made from it, will, therefore, first be given.

Isoprene, as is well known, has the structural formula $CH_2:C(CH_3):CH:CH_2$, corresponding to the empirical formula C_5H_8 . It is a member of the diolefin series of hydrocarbons, containing two double bonds, and is chemically beta-methyldivinyl, or 2. methyl-1,3-

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butadien. It was first identified and studied by Williams in 1860 (*J. Chem. Soc.*, 1862, Vol. XV, p. 110), who isolated it from among the products of the destructive distillation of caoutchouc. Mention of its polymerisation was also first made by Williams at this time. He observed that when isoprene was left standing for some months it absorbed oxygen from the air, became viscid, and acquired powerful bleaching properties. When this product was carefully distilled, unchanged isoprene first passed over, the temperature suddenly rose with evolution of ozone, and the contents of the retort solidified to a pure white, spongy elastic mass having but slight tendency to adhere to the fingers. When burnt it gave the characteristic odor of caoutchouc. The composition of the oxidized product (apparently before removal of the ozone from it), was found, on careful analysis, to be C_5H_8O .

The next recorded polymerisation of isoprene was in 1879 when Bouchardat studied the action of the haloid acids on it. (*Comptes rendus*, 89, 1117-1120). When dry hydrogen chloride gas was slowly passed into isoprene at 0° , it was slowly absorbed and from the resulting product there was obtained some of the unchanged hydrocarbon and a large amount of the monochlorhydrate of isoprene, C_5H_8Cl , boiling at $86-91^\circ$. Under these conditions (3 hrs. action) the formation of a substance of higher boiling point was not observed.

On the other hand, when saturated hydrochloric acid at 0° acted on isoprene for 15-20 hours in a sealed tube with occasional agitation, and the resulting product was distilled, a solid residue, in appreciable amount, remained behind.

This solid residue persistently retained about one per cent of chlorine, its analysis otherwise giving the same percentage composition as isoprene. ($C=87.1$; $H=11.7$; $Cl=1.7$). It possessed the elastic, and other characteristics of caoutchouc; it was insoluble in alcohol, swelled in ether, and dissolved in carbon bisulfid in the same manner as natural caoutchouc. When submitted to dry distillation it gave the same volatile hydrocarbons that caoutchouc gives. All these properties seem to identify this isoprene polymer with the parent material of the isoprene itself, caoutchouc.

It should be observed, however, that in the above reaction in which the caoutchouc was obtained, the principal products of the reaction were the mono- and di-chlorhydrates, while the caoutchouc was merely a by-product, constituting not over one-sixth of the resulting product.

Hydrobromic acid, in saturated solution, was found to act in the same way as hydrochloric; it formed the elastic polymer which retained not over two per cent of bromine.

Fuming hydriodic acid acted very violently on isoprene. The elastic polymer was apparently formed, but not isolated.

Tilden, in 1882 (*Chem. News*, Vol. 46, p. 120), first reported the formation of isoprene by the depolymer-

isation of turpentine, the turpentine vapors being passed through an iron tube heated to redness. Only 20 c.c. of the isoprene fraction, however, were obtained by this process from a liter of turpentine. The isoprene thus obtained was found to act in the same way as the isoprene from caoutchouc, and gave a tough substance closely resembling caoutchouc when acted on by concentrated hydrochloric acid. The conversion of isoprene to caoutchouc by nitrosyl-chlorid is also reported by Tilden in this article.

Again in 1884 (*Trans. Chem. Soc.*, 1884, p. 410), Tilden reported his further study of the products obtained by the decomposition of turpentine vapors by heat. As in the former experiments a small amount of isoprene was obtained (200 c.c. isoprene from 4 liters turpentine). The turpentine vapors were passed through an iron tube heated to the lowest possible redness just visible in a darkened room. Benzene, toluene, m-xylene, cymene, and terpineol were among the other products identified. About 15 per cent of the product boiled above 200° , and was not further studied. Nearly 30 per cent of the product was lost in the form of gas. If the iron tube in these experiments was heated to visible redness or to a higher temperature isoprene was no longer found in the products formed. It does not appear that a yield of as high as 10 per cent of isoprene was ever obtained by this process (Tilden, *India-Rubber Jour.*, 36 (1908), p. 322). This isoprene, as in the case of the preceding experiments, was converted into caoutchouc by polymerization, contact with strong acids in the cold effecting the change.

In 1885 Wallach (*Annalen der Chemie*, 238, p. 88), found that isoprene, when it remained placed in the light for a long time, polymerised, and on adding alcohol to the resulting product there separated out a caoutchouc-like mass which hardened on exposure to the air.

Apparently unaware of Wallach's experiments Tilden in 1892 (*Chem. News*, Vol. 5, p. 265) reported a similar observation of the spontaneous polymerisation of isoprene in the following language:

"I was surprised a few weeks ago at finding the contents of the bottles containing isoprene from turpentine entirely changed in appearance. In place of a limpid, colorless liquid the bottle contained a dense syrup, in which was floating several large masses of solid of a yellowish color. Upon examination this turned out to be india-rubber. The change of isoprene by spontaneous polymerisation has not to my knowledge been observed. I can only account for it by the hypothesis that a small quantity of formic or acetic acid had been produced by the oxidizing action of the air, and that the presence of this compound had been the means of transforming the rest. The liquid was acid to test paper, and yielded a small portion of unchanged isoprene.

"The artificial india-rubber, like natural rubber, appears to consist of two substances, one of which is more soluble in benzene or in carbon bisulfid than the

other. A solution of the artificial rubber in benzine leaves on evaporation a residue which agrees in all characters with a similar preparation from Para rubber. The artificial rubber unites with sulphur in the same manner as ordinary rubber, forming a tough elastic compound."

Tilden's observations of the spontaneous polymerization of isoprene were later confirmed by Weber (Jour. Soc. Chem. Ind., 1894, Vol. 13, p. 11). From 300 gms. of isoprene Weber obtained, after nine months' standing, and by treatment of the resulting viscid, treacly mass with alcohol, a solid spongy substance of almost white color, which on drying became a light brown and was in all respects identical with india-rubber. The weight of india-rubber thus obtained was 211 gms. The principal by-products were dipentene and polyterpenes,—products of very little value.

Again in 1906 (Chem. News, 94, p. 90) Tilden reports that the spontaneous polymerization of isoprene to caoutchouc takes place slowly, requiring several years. He further states that if any attempt be made to hasten the operation, as by heat or contact with strong reagents, the greater part of the hydrocarbon is converted into dipentene, and a mixture of viscid compounds of high boiling points known as colophene,—the same product as results from the polymerization of the terpenes.

To the same effect is a further communication from Tilden in the India-Rubber Journal, Vol. 36 (1908), pp. 321-2. A review is here given of his prior experiments to date, together with a letter from which the following is excerpted,—

"The conversion of isoprene into rubber occurs, so far as observed, under two conditions, (1) When brought into contact with strong aqueous hydrochloric acid or moist hydrogen chlorid gas; (2) By spontaneous polymerization.

"In the former case the amount of rubber produced is small, and it is only a by-product attending the formation of the isoprene hydrochlorids, which are both liquid. In the latter case the process occupies several years.

"Of course many attempts were made by me to hasten the process, but it was found that contact with any strong reagent, such as oil of vitriol, pentachlorid of phosphorus, and others of milder character, led only to the production of the sticky 'colophene,' similar to the substance which results from the polymerization of the terpenes, and after a course of experiments which were carried on for about two years, I was reluctantly obliged to abandon the subject."

A more recent process for the production of caoutchouc from isoprene is that of Harries, India-Rubber Journal, May 16, 1910, pp. 630-1. This process, together with the route which led to its discovery, is described as follows:

"I have shown you that the insoluble caoutchouc can be converted into the soluble form by boiling with

glacial acetic acid. So I came to the conclusion that an equilibrium occurred, for whilst caoutchouc is truly depolymerized by acetic acid, then, however, it is equally reconverted into rubber. From this point of view I likewise heated isoprene with glacial acetic acid, and as it is very volatile I employed a closed tube. I now observed that rather over 100° C. a product separates which is actually rubber. It was noticeable that pure synthetic isoprene is polymerized more readily than natural isoprene from rubber. Later I found yet other methods. If the conditions, however, are not strictly adhered to, all sorts of thick greasy oils, resins and gums are obtained, which are not rubber. * * * The artificial rubber is quite as tough and elastic as the natural product, and of a light brown to a white color."

The ozonide and nitrosite formed from this synthetic rubber also corresponded with those from natural rubber.

Another report of the spontaneous polymerization of isoprene to caoutchouc was made by Pickles in 1910 (Trans. Chem. Soc., June, 1910, pp. 1086-7), the polymerization having been effected by standing, for the greater part of the time in the dark, for three and a half years. All the numerous tests applied to the product thus obtained, and separated from the viscous polymerized mass by alcohol, identified it as the same in composition and properties as natural caoutchouc.

Lebedeff, in a still more recent article, to be hereinafter referred to more at length, has obtained the caoutchouc polymer from isoprene by heating in a closed vessel at 150° for several days.

Turning now from the periodical literature to the patent literature, we find that the first patent for synthetic rubber was the Br. patent to St. George, 15, 544, of 1892. According to this patent turpentine vapors are passed through a heated tube and condensed by a spray of hydrochloric acid; or the vapors are condensed and then agitated with hydrochloric acid to give the solid caoutchouc. In the light of Tilden's experiments it is probable that isoprene was formed in this process as an intermediate product.

The Heinemann patents, Br. 21,772 of 1907, and French 394,795, describe the condensation of isoprene to caoutchouc by concentrated hydrochloric acid. According to these patents acetylene and ethylene, when passed through a heated tube, give divinyl, which is converted into methyl divinyl or isoprene by treatment with methyl chlorid; or the three gases may be passed through the tube together to effect the same result.

A still more recent patent for the production of caoutchouc from isoprene is the French patent 417,170, to the Badische Anilin & Sodafabrik, according to which isoprene is heated either alone for 20 hours at 120°, or with 10 per cent of its weight of conc. caustic soda at 100°. The caoutchouc is separated by precipitation with alcohol, or by steam distillation of the unchanged isoprene.

The hydrocarbon, next to isoprene in point of inter-

est in connection with synthetic rubber, is diisopropenyl, or the 2,3-dimethyl-1,3-butadien, $\text{CH}_2:\text{C}(\text{CH}_3):\text{C}(\text{CH}_3):\text{CH}_2$.

Couturier in 1892 (*Annales de Chimie*, 6 Ser., Vol. 26, p. 489) described this hydrocarbon in the following language, the hydrocarbon having been obtained in small amount by the dehydration of pinacone.

"Beta-bipropenyl polymerizes with extreme ease. * * * This properly renders all reactions with this hydrocarbon difficult. The polymerization is effected by heat alone, and the liquid is transformed into a viscous product which does not distill. Chloride of calcium acts even without heating, when left for a long time in contact with the hydrocarbon."

With sulphuric acid Couturier obtained resinous polymers.

The preceding brief description is valuable as indicating the peculiar properties of the hydrocarbon. Of even more interest, however, are the two articles by Kondakoff in *Journal für praktische Chemie*, 62 (1900), p. 175, and 64 (1901), p. 109.

The first of these articles describes the heating of the hydrocarbon with alcoholic potassium hydroxide (1:KOH, 3:EtOH) to 150° for 5 hours. A part of the hydrocarbon remained unchanged; a part was polymerized to a white leathery elastic mass, insoluble in water, but soluble in hydrocarbons, ether and alcohol, and which did not distill with steam. The similarity of this product to caoutchouc was noted.

Again in the second article Kondakoff records a similar polymerization of this same hydrocarbon, by letting it stand in a closed bottle in diffused light for about a year. The hydrocarbon in this case was completely converted into a solid white spongy mass. Under the microscope this mass appeared amorphous; it was tasteless and odorless and as elastic as caoutchouc. It did not appear to undergo change in the air and was entirely insoluble in benzine, ligroin, chloroform, carbon bisulfid, ether, alcohol, acetone and oil of turpentine, swelling only in benzine. The author observed that this polymer appeared to be a higher product of polymerization than the one referred to in the preceding article.

The polymerization of this same hydrocarbon into its caoutchouc-like polymer has also been effected by heating for several days under pressure. An account of such polymerization is reported by Lebedeff in the article referred to below.

The most recent publications on the polymerizations of this hydrocarbon (diisopropenyl) are the Br. patent 14,281 of 1910, and the French patent 417,768 (See the *India-Rubber Jour.*, Feb. 4, 1911, p. 14, and *Gummi Ztg.*, Feb. 10, 1911, p. 702, respectively), to the Badische Anilin & Sodafabrik, according to which the hydrocarbon is polymerized by heating, either alone or with the addition of such indifferent agents as water, a solution of common salt, or alcoholic caustic potash. After the polymerization any unchanged hydrocarbon is distilled off with steam. The product

is a white elastic substance, soluble in benzine, from which it is precipitated, unchanged, by alcohol and it possesses the typical properties of caoutchouc.

Closely related to isoprene and to diisopropenyl, being in fact the mother substance of them, is erythrene, or divinyl, the 1,3-butadien, $\text{CH}_2:\text{CH}:\text{CH}:\text{CH}_2$. British patent 15,254 of 1909 (*Gummi Ztg.*, Nov. 18, 1910, p. 261) to the Farbenfabriken vorm. Fr. Bayer & Co. of Elberfeld describes the polymerization of this hydrocarbon to caoutchouc, the conversion being effected by heating under pressure either alone or with the addition of a reagent which assists in the polymerization. Lebedeff has also studied this hydrocarbon and its caoutchouc polymer, the polymerization having been effected in a similar manner, by heating at 150° in a sealed tube for several days. The polymerization of this hydrocarbon is also briefly referred to in *Chemiker Ztg.*, Feb. 7, 1911, p. 63, and it is here observed that the polymerization takes place much more readily with this hydrocarbon than with isoprene.

Another hydrocarbon belonging to the same group as the preceding, and closely related to isoprene (beta-methyldivinyl) is piperylene or the alpha-methyldivinyl, $\text{CH}_3:\text{CH}:\text{CH}:\text{CH}_2$. Thiele (*Annalen der Chemie*, 319 (1901), p. 227, has found that this hydrocarbon also, after several months standing in the dark, gives "a very small amount of a rubber-like (gummiartigen) substance, probably a polymerization product." Most of the hydrocarbon in this experiment, however, remained with its boiling point unchanged.

The intimate relation to each other of the four hydrocarbons which have been described, and from which synthetic caoutchouc has been obtained, will be much clearer from a comparison of their structural formulae,—

$\begin{array}{c} \text{CH}_2 \\ \\ \text{CH} \\ \\ \text{CH} \end{array}$	$\begin{array}{c} \text{CH}_2 \\ \\ \text{C}-\text{CH}_3 \\ \\ \text{CH} \end{array}$	$\begin{array}{c} \text{CH}-\text{CH}_3 \\ \\ \text{CH} \\ \\ \text{CH} \end{array}$	$\begin{array}{c} \text{CH}_2 \\ \\ \text{C}-\text{CH}_3 \\ \\ \text{C}-\text{CH}_3 \\ \\ \text{CH}_2 \end{array}$
Erythrene, divinyl, or 1,3-butadien.	Isoprene, 2-methyl- divinyl, or 2-methyl-1,3- butadien.	Piperylene, 1-methyl- divinyl, or 1-methyl-1,3- butadien.	Diisopropenyl, 2,3-dimethyl- divinyl, or 2,3-dimethyl- 1,3-butadien.

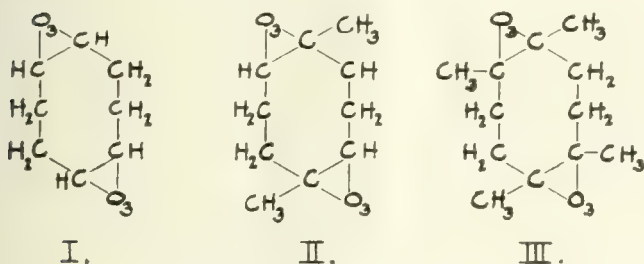
It will be seen that all four of these hydrocarbons belong to the divinyl series, containing the following common nucleus, $\text{C}:\text{C}:\text{C}:\text{C}$, and having respectively the empirical formulae C_4H_6 , C_5H_8 , C_5H_8 , and C_6H_{10} . It is known that the caoutchouc from isoprene has the same percentage composition, and hence empirical formula, as natural caoutchouc, viz.: $(\text{C}_{10}\text{H}_{16})_n$. It should follow, since the formation of synthetic caoutchouc is by a polymerization reaction, that the caoutchoucs from erythrene, piperylene, and diisopropenyl should also have the same empirical formulae as the hydrocarbons from which derived, or $(\text{C}_8\text{H}_{12})_n$, $(\text{C}_{10}\text{H}_{16})_n$, and $(\text{C}_{12}\text{H}_{20})_n$, respectively. Such formu-

lie are also indicated by the ozonides referred to be low.

A valuable contribution to the literature on the subject of these diethylenic hydrocarbons (containing the nucleus C:C:C:C) is an article by Lebedeff dealing with their autopolymerization found in the Journal of the Russian Physical-Chemical Society, 1910, Vol. 42, No. 6, p. 949. According to this article,—

"The polymerization of these hydrocarbons takes place in such a typical manner that it may be considered a general characteristic of the whole group. The rapidity of the process, sometimes exceedingly slow at normal temperatures, increases very rapidly with rising temperature."

The polymerization of the three hydrocarbons, erythrene, isoprene, and diisopropenyl was investigated in detail by Lebedeff, a caoutchouc polymer being obtained from each. The polymerization was effected by heating in a sealed tube at 150°, the process requiring 6 to 7 days for its completion in the case of erythrene, and 8 to 10 days in the case of isoprene and diisopropenyl. Ozonides were formed from each of these polymers, the ozonides derived from the erythrene, isoprene and diisopropenyl polymers having the formulae $C_8H_{12}O_6$, $C_{10}H_{16}O_6$, and $C_{12}H_{20}O_6$, and yielding on decomposition with water succinic aldehyde, laevulinic aldehyde, and acetylacetone respectively. To these ozonides, therefore, the following structural formulae were assigned,—



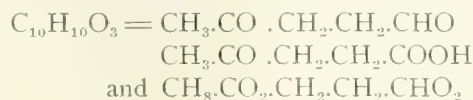
The ozonide reactions of caoutchouc are particularly valuable because of the light they throw on the constitution of the caoutchouc molecule.

Two theories as to the constitution of this molecule have thus far been proposed. In view of the importance of the subject to which they relate it is desirable to examine these theories in detail. The first is the cyclooctadiene theory of Harries; the other is that proposed as an alternative by Pickles before the British Association last year. The former theory is given in two articles appearing in No. 36 of the Gummi Ztg., March, 1910, and Chemiker Ztg., 1910, March 26, p. 815, and translated into the India-Rubber Journals of June 13, 1910, p. 772, and May 16, 1910, p. 630, respectively. The following is excerpted from the first of these articles,—

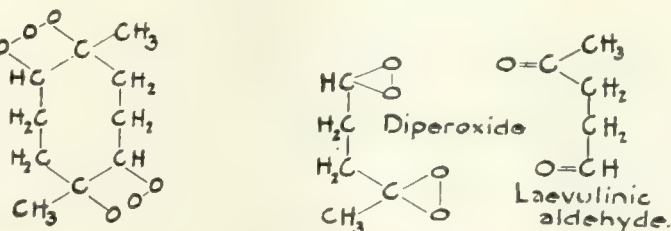
"As I will subsequently show, rubber is to be regarded as a polymerization product of a hydrocarbon ($C_{10}H_{16}$) with a ring-like arrangement of eight carbon atoms. It might be assumed that by suitably strong depolymerization treatment it would be pos-

sible to reduce the rubber to such a hydrocarbon. I have found that depolymerization can, in fact, be effected, especially by long boiling of the rubber in toluol or zylol. However, the actual product which should be first produced, according to my theory, is not obtained, probably on account of its instability, but in its place allied compounds, such as dipentene and other hydrocarbons resembling turpentine. The fact, however, that rubber is depolymerized by protracted boiling in solvents of high boiling points is of importance for the question of the determination of its constitution. Regarding its molecular weight we know nothing definite; the experiments undertaken lately by Henrichsen cannot be regarded as decisive."

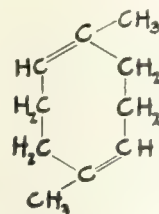
Caoutchouc is a hydrocarbon of the empirical formula $C_{10}H_{16}$, it is optically inactive, and has therefore no asymmetrical carbon atom. By bromination it takes up four atoms of bromine and accordingly possesses two ethylene linkings. Dissolved in chloroform and treated with ozone two molecules of ozone are added; and being readily soluble the formula of this so-called diozonide can be determined; it is $C_{10}H_{16}O_6$. At the same time the ozone treatment causes a depolymerization of the high rubber molecule. On boiling this diozonide with water it decomposes into laevulinic aldehyde, laevulinic acid, and a crystalline body which I have named laevulinic aldehyde diperoxide.



"From this it would appear that caoutchouc ozonide must contain an eight carbon ring, for the ozone is situated at the ethylene linkings, and in splitting up, division of the molecule occurs at the positions where the ozone has entered, with the formation of compounds, aldehydes, or acids, containing oxygen. We come then to the following graphic formulae:



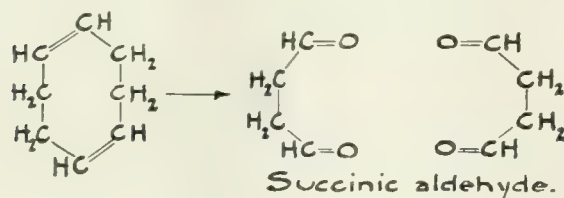
and the hydrocarbon which forms the basis of the caoutchouc molecule, and by the polymerization of which the same is produced:



1:5-dimethylcyclooctadiene (1:5)

Before I showed that in all probability the eight

carbon ring occurs in caoutchouc, this carbon combination had not been discovered in nature. Shortly afterwards Willstaetter found an alkaloid in the root of the pomegranate, having likewise an eight carbon ring combination, and eventually he built up therefrom the lower homologue, which forms the basis of rubber chemistry. From this hydrocarbon, cyclo-octadiene, I have proved that the two ethylene linkings are situated in 1:5 position as caoutchouc, for it yields in the splitting up with ozone, succinic aldehyde.

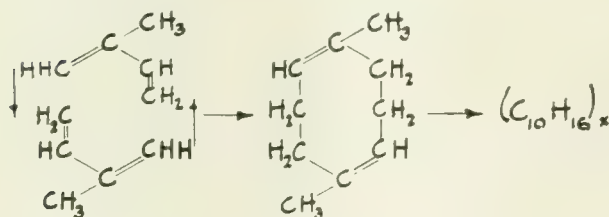


On heating to 70° C. this cyclo-octadiene is polymerized, and under special conditions I obtained therefrom a product extraordinarily similar to rubber. * * *

All attempts to extract the two molecules of ozone from caoutchouc by suitable reduction, and even to regenerate the hydrocarbon, have hitherto failed; its synthesis also has not been accomplished."

Again in the India-Rubber Journal of May 16, 1910, in discussing the formation of caoutchouc from isoprene,—

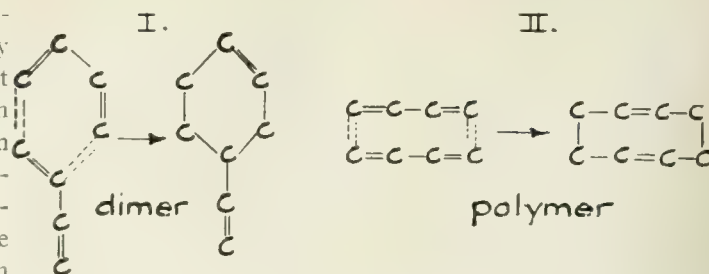
"In seeking to discover how the reaction occurs in the polymerization, the conclusion is reached that isoprene first changes into dimethylcyclooctadiene, with condensation at the carbon atoms in 1:4 position, as in all addition reactions, results from bodies with conjugated double linkings.



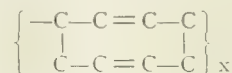
"The condensation must take place in this way because on oxidation with ozone laevulinic aldehyde is formed."

A conclusion similar to that reached by Harries is also reached by Lebedeff in his article above referred to. This article, however, discusses the polymerization of the whole class of diethylenic (C:C.C:C) hydrocarbons, and from a broader aspect. A further discussion of this article bearing both directly and indirectly on the constitution of rubber will therefore be given. Quoting further from this article:

"A closer examination of the products of polymerization, which consist of dimers and polymers of the diethylenic hydrocarbons, shows that we have to do with two parallel processes:



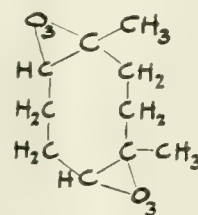
"The first process leads to the formation of a six-membered ring with two double bonds, one in the ring, and the other in the side chain. The second process leads to the formation of an eight-membered ring with two double bonds, closely related to



"From a consideration of the above proposed system it is obvious that a symmetrically arranged molecule can give rise to only one dimer with a six-membered ring. Such are divinyl and diisopropenyl. As a matter of fact, the dimers of divinyl and diisopropenyl consist of only one hydrocarbon. * * * An unsymmetrically arranged molecule may give rise to four dimers. In the case of isoprene two such dimers have been observed; the other two it has not been possible to identify. Those observed are dipentene and a hydrocarbon of boiling point 160-161° at 760 mm."

Lebedeff's experiments with erythene, isoprene and diisopropenyl and the ozonides obtained from them have already been referred to above. In discussing the ozonide from the isoprene polymer it is further observed,—

"The system of polymerization proposed by me foresees the possibility of the formation of another isomeric ozonide



"Whether this isomer is formed is not yet clear."

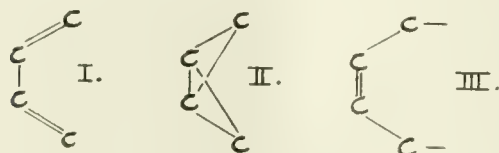
The non-existence, or the existence if at all, only in very small amounts, of such an isomeric ozonide would seem to indicate strongly that the unsymmetrical position of the methyl group in the isoprene molecule exerts a marked influence on its polymerization. This lack of symmetry in hydrocarbons such as isoprene is indeed mentioned by Lebedeff as one of the factors influencing the polymerization reactions of these hydrocarbons. It was found by him that the polymerization by light is much slower in the case of isoprene than in the case of diisopropenyl. It is also interesting to observe in this connection that divinyl has similarly been found to polymerize much more readily than isoprene (Chem. Ztg., above).

The conditions under which the polymerization is

The existence of such bonds in rubber is strongly indicated by its vulcanization reaction. Weber (Chemistry of India-Rubber) states that as little as 2 to 2.5% of sulfur is sufficient to effect complete vulcanization and that the resulting vulcanized rubber possesses the highest degree of elasticity and distensibility combined with the highest degree of tensile strength. Rubber possessing a higher coefficient of vulcanization sometimes shows higher tensile strength, but at the expense of the other physical constants.

In summarizing the foregoing facts and theories it would seem that any acceptable theory of the constitution of the caoutchouc molecule must not be inconsistent with the following observed facts,—Caoutchouc yields on treatment with ozone a product of depolymerization and addition, caoutchouc ozonide; it gives on treatment with bromine a product of addition but not of complete depolymerization, the so-called tetrabromide; it gives on depolymerization by boiling in solvents of high boiling point, not the cyclooctadiene, but the more stable six-membered-ring terpenes, such as dipentene; on destructive distillation it gives a series of products of widely varying complexity from isoprene through dipentene to the more complex and higher boiling products which result particularly from vacuum distillation; it is converted into hydrocarbons of the paraffine series by hydrogenation; it may be formed by the polymerization of isoprene and similar hydrocarbons, but not from dipentene, and it may itself be converted from a lower to a higher state of polymerization and vice versa; and finally it may be completely vulcanized by a very small amount of sulfur.

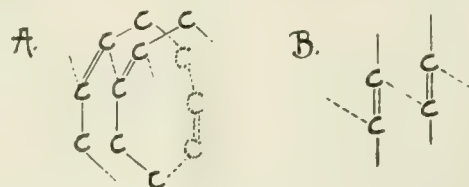
A valuable suggestion bearing indirectly on the present subject is found in a communication by Wechsler in Chem. News, Vol. 100 (1910), p. 279. In discussing the reactions of bodies containing in their molecule the group $\text{C}=\text{C}-\text{C}=\text{C}$ Wechsler suggests that if we write the carbon atoms more according to their relative positions in space, as in (I), then, if the double bonds attract each other, we have (II), in which the end atoms are much more open to attack than the middle ones.



From the formula (II) suggested by Wechsler it is but a step to formula (III), which has been hereinbefore referred to.

If we apply the above suggestion to the long chain proposed by Pickles, and write the carbon atoms more according to their relative positions in space, we would not expect to obtain a single ring of at least forty carbon atoms, but we would expect this long ring to double back upon itself at about the sixth carbon atom. Then if the double bonds, which recur regularly at the fourth and eighth carbon atoms, attract each other, and are able mutually to satisfy each other, they might

be expected to join together and give a molecule with a configuration similar to that of a helix or spiral spring,—the contiguous alternative double bonds being thus joined to, and satisfied by, each other. A fragment of such a molecule might be represented graphically as in (A) with the double bonds joined together as in (B).



Such a spiral or helical molecule would be closely related to the cyclooctadiene ring, as will be apparent from the following diagrams. From these it will be seen that the alternative double bonds are in practically the same relative positions whether we have a series of cyclooctadiene rings (B) or a spiral (A). An explanation of the eight membered ring, or a structure closely related to it, is thus made possible by the modifying action of these contiguous alternative double bonds.



Such a spiral molecule could form the tetrabromide by addition of bromine at each double bond without depolymerization; it could break completely at each alternative double bond with rearrangement of the ring to give the stable six-carbon-ring terpenes; it could break at each alternative double bond in a different manner to give the ozonide; on pyrogenic decomposition by heat this molecule might break at any of its double bonds to give products of varying complexity, but all having the empirical formula $(\text{C}_5\text{H}_8)_x$; by hydrogenation such a molecule might be expected to give a saturated hydrocarbon of the paraffine series. It is proposed that the bonds at the ends of such a molecule are free, or relatively free, so that this molecule can further react to give a more highly polymerized product; and it is further proposed that by the saturation of these bonds by sulfur vulcanization is effected.

This theory requires that the spiral molecule have its alternative double bonds joined together and saturated by each other, but joined in such a manner that upon treatment with suitably strong reagents addition may take place much as if the double bonds still exist in a modified and less reactive form.

The spiral or helical theory above suggested has not, to the knowledge of the present writer, heretofore been published. It is with not a little hesitation that it is offered at this time. But if it shall aid even a little in the ultimate solution of the nature of the rubber molecule its object will have been accomplished.

TECHNICAL EDUCATION—ITS FUNCTION IN TRAINING FOR THE TEXTILE INDUSTRY.*

By CHARLES H. EAMES.†

Perhaps no subject is more broadly discussed, outside of religion and politics, than education. While it is recognized by every civilized nation that a highly developed system of education in all departments of human life is absolutely necessary to the advancement of a people, the effect of over or under stimulating any particular department cannot be measured in the same degree of accuracy that is practiced in mathematics or science. In the study of mathematics or science it is comparatively easy to manipulate various quantities and determine the effect of each upon the result in any given experiment or test. But the same methods cannot be practiced quantitatively when problems involve the analysis of the net work of social and industrial relationships.

Within the last fifteen years the particular branches of industrial and technical education have received more attention from the educator, the manufacturer, and the engineer, than have any of the older departments in our educational system. The practical man and the theorist, the philanthropist, and the reformer, men of wisdom as well as men with fads, have seen in the development of technical education either unlimited possibilities in the uplift of their brother man, or a necessary means of solving some real and complicated problem in the work-a-day world.

Germany's increase in wealth, industry and commerce is frequently given as the direct result of her highly developed system of industrial and technical education, and some would have us believe that this system is the sole cause of her growth. What would be the effect upon Germany if her educational system be removed no one can accurately determine yet no one would hesitate to reason that such a change would be seriously detrimental to the welfare of the country.

To what extent manufacturing, engineering and commerce owe their development in our own country to the technical world cannot be calculated or even estimated. We may fairly reason that in the absence of the technical school of applied science, pure science and mathematics would have rested in a comatose state and have been of little practical value.

Our forefathers found wealth in the soil, the streams and the forests, and so long as their resources were immediately at hand and the supply adequate to meet all wants there was no demand for manufactured material. Time was not so important an element as at present. The increase in population and the recognized limit in natural resources, however, soon caused necessity to turn to other pursuits. The manufacturing of higher class articles from the natural

products was a logical step. The desire to extend the markets to the inland settlements as well as to facilitate communication and travel with them brought the problem of waterways and the necessity for skilled engineering. The lack of men to intelligently undertake such work, and the outlook for greater engineering tasks in the future made a training school obligatory. The development of this school naturally drew about it so much of mathematics and science as might be considered essential for the engineer. Later he in turn found new fields requiring special application of other branches of science. He returned to the school with his problems and demonstrated the need of an ever widening curriculum.

The co-operation of the graduate engineer with the instructing staff of his alma mater has been perhaps the greatest factor in extending the usefulness of these schools. This progressive development has led, in some cases, to the establishment of other schools for particular training. In some such way as this, industry has developed a great many types of special schools, each working in its particular department of the whole technical and industrial educational field.

Although not permitting of actual measurements the influence of this growth is recognized by all to add much to the wealth of this country. The technical school as first organized was to train men for civil engineering, but soon it became apparent that manufacturing industries, too, would profit by having the scientifically trained men, so today we find the largest part of the technical school alumni engaged in some branch of manufacturing. Many of these industries owe their origin to the product of these technical schools, and there are many cases where an industry or concern has been rescued from the class of industrial derelicts by a manager who has had technical training.

While the iron, steel, electrical and chemical industries are more dependent upon scientifically trained managers, the textile, an older and more conservative industry, is fast finding the same need. Some years ago a treasurer of a successful mill was heard to remark with a sneer, "I don't want any technology graduate coming down here in his patent leather shoes and telling me how to run a mill." The silent reply to this man's belittling estimation of technical education may be found by comparing the change in market values of his mill stock and that of a mill which has made a practice of employing an increasing number of technical graduates. The stock of the former has constantly fallen until it is nearly one-half its original value and below par, while the latter has constantly advanced until a share is now worth more than three times its par value.

We hesitate to resort to statistics for they do not always tell the whole story. They are nevertheless some measure to use in estimating the results. If our summarization from the catalog of the Massachusetts Institute of Technology be correct there are consid-

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erably more than a hundred of the graduates engaged in the textile industry or in a business directly dependent upon the textile industry. This does not include those who are mill engineers, or those in steam, hydraulic or electrical machinery design and construction, whose products enter to a great extent in the success of textile manufacturing. Neither does this figure, of course, include those who are not graduates but who have found profitable positions in the textile field and are surely exerting their influence and helping to make technical training felt in the industry. It would be necessary to consult the alumni lists of the other technical schools to gain a fuller appreciation of the increasing value placed by the textile industry upon this department of education. Further than this in making up our account we should not forget to give due credit to the fact that the industry has deemed technical education of sufficient importance to establish special schools that its particular requirements may be better met. In one of these schools four Massachusetts Institute of Technology men are on the instructing staff, and there have been connected with the school during its brief existence nine instructors who received their training at the Massachusetts Institute of Technology and who in turn are bound to instill into the training of their pupils much which cannot better be described than "Tech. Spirit." On this same instructing staff have been graduates from at least ten other technical institutions, which fact serves as another example of the extended influence of scientific education in the textile industry.

One could hardly conceive of the increase in active cotton spindles in the United States of from $2\frac{1}{2}$ million in 1860 to over 29 million in 1910, accompanied by an increase in the consumption of cotton from 840,000 bales to 4,500,000 during approximately the same period without realizing that engineering in the designing, constructing and operating of the cotton mill must play an ever increasing part. In comparing the size of the woolen mill of 1860 with the magnitude of the present day worsted mill plant of the type of the Arlington, the Ayer, or the Wood, one has some appreciation of the engineering problems involved and the training which has made the successful solution possible.

The increase in production with demand for improved efficiency naturally places problems of improvements in machinery upon the builders, and they too have found the need for specially trained men of technical preparation. Hardly a machine shop building textile machinery can afford to be without its corps of expert engineers. The increase in pounds of cotton consumed per active spindle may be taken as an indicator that finer grades of material are being manufactured in this country and this too will bring its more complex problems for the manufacturer and designer of machines?

Within the past few years we have learned of a new kind of engineer, or better, an engineer with a

new title, indicating again the further subdivision of the field of industrial training. While it was undoubtedly one of the recognized tenets in the establishment of the technical school that increased efficiency of operation would accrue, yet the field of the Economist or Efficiency Engineer was not conceived, but today manufacturers are realizing more and more that dividends can be made or lost in the waste, not only of materials but by the improper direction of labor. The time element in production demands in many cases more consideration than the material, and the efficiency of a human being or process must be and is determined with almost as great an accuracy as any mechanical or electrical device. The industries working upon the smallest margin of profit find the greatest need for this engineer. One of the most lucrative fields for him is the textile, and if careful examination were made one would find many of the successful mills quietly employing engineers with broad technical training to study their plant, methods and employes that the efficiency of all of these elements may be made the highest.

The technically trained chemist is more and more in demand by the textile industry. His chief work is to increase the efficiency of insuring the quality of the raw material by reducing the cost of manufacture, and by reducing waste or redeeming valuable by-products from this waste. The chemist's work has prevented the increasing amount of refuse from the textile mills from polluting the water supplies of nearby communities. Thus while he may not in such cases be engaged to directly improve the industry, he has made it possible for business to expand without being a menace to any particular locality. Logically, then, we should add to the number of Massachusetts Institute of Technology graduates earlier referred to, as connected with the industry, those who as consulting chemists or as employes of the various boards of health are helping to promote indirectly the textile industry.

The increasing quantity of colored goods produced yearly in the United States with the range of dyestuffs made possible by the development of coal tar products, coupled with the impure water supply, shows another pressing need for the technically trained chemist. The day of the "rule-of-thumb" dyer with his stereotyped receipts applied with little regard for varying conditions is fast receding. The dyer finds a knowledge of the fundamental laws of chemistry an absolute necessity in coping with the perplexing problems of more intricate dyestuffs, complexity of material and contaminated water supply.

Germany is a recognized leader in the manufacture of dyestuffs and chemicals. The early introduction of her products to this country required her own experts as demonstrators, but only recently a technically trained chemist in the employ of one of these importing houses was heard publicly to state that the German experts here were no longer required, for their places in this country are being taken by our own

technically trained chemists. This means but a step toward the increase of dyestuffs manufacture in this country.

The tendency of larger growth of the textile industry with its dependent branches has been indicated and I have endeavored to show that technical training is to be a prerequisite for success. If this condition of affairs be accepted certainly one of the most attractive fields is open for the further application of scientific laws and principles. A few of the early graduates of technology have entered textile manufacturing, but nearly every one of the later classes has added some new representative. The fact that the earlier graduates secure these later class men to fill positions may be taken as an indication that the expansion of the textile industry has opened a new field where there is an ever increasing demand for the graduate with a technical education.

The fourth annual meeting of the American Institute of Chemical Engineers will be held in Washington, D. C., Wednesday to Friday, Dec. 20th to 22d. A number of important papers will be presented on the general subject of Patents, and the Manufacture and Testing of Explosives, as well as on a number of other important chemical engineering subjects. One day will probably be devoted to visits to the technical chemical engineering plants in Baltimore and vicinity. Visits to laboratories and other points of interest in Washington will also be arranged for.

According to the United States Geological Survey, the production of natural graphite in the United States in 1910 was 35,945 net tons of amorphous graphite, mainly used for fertilizer filler, valued at \$81,443, and 5,590,592 lbs. (2,795 net tons) of crystalline, valued at \$295,733.

The production of artificial graphite at Niagara Falls, which had averaged 6,000,000 lbs. (3,000 net tons) for a number of years, increased to 13,149,100 lbs. (6,574 net tons) in 1910, being valued at \$945,000.

The imports for consumption of graphite into the United States in 1910 came mainly from Ceylon and Mexico, and were nearly the same in quantity and value as in 1909. It is significant to compare the total value of the imports, \$1,872,592, with the total value of the domestic product, both natural and artificial, which was \$1,322,176. The domestic demand for graphite and graphite products is undoubtedly increasing, and it is encouraging to note that this demand is being met principally by increased domestic production rather than by increased importation. Domestic flake graphite is replacing the Ceylon graphite to some extent in the manufacture of crucibles for the metal industries. There appears to be no reason why this replacement should not increase, for in Germany flake graphite similar to much of the American product has for generations been successfully applied to this use.

THE FUNCTION OF TECHNICAL SCHOOL LABORATORIES.*

By H. W. HAYWARD.†

The keynotes in the history of all successful industrial enterprises of recent years have been standardization and efficiency. The profits returned in many cases have been about in proportion to the completeness with which these features of the organization have been worked out with regard to raw material, process of manufacture and finished product.

In order that any industrial operation may be carried out economically, it is necessary that the organization be complete and perfectly adapted for the work, and that the different departments be supplied with material of a suitable and uniform quality. On this account the success of any manufacturer depends largely upon his ability to control his raw material, or what is better, to adapt material from a great many sources to his needs by preliminary treatment.

In large plants complete laboratories for testing materials, chemically and physically, are unusually maintained, where the treatment required for any raw material may be determined and its course from department to department through the works be closely watched in order that the product shall be of standard quality. Smaller plants cannot afford the complete laboratory and get along with very meager equipment; others make no laboratory tests at all and depend entirely upon the judgment of their superintendent and foreman, or upon experiments made in the works.

In order that a steady advance may be made along all lines it is necessary that constant study and research shall be carried on to improve the processes of manufacture in use, to develop new and more economical methods and to find uses for by-products that may be useless at present. Only the very large companies with their complete laboratories can give much time to research, and only a comparatively small number of them care to employ competent men for this purpose, as the special apparatus required is costly and often very little immediate return is realized from considerable outlay. The smaller plants cannot afford to go into the question to any very great extent, as their equipment is inadequate, and experiments carried on in the works are usually rather expensive and interfere with the routine of the plant.

Large private laboratories are conducted in different cities and they do splendid work for their clients. Although a great deal of their work is of a confidential nature, and on this account not available to the public, some of these laboratories devote a considerable amount of time to general questions of importance and contribute valuable data for publication in various journals.

Many consulting engineers maintain laboratories,

*Paper presented before the Congress of Technology at the 50th Anniversary of the Granting of the Charter of the Massachusetts Institute of Technology.

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but they are usually for special purposes and not very completely equipped. Very valuable data have been obtained in many of these laboratories, however, and has been published for general use. The consulting engineers are very active in the engineering societies and their work in this line has been of great service to their profession.

There are two other classes of laboratories to be considered: those conducted by the Government, and those in the various technical schools. The Government laboratories do very efficient work along very broad lines, but it is rather difficult for other interests than the Government to get work into them.

The laboratories of a technical school are particularly adapted for work that cannot be carried on efficiently in other places. The complete equipment of the technical school laboratory allows the undertaking of almost any problem, and among its instructing staff some man can usually be found who is specially fitted to attack it. The large number of students who, every year, are required to carry out some special work as a graduation thesis, furnish the means where problems of considerable magnitude may be thrashed out under competent supervision and at very little expense to the outside interest. In almost all technical schools special men can be put to work on any problem if the interested party will pay a reasonable sum for the man's time, the laboratory being placed at his disposal, it of course being understood that the work shall not in any way interfere with the educational requirements of the laboratory. Co-operation of this kind makes possible research along industrial lines which cannot be carried on in any other way. The instructing staff in many technical schools are encouraged to work upon problems of general interest and the results of their labors are many times of great value.

A great many important problems have been solved in the laboratories of the Massachusetts Institute of Technology, and every line of industry has been benefited by the data resulting from work accomplished by members of the instructing staff or by students in their thesis work. If a study be made of the personnel of many important commissions and committees, it will be found that many Institute professors are doing valuable work in the solving of industrial problems with assistance of the laboratories, and many of them are known the world over for their labors in the interest of science and engineering.

The work that has been accomplished by the laboratories of the Institute of Technology, in the engineering field, needs no praise. It speaks for itself. The efficiency of the graduates of the Institute shows just as well that the educational side of the problem has been well handled, and it would seem that the proposed new Institute with its better facilities, closer co-operation with state, city and industrial interest, should stand at the head in everything pertaining to standardization and efficiency in both the industrial and educational world.

THE EARNING POWER OF CHEMISTRY.*

By ARTHUR D. LITTLE.

It may fairly be claimed for chemistry that it is at once the most fundamental and the most comprehensive of all the sciences. Its province, in the classical definition of Ostwald, is "The study of the different forms of matter, their properties, and the changes which they undergo." Thus defined, chemistry embraces the material universe, our solar system, the most distant stars and the flaming nebulae no less than the dust speck within the universe, on which we live and which we call the earth. It includes within its subject matter the physical basis of our own bodies and of those of every living thing upon the earth. It is directly concerned with the air we breathe, the water we drink, the food we eat, the materials upon which we expend our labor, and the things which we buy and sell.

Modern chemistry had its birth in the eighteenth century study of the air and its relation to the processes of respiration and combustion. Prof. Ramsay has said that "To tell the story of the development of men's ideas regarding the nature of atmospheric air is in great part to write a history of chemistry and physics." The story is one which has reached its culminating interest in our own most recent times.

Within the memory of the youngest undergraduate in chemistry the brilliant researches of Ramsay, Raleigh and other chemists have disclosed the presence in the air we breathe of five new gases of remarkable and in some respects unique properties.

The heavy toll of life in mine disasters would be unsupportably heavier were it not for the Davy lamp, the fire-damp indicators, the rescue outfits and the regulation of explosives, all of which have become possible only through the growth of chemical knowledge. Ventilating systems as applied to theatres, halls and dwellings are based on chemical studies of the rates and causes of increase in the carbonic acid content in the air of rooms. The proportion of sulphur permissible by law in illuminating gas finds its justification in similar studies on the air in rooms in which such sulphur-bearing gas is burned.

The chemical and biological study of public water supplies, which received its first systematic development little more than twenty years ago at the hands of Drown and Mrs. Richards in the laboratories of the Massachusetts Institute of Technology, has been the means of saving countless lives throughout the world and has led to such understanding and made possible such control of sources of pollution as to almost justify the statement that for every case of typhoid fever some one should be hanged. Chemistry can now determine in advance of use the suitability of a given water supply for use in boilers or for the requirements of any special line of industry, as paper-making, dyeing, cloth finishing, brewing and so on. Fur-

*From a public lecture to business men delivered June 29, 1911, under the auspices of the Indiana Section of the American Chemical Society.

thermore it supplies the means for correcting undesirable characteristics in a water supply as by use of filtration apparatus, coagulants, water-softening systems and the Moore method for the destruction of the algae which in many waters are the cause of unpleasant tastes and odors.

Nowhere is the practical value of chemistry in its relation to the affairs of every-day life more strikingly demonstrated than in connection with our food supply. Chemical fertilizers are in large and constantly increasing measure responsible for the enormous total of our agricultural products. The whole fertilizer business is under the strictest chemical control, and the farmer buys his fertilizer on the basis of a knowledge of its composition and effective value which puts the average purchasing agent of a manufacturing company or public service corporation to shame. The Association of Official Agricultural Chemists, and the laboratories of the agricultural colleges and experiment stations throughout the country are doing more to keep down the cost of living than all the lawmakers we send to state capitols and Washington.

One of the most insistent of the demands of growing plants is that for nitrogen in form available for plant food. A small proportion of the necessary supply of nitrogen in the assimilative form is derived from the manure of farm animals and from animal wastes of various kinds, but for many years the world has depended upon the nitrate beds of Chili as the chief source of this indispensable element of plant growth. It is bad enough to be tied in this way to a single faraway deposit, but the situation becomes alarming when we discover that this deposit can hardly meet the world's demand for nitrate for another twenty years. One may contemplate the Malthusian theory with indifference or even with disbelief, but here is a condition not to be gainsaid. The world must do something to meet it within twenty years or the world must make up its mind to starve. Fortunately for the world the chemists are already doing something. They have recognized that 33,800 tons of nitrogen are pressing down upon every acre of land and have boldly attacked the problem, of rendering available such portion of this inexhaustible supply as the world may need. The methods employed have been daring and brilliant in the extreme.

In 1785 Cavendish in a paper before the Royal Society describes the production of nitric acid by the passage of an electric spark through air. A hundred years later Bradley and Lovejoy at Niagara Falls, by drawing air through an apparatus by which 400,000 arcs were made and broken each minute, demonstrated the possibility of the commercial manufacture of nitrates from atmospheric air. Birkeland and Eyde in Norway pass the air through furnaces in which it comes in contact with enormous flaming and rotating arcs. Rossi in Italy brings the air in contact with highly incandescent material of special composition. Although by these several processes nitrate

has been produced by thousands of tons it is doubtful if the artificial product can yet compete with Chilian niter. Even now, however, the margin is not a wide one and the results already accomplished amply prove that when our agriculture begins to feel the pinch of a failing nitrate supply the chemist may safely be relied on to meet the situation. This assurance is rendered doubly sure by the fact that a solution of the problem along altogether different lines is already nearly or quite within our hands. Dr. Frank has shown that by heating calcium carbide, itself a comparatively recent product of the laboratory, in a stream of nitrogen there is formed a new compound, calcium-cyanamide. The practical interest in this compound depends upon the fact that when exposed to a current of steam it decomposes into ammonia and carbonate of lime and that the same reaction takes place slowly in the soil when the cyanamide is mixed therewith. Since the nitrogen in ammonia is directly assimilable by plants and since calcium carbide requires for its production only lime and coke and power we may view without serious concern the approaching failure of the Chilian nitrate beds.

But it is not only on the side of agriculture that chemistry touches our food supply. Chemistry pervades the packing industry, reducing the cost of food by utilization of by-products of the most varied character from oleomargarine to glycerine and soap and from soap to pepsin and adrenalin. To Atwater and his co-workers we owe our knowledge of the energy-producing value of different foods in the human economy and to Wiley and those other chemists behind him on the firing line we are indebted for the far-reaching benefits of the Pure Food Law.

Carbon disulphid made in the Taylor electric furnace has preserved the wine industry of France by destroying the phylloxera as it is ridding our own fields of prairie dogs and our elevators of rats and mice. Bread-making and brewing are coming each year more and more within the recognized domain of chemistry¹ which is at the same time greatly enhancing the value of our staple crop by the increasing production of glucose, corn oil and gluten. Exactly one hundred years ago Kirchhof discovered the inversion of starch to glucose by dilute acids. Today the United States alone is richer by \$30,000,000 a year by reason of that discovery.

The relation of chemistry to the clothes we wear is perhaps less obvious but still of the first importance. More land is planted to cotton and cotton itself is cheaper because chemistry has taught the planter how to secure increased yields by proper fertilization and how to obtain increased profits by utilization of the cottonseed for oil and cattle feed. Chemistry is even now developing new sources of profit for the planter by adapting the short fiber adhering to the ginned cottonseed hull to the making of smokeless powder and the stalks of the cotton plant to paper-making.

The woolen industries are dependent upon chemistry

for the processes of separating the pure fiber from the grease and dirt with which it is associated in the raw wool and for the methods of working up this wool waste into oleic acid, soap, lubricating oils and potash and ammonia salts, as well as for the process of carbonizing by which the wool is separated from the burrs and other vegetable material with which it is admixed in the fleece.

Many of the most brilliant achievements of chemistry have been directly concerned with the textile industries. A little touch of chemistry to cotton yarns and fabrics in the mercerizing process gave the world what is practically a new textile fiber—cotton with the beauty and luster of silk. A history of absorbing interest replete with struggle, the capture of positions of temporary advantage, the constant shifting of the fighting line, crushing defeats and signal victories might be written of the development of the bleach and alkali industry, upon the products of which the textile manufacturer depends for the finishing of his goods. We see the pathetic figure of Le Blanc dying in the poorhouse after enriching the world which Napoleon was devastating. No less interesting in its human and scientific aspects is the long story of the coal-tar colors in which chemists take so large a measure of justifiable pride. An investment of \$750,000,000 follows Perkin's discovery of mauve.

Less notable, but nevertheless an industrial achievement of the highest order is the very modern development of artificial silk which, though made from wood pulp, far surpasses in brilliance and beauty the finest product of China and Japan. Closely related, thereto, is the artificial horsehair of which so large a proportion of women's hats are made and the still more recent artificial bristles of cellulose acetate with which you may have brushed your hair this morning.

A complex series of chemical reactions has its origin in the striking of every match, and civilization as we know it could hardly exist without the modern facilities for securing artificial light. For the extraordinary extension of these facilities during the past century the chemist has mainly been responsible. The immortal Faraday selected "The Chemistry of a Candle" as the subject matter of a classical series of lectures to audiences of children. From the rush candle and the tallow dip to the candles of stearin and paraffin is in itself a long journey, the milestones on which were set by Scheele, Chevreul, Heintz and Tilghman.

The refining of petroleum involved the solution of many difficult chemical problems. The Chicago fire is said to have been started by Mrs. O'Leary's cow which kicked over a kerosene lamp. In those days, however, it was not necessary to invoke the cow to start a conflagration with kerosene. Much of the lighting oil upon the market at that time would flash below 100° F. We owe our present safety in the use of kerosene largely to the work of Prof. Chandler.

The production of illuminating gas is wholly a

chemical process. When coal gas was first employed for lighting the Houses of Parliament the members might be seen gingerly touching the pipes to discover if they were not indeed hot from carrying such flame. That gas is now so cheap is due in large part to the development by Lowe of the chemical process for making water gas by passing steam through a bed of glowing coals and to the chemical processes for gas enrichment. By the Blaugas system illuminating gas is now produced in liquid form and distributed in steel bottles to isolated consumers like so much kerosene.

The gas mantle by which the illuminating power of gas is raised from 16 to 60 candles on a consumption of $3\frac{1}{2}$ feet an hour constitutes one of the most signal triumphs of chemical research. Certain sands found in Brazil and known as monazite sands had long been a happy hunting ground for chemists by reason of the number of rare metallic elements to be found therein. They seemed to be a sort of chemical garret where everything not otherwise used up during the process of creation had been stowed. Dr. Carl von Welsbach was investigating the rare elements in these sands some thirty years ago and studying their spectra. It occurred to him that a better flame for his purpose or rather a better distribution of the metallic vapor in the ordinary Bunsen flame might be secured by distributing the metallic compound through the substance of a bit of cambric. He dipped the cambric in a solution of the salts, suspended it in the flame, burned off the cotton, and found that the fragile ash glowed with an amazing brilliance. So came into being the gas mantle which has revolutionized and saved the illuminating gas industry, though not until the initial discovery had been followed by years of the most painstaking and refined research.

In the development of electric lighting the chemist has played a part scarcely less important than that of the electrician.

The arc light was first shown by Davy between charcoal points and was maintained by the current developed by the action of chemicals in the enormous battery of the Royal Institution. To Faraday, whose achievements in electricity have overshadowed his renown as a chemist, we owe the discoveries upon which our modern methods of generating electricity are based. The early history of the incandescent lamp is a chronicle in equal measure of the difficulties of finding a proper material for the filament and those of producing the requisite degree of vacuum in the bulb. Both problems were solved by chemistry which first supplied the carbon filament made by dissolving cellulose, squirting the solution into a thread of the required diameter, drying and carbonizing the thread and thereafter flashing in an atmosphere of hydrocarbon vapor to deposit carbon on the filament precisely where and in exactly what proportion its original inequalities of resistance to the current made necessary. More recently Whitney and other chemists working in the same field first greatly raised the

efficiency of the filament by the process of metallizing, so-called, and have since given us lamps of an altogether new order of usefulness by employing new materials, as tungsten, for the filament.

The second great problem, that of securing rapidly and cheaply the necessary high vacuum in the bulb, was solved in the most elegant manner by the extraordinary Malignani process. Malignani placed within the tubulature leading from the bulb and connecting the bulb and pump, a minute quantity of red phosphorus, started the pump and roughly exhausted to about 2 mm. mercury. He then sent through the filament a current so heavy as to bring the filament to intensive incandescence and cause the gaseous residue within the bulb to faintly glow so that the bulb was filled with a luminous blue haze. He then sealed off the pump by fusion of the walls of the tubulature below the phosphorus and with the bulb still glowing touched the tip of the blowpipe flame to that portion of the tubulature wall against which the phosphorus rested. With the vaporization of the phosphorus the blue haze instantaneously disappeared and an almost perfect vacuum was secured within the bulb. The process is not one of oxygen combustion as might on first thought appear and its ultimate mechanism was not understood until many years subsequent to its discovery.

The improvements in incandescent lamps in the last ten years have resulted in the saving of \$24,000,000 a year in the cost of lighting as compared with the cost of equal illumination by the older types of lamp.

To the art of illumination Wohler and Willson have contributed the calcium carbide and acetylene found on every automobile and in a hundred thousand isolated homes; Pintsch and Blau have developed separate systems permitting the transport of illuminating gases in steel tanks for the lighting of trains and houses; to Hewitt we owe the mercury lamp, to other inventors the flaming arc, to Nernst the high efficiency lamp which has his name, and, long before them all, to Bunsen the blue flame burner utilized by Welsbach and which constitutes the basic element in every gas stove.

I have endeavored in this cursory and most inadequate survey to indicate something of the extent to which chemistry contributes to the satisfaction of the demands and needs of every-day life. The earning power of the science becomes more directly apparent in its relation to general industry.

American manufacturing is in many respects the most intensive in the world. Nowhere is plant scrapped so quickly to be replaced by larger, faster and more efficient machines. Nowhere else is labor so speeded up by piece work, bonuses, motion studies, gang organization and the other devices of the efficiency engineer. In no country can new office systems, typewriters, adding machines, time recorders, memory ticklers, duplicating devices and all the paraphernalia of the follow-up be sold as quickly and in none are

they utilized so thoroughly. Our manufacturers understand these things, and what they understand they want, and are quick to make the most of, provided always they can use it in their business. They do not understand chemistry, naturally they do not propose to have any chemist teach them their business. This is reflected in the attitude of their subordinates which is commonly one of militant skepticism. They, like their masters, cut themselves off from that great coördinate and organized body of knowledge brought together by thousands of highly trained minds through the incessant questioning of nature during a hundred years. They pay less regard to many of the laws of nature than they do to city ordinances. When under these circumstances they fail to make a satisfactory profit in competition with more enlightened Germany, they jack up the tariff. They ignore applied chemistry which offers them better protection than the highest schedules of the Aldrich bill.

Let us consider a few concrete examples of the earning power of chemistry. A large pulp mill found itself with over 100,000 cords of peeled wood piled in its yard and this wood was beginning to rot. A few thousand gallons of sulphite liquor sprayed over the pile from a garden hose killed the fungus and saved the pile. The same mill was losing 23 per cent of its wood as barker waste. Laboratory trials proved that an excellent quality of paper could be made from this waste, all of which in this mill is now profitably worked up. Other mills still throw 20 per cent or more of their initial raw material away. The mill was cooking in 16 hours. Laboratory cooks were made in $7\frac{1}{2}$ hours and the time of the mill cook reduced to 10. Finally, by a proper spacing of the digesters, the production of the plant was brought from 97 tons a day to 149 tons.

Cylinder oils generally cost about what you are accustomed to pay. Plants which employ a chemist pay from 19-27 cents. Manufacturers who do not need a chemist commonly pay 45 cents, 65 cents or even, if they know their own business very well, \$1.50 a gallon. There is probably not a large plant in the country in which, if it is not already under chemical control, the lubrication account cannot be cut in two. In the engine room of one large cement plant the average monthly cost for lubricants had been \$337. It is now \$30. A concern paying 37 cents a pound for a special grease which the superintendent needed to run the mill now buys on specification for $5\frac{1}{2}$ and the mill still runs. Another company within our knowledge saves \$12,000 a year on cutting oils alone.

In a plant near Boston using two tons a week of special steel rolled very thin, their chemist was able in about two years to reduce the cost of the material from 80—40 cents a pound while at the same time standardizing and greatly improving the quality of the steel. We recall savings of \$2,100 a year on wrapping paper, \$3,600 on boiler compounds, \$6,800 on a minor

article of supply, \$100,000 a year on a single raw material.

Chemistry points out the only proper way to buy supplies which is on the basis of their industrial efficiency by means of specifications defining the quality desired and rigid tests to make sure that quality is secured. Independent estimates by those in exceptional positions to know place the efficiency value of supplies as purchased and used by American manufacturers at 60 per cent of what it should be.

Comparatively few American manufacturers light their cigars with \$20 bills. It is too slow a method of burning money. They prefer to burn it by shovelfuls, so they burn it in the boiler room. They forget that in ostensibly buying coal they are really buying heat and they pay good money for slate and sulphur balls with no knowledge of the actual number of British thermal units they are receiving for a dollar. Perhaps they depend upon a trade name ignoring the fact that coals from different mines in the same district vary greatly as does also coal from the same mine. Moreover coal, like some other things, is not always true to name. A few years ago the Boston School Committee decided to buy its coal on specification. It had previously bought "New River coal of the best quality" and that definition of its desires was included in the specification which, however, also included a chemical definition of what coal bearing this name should be. When deliveries were made by the same dealer who had previously supplied school coal they proved to be an inferior grade of Pennsylvania coal with sulphur in some samples running up to 6 per cent. When the contractor was called to account, he admitted that he did not know the state in which New River coal originates nor the transportation route by which only it could come to Boston. His comment to the committee was, "I don't see what you are fussing about, it's the same coal you've always had." Later when the temperature in the piles in a certain school ran up 90° in one day he was called upon to remove all coal delivered by him to schools in that district and substitute therefor New River coal, which he did at heavy loss to himself and corresponding gain to the city.

Important as are the losses in the initial purchase of coal, they are small compared with those which attend its burning. Many a mill owner looks out of the window and sees, without knowing, his dividends go up the chimney. Under well regulated conditions of combustion the flue gases should contain not less than 12 per cent of carbonic acid gas. They frequently contain no more than 3 per cent. This means that for every ton of coal burned under the latter conditions more than 52 tons of excess air are heated to the high temperature of the flue gases. Chemistry meets these conditions by analyzing the flue gases and regulating the draft as indicated by the percentage of carbonic acid found. At \$2.25 a ton, which is much below the average price, the fuel bill of the United States was

over \$1,000,000,000 in 1910. Of that amount chemistry could easily have saved \$100,000,000.

Chemistry aids the manufacturer who will listen to her teachings in countless other ways. It substitutes a rigid control of processes for the guesswork and uncertainty of the rule of thumb. It increases the productivity of labor by supplying more efficient processes.

In the sulphur mines of Sicily young boys called *carusi* climb with groans and curses for four hundred feet bearing in a stifling atmosphere 40-pound loads of sulphur ore upon their backs. In Louisiana, thanks to Frash, two concentric pipes are driven to the ore, a hot solution of calcium chloride is forced through one pipe to melt the sulphur which is then pumped to the surface through the other, at a trivial fraction of the cost of raising the ore in Sicily.

In the old days of making paper the rags were piled in a heap, moistened and allowed to stand for weeks until fermentation had proceeded far enough to soften them. Now they are boiled with lime for a few hours. They used to be bleached by the slow action of the sun and dew as they were spread upon the grass. They are brought to better color now overnight by bleaching-powder. Cutting tools made from high-speed steels multiply the output of the lathe and planer. The addition of 1 per cent of calcium chloride to the electrolyzing bath doubles the yield of potassium chlorate.

Chemistry aids the manufacturer by standardizing his product and reducing seconds and rejections. It costs just as much to tan goat skins into seconds as into firsts though seconds bring a third as much. Chemistry even comes to the front bearing ammunition during an advertising campaign. You may remember the offer of a blowpipe and a bit of charcoal coupled with the information that if your paint was a lead paint as the advertiser believed it should be you could quickly prove its quality in the laboratory of your kitchen by reducing from the paint a little pellet of metallic lead. You do not see that advertisement now. It disappeared about the time that some one else informed the world that zinc paints are "unalterable even under the blowpipe."

Nowhere, however, does chemistry render such efficient service to the manufacturer as in turning to profit waste and nuisance. To this phase of its service we shall return again.

To quote Prof. Duncan:

"During the next five years the small manufacturer who is swept out of existence will often wonder why. He will ascribe it to the economy of large scale operations, or business intrigues or what not, never knowing that his disaster was due to the application of pure science that the trust organizations and large manufacturers are already beginning to appreciate."

A few of us have been surprised, and none more than the railway managers themselves, by the well supported statement before the Interstate Commerce

Commission that the railroads of the country could save \$300,000,000 a year by the application of scientific management to the operation of their properties. Every chemist who has studied the problem is well aware that the entire amount in question could be saved through utilization of the proved results of chemistry alone.

Abraham S. Hewitt is authority for the statement that the Bessemer process has added \$2,000,000,000 yearly to the world's wealth. By far the greatest portion of this increment has come through the economies which this process of steel-making has rendered possible in transportation.

Our own study of car painting practice on 21 electric roads has developed the fact that 50 per cent of the cost of materials and labor is wasted and more than 50 per cent of the time spent by the cars in the shops is unnecessary.

The classic work of Dr. Dudley as the head of the laboratories of the Pennsylvania system has gone far to standardize railroad practice throughout the country. Few even among railroad men realize how greatly the whole community is in his debt. His specifications cover rails, soaps, disinfectants, oils for signals and for lubricating, paints, steel in special forms for every use, car wheels, cement, signal cord and every detail of equipment. He has made the transportation of life and property cheaper, safer and more expeditious by reason of his application of chemistry to the problems of railroad management.

In a recent address Dr. Frankforter, voicing the opinion of every thoughtful chemist, said: "The United States is the most wasteful nation in the world; wasteful in living, wasteful in manufacturing, and wasteful in conserving its natural resources." So heedless and appalling is this waste that the mind trained in chemistry stands aghast. I have lately visited a southern lumber mill which burns 1,900 cords of wood a day in its incinerator. There are two hundred such burners in the country limited in destructiveness only by the amount of material sent to them. From such wood chemistry is prepared to extract three gallons of turpentine a cord, 10 gallons of ethyl alcohol, or paper pulp to the value of \$20. We waste each year 500,000,000 tons of coal and each day a billion feet of natural gas. With peat deposits fringing our entire eastern coast we pay \$4 a ton for coal delivered on the bog. Beehive coke ovens flame for miles in Pennsylvania and excite no comment while the burning of a \$1,000 house would draw a mob. We fill the Merrimac River with wool grease, making it a stench, while the towns along its course buy soap and fertilizer and lubricants from Chicago, Chili and Pennsylvania. We burn coal-tar in Massachusetts and import coal-tar colors at high prices from Germany. Over the great northwest we burn each year 5,000,000 tons of flax straw while we pay \$40 a ton for imported paper stock from Norway. In the South 300,000 tons of paper fiber of the highest grade are burned

with the cottonseed hulls to which it is attached or used with them to adulterate cattle feed. Corn-stalks, to an incalculable tonnage rot or are burned each year while chemistry stands ready to convert them into feed containing 30 per cent of sugars on the dry basis, or into alcohol for light and power. Waste molasses is sold for three cents a gallon or dumped into the stream while alcohol sells for 40 cents a gallon. Skim milk is fed to hogs or thrown away because no one has the enterprise to extract its casein which is worth more than beefsteak for food.

In the face of such conditions we still meet young men who would inform us that the day of opportunity is past. The truth is that opportunity is knocking not once but insistently and long at every entrance to the chemist's laboratory.

Nowhere is the earning power of chemistry better shown than in its ability to transform cheap raw materials into products of exceptional value. A cord of wood is worth perhaps \$10 with a dry weight of a little over a ton. Its value, therefore, is about a half a cent a pound. In the form of chemical fiber for paper-making half the weight is lost but the remainder is worth $2\frac{1}{4}$ cents a pound. As paper it finds a market at 4 cents. Made into artificial silk by more refined chemical processes it commands \$2.00 a pound, while as cellulose acetate bristles it is worth \$4.00.

Many of our great industries are founded on minute chemical facts. Goodyear drops a bit of gum mixed with sulphur on a hot stove and the rubber industry results. The fact that silver salts happen to blacken when exposed to light is responsible for a corporation with \$35,000,000 capital on which the earnings are over 20 per cent a year. The dipping of cotton yarn in caustic soda while tightly stretched has revolutionized the manufacture of the better grades of cotton textiles. Because the chemist learns that glycerine treated with nitric acid becomes explosive our army engineers are able to separate two continents. Becquerel, having placed a bit of uranium upon a photographic plate in a black paper wrapper, finds on development that the plate has blurred. The observation leads Prof. and Madame Curie to study similar actions by uranium ores and presently the thought of the world is enriched by altogether new conceptions of the constitution of matter, and our minds are awed by the magnitude of forces previously unrecognized.

Two classes of securities find a ready sale in Massachusetts— $3\frac{1}{2}$ per cent bonds and gold bricks. It is not an easy matter to raise money for a sound chemical proposition which promises 20 per cent. Much the same conditions undoubtedly prevail throughout the country. Boston, which invested largely in sea water gold, the Hickman machine for converting starch to cane sugar, and the electrical process by which spruce wood was transformed into Australian wool with the grease in and the burrs attached, is just now figuring its losses on synthetic rubber. It left to other communities the formula of the Altoona cob-

bler for burning ashes, the process for converting water into kerosene, and the Lamoine diamonds. Men who turn a box of strawberries upside down and require a pastor's certificate of character from the office boy, rush into misapplied chemistry with never a thought of expert investigation or advice. The pity is the greater when one realizes, as every chemist does, the generous scale by which are measured the rewards of chemistry properly applied and wisely administered. Ten years ago a Massachusetts company with a capital of \$20,000 was organized to conduct a manufacture based on chemistry; two years ago it charged off \$700,000 on real estate and equipment; to-day it has a surplus of over \$1,000,000. The great Badische Anilin und Soda Fabrik, the Elberfeld Co., Brunner, Mond & Co., the E. I. duPont de Nemours Powder Co., Meister, Lucius & Bruning, the Solvay Process Co., and many others well known to every chemist are among the most profitable industrial organizations in the world. The one thing lacking for an enormous development in this country of equally profitable enterprises based on chemistry is a reasonable appreciation by our business men of the earning power of chemistry.

The ordinary investor who may safely trust his own judgment in matters involving cotton, wheat, mortgages, railroad shares or telephones is not equipped by training or experience to decide upon the validity of propositions involving chemistry. He must, if he would avoid disaster, rely upon the opinion of disinterested experts. Such opinion should cover the soundness of the chemistry involved, the state of the art relating to the manufacture, the patent situation, the available market, the nature and extent of competition, the supply of raw material, the stage of development of the process, the cost of plant and the costs of production. These last should be itemized and the basis for conclusions regarding every item should be fully stated. Large allowances should invariably be made for depreciation and in most cases equally liberal allowances for contingencies. Secret processes should be left to the fool and his money.

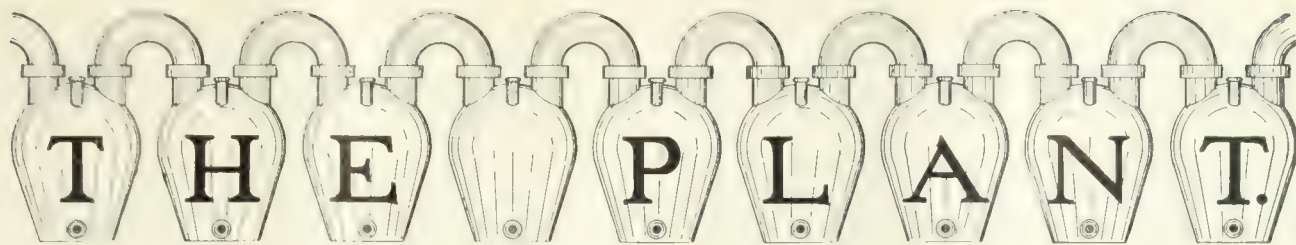
There are in the country at least 100,000 doctors and nearly 125,000 lawyers. There are only 10,000 chemists to carry on a work incomparably more important than litigation and no less beneficial than medicine to the life of the community if that life is to be worth living. Some measure of the mere material benefits which chemistry can offer may be found in the fact that the annual production of the chemical industries of the United States is already nearly equal in value to our agricultural products. Let us, however, not forget that these benefits have come, as many more will follow, because chemists have never faltered in pursuing truth for years through the labyrinth of difficult researches with no better guide than the slender and often broken thread of an hypothesis. Turgot has said: "What I admire in Christopher Columbus is not that he discovered the new world but that he went to look for it on the faith of an idea."

The fifth, sixth, and seventh public sales in Batavia for 1911 of Billiton tin, in slabs of 74.8 lbs. each, resulted as follows: May 3, 5,500 slabs at 40.1 cts. per pound; June 7, 5,500 slabs at 40.94 cts.; and July 5, 5,500 slabs at 40.233 cents. During 1910 there were shipped from the colony 16,370 tons Government Banca tin and 2,491 tons private Billiton tin.

According to the preliminary report on the mineral production of Canada for 1910, the revised figures of production for 1909 show that in 1909 there were 179 short tons of barytes produced in Canada, valued at \$1,120. No statistics are given as to the production of barytes in 1910, except a reference to 5 hundredweight having been exported, valued at \$150.

According to the statistics of the Priamur mining inspector, 17 gold mines were being operated in the Amgun mining district of Siberia. These mines include 40 claims and produced 27,595 ounces troy of gold. There were 831 Russians and 4,146 Chinese laborers employed, and in all about 500,000 tons of gravel were washed. The total sum paid for labor amounted to \$186,856. In addition to the above, 150 Russians and 2,566 Chinese worked on these claims on a percentage basis and from 12 mines produced 21,312 ounces troy of gold, the total value of their share being \$178,503. These figures do not include gold illegally obtained by Chinese and exported to China.

The field force of the United States Geological Survey is prosecuting searches for deposits of minerals which shall furnish the three necessary elements of plant food that are contained in "complete fertilizer," namely, phosphate rock, nitrate salts, and potash salts. The Survey has already discovered and surveyed enormous deposits of phosphate rock, and 2,398,590 acres of public land containing phosphate, withdrawn at the recommendation of the Geological Survey, are now waiting legislation by Congress to enable their development. In order, however, to insure an "all-American" fertilizer, regardless of importations from other countries, it remains to discover deposits of nitrate of soda and potash salts. As both of these minerals are readily soluble and are not to be found as "outcrops" like ordinary rocks, the mission of the Survey is not an easy one. Nevertheless, it is believed that the geologic conditions prevailing throughout a large portion of the arid West favored the accumulation, during earlier periods of the earth's history, of both of these salts, and that if these still exist in concentrated deposits it is only a question of search to discover them. Hardly anything could be suggested which would be of greater assistance to the American farmer than the discovery of commercial deposits of either of these necessary fertilizing minerals.



TWO PROBLEMS OF FACTORY BUILDING.*

Every day new manufacturing enterprises are being incorporated. A large proportion of these are inaugurated by men who are for the first time "going into business for themselves." Immediately the new company is likely to be confronted with at least two questions of prime importance: 1. What kind of factory buildings? 2. Where shall the plant be located? Experience indicates that these two problems are handled unwisely with a frequency beyond apparent excuse. Moreover, the complaints one hears on every hand disclose that "going" concerns that have thrived and prospered, grievously err at some period of expansion in the solution of the same two problems. The mistakes of the wealthy corporation are not of so much concern, but most of the others begin business with too little rather than enough money, so that what may be saved to them is saved to good purpose.

The first mistake usually arises from a failure to realize that a new business should be proved to be a "good" business before one dollar is permanently invested in buildings. This may be termed a principle rather than a hard and fast rule, for in some instances buildings already erected and perhaps suitably equipped can be obtained at bargain prices. At the same time, it may be stated without any exception that a factory building project should not be entered into on an ownership basis before the merits of a projected business as a money-making proposition have been thoroughly demonstrated. Much too frequently the proposed factory looms up as the biggest thing in the new company's horizon, when it should be only incidental to the great problem of building up a business. Where a new and untried business may easily cripple its prospects seriously, through too great an investment in plant, leaving too little of its original capital for working purposes, the "going" business has surprisingly greater resiliency under stress, and, moreover, offers an infinitely better basis for financing and for planning the new plant. Therefore, beginners should establish their business temporarily in any quarters that will do, and then wait until the business proves that a plant is necessary before building.

Deciding upon a location is scarcely less important. While many concerns succeed in spite of their location, a mistaken choice is a source of continual annoyance and ordinarily is possible of change only at great cost.

The new company is the prey—and, strange to say, is often flattered thereby—of the enterprising "improvement associations" of a dozen towns, to say nothing of the now numerous "land associations." It is indeed unfortunate that so many too energetic folks in suburbs and villages are inspired to make manufacturing towns of these hamlets, wholly regardless of their qualifications therefor. The tendency to boom towns that never should be anything but farming or residence communities, by enticing unwary enterprises to locate, has resulted in no end of harm to those industries and no ultimate good to the community. There are now for sale hundreds of abandoned plants all over the country that stand as a striking condemnation of such incongruous and ill-advised endeavor.

Before considering the question of a building site a company should have available sufficient resources to acquire ownership of the land without fear or favor. So situated, the company is not likely to be influenced to forego really necessary facilities in order to take advantage of the tempting concessions offered. More often than otherwise the new company starts out with too little money. It follows that the cash bonuses, free sites and building material, easy taxes, etc.—the common bait held out—come to be reckoned upon as a part of the company's available capital and as necessary to the inception of the enterprise. To so consider these inducements is unqualifiedly wrong. Thus, the most liberal offer becomes the most attractive, while considerations having to do with operating conditions, in the particular locality, are made secondary or entirely submerged. It may be accepted as almost axiomatic that the most liberal offer will emanate from the least desirable location. It cannot be too strongly emphasized that no such inducements, however liberal they may be, are sufficient to justify the permanent handicapping of operating economy. The merits of every location should first be considered wholly apart from the terms of acquisition, and the unsatisfactory site should be rejected regardless of the gifts which are handed out with it. This may seem an extreme ruling, but if a business is worth the building of a plant it may be expected to continue for many years, and the earnings of the business will dwarf original considerations into insignificance. On the other hand, the disadvantages of location that at first may seem trifling will roll up into mountains of annoyance as they are accentuated by the growth of the business and the passing of time.

*From The Iron Age.

THE FLOW OF HEAT THROUGH FURNACE WALLS.*

By **WALTER T. RAY AND HENRY KREISINGER.**

The data upon the flow of heat through furnace walls was obtained in the course of an extended experimental study of combustion carried out by the United States Geological Survey at its fuel-testing plant in Pittsburgh.

These incidental experiments have yielded results bearing upon the quantity of heat dissipated through the walls of furnaces, and, particularly, have furnished a conclusive answer to the question as to the relative efficiency of walls of the air-space type, and walls of solid brick or other material.

It is a very common belief that, since air is a very poor conductor of heat, an air space between inner and outer walls must be an efficient heat-insulator. This reasoning does not take into account the fact that radiation over an air space is of even more importance under the conditions than conduction, and that radiation over an air space takes place very readily.

The experimental furnace used in these tests is about 43 feet long. It is fed by a mechanical stoker at one end, and the combustion takes place in a tunnel 3 ft. wide by 3½ ft. high and 38 ft. long, the hot gases discharging into a water tube boiler. The tunnel has double walls and double arch. The inner walls are built of best fire brick, 9 ins. thick, and the outer of common brick 8 ins. thick. A 2-in. air space separates these walls. The inner arch is of 9-in. fire brick, and the outer of repressed brick 4 ins. thick. Between the two arches is a 1-in. layer of flake asbestos. The interior of the chamber is kept as nearly as practicable at atmospheric pressure to minimize air leakage through the walls.

Three sets of thermocouples of four units each were placed in the walls of the furnace at distances representing about ¼, ½ and ¾ the length of the furnace from the grate end. Two similar sets were placed in the roof at the ½ and ¾ distances.

The arrangement of the units of each set for getting the heat gradient through the walls or roof is shown in Fig. 1. The junction of each couple was in contact with the bottom of its hole, and the holes were packed with asbestos after the couples were in place.

The thermocouples used were obtained from the Hoskins Mfg. Co., of Detroit. The connections were such that each couple could be thrown into series with the indicating instrument by a snap-switch, so that a complete series of readings could be taken almost simultaneously. "The couples proved durable and dependable up to 1,400° C. (2,522° F.) —."

No attempt was made to get the highest accuracy in reading temperatures, for relative results were sought rather than absolute. Also the conditions in the bottoms of the holes were not subject to inspection,

and many chances for small errors were always present on that account. The complete results of one representative test are given. Starting with a cold furnace, the test was run through 29 hours. Curves were plotted for each couple, temperature readings against time as abscissas, the shape of the curve showing the rate of increase in temperature. Each set of couples showed entirely similar curves, the rate of change being, of course, faster for the couples nearer the inner surface of the furnace. After about 19 hours, the temperatures of all couples ceased to increase, the curves at this point becoming horizontal lines. This conditions means that, whereas during the first 19 hours of the run most of the heat entering the walls went to raise the temperature of the walls, after the equilibrium was attained the heat flowed constantly, and in equal quantity in all parts, through the walls and was dissipated into the air by radiation.

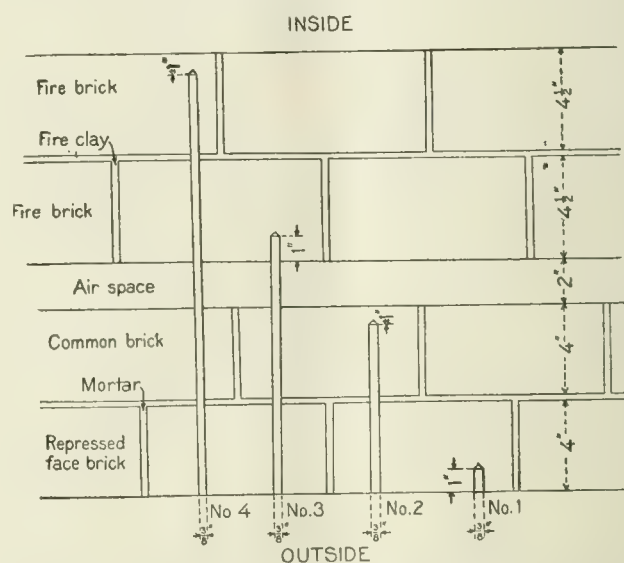


Fig. 1—Construction of Furnace Walls and Arrangements of Thermo Couples.

The next series of observations relates to the temperature drops in various zones of the furnace walls.

The quantity of heat flowing through a wall from one plane to a parallel plane depends upon two factors: the differences between the temperatures of the two planes and the resistance to the flow of heat. Given a constant quantity of heat, if the resistance to its flow is great, the temperature difference between the planes will be great; if the resistance is small, the difference will be small.

This being true, in the case of a furnace wall at equilibrium (when the quantity of heat flowing through all planes parallel to the wall surfaces is constant) the difference in temperature between any two planes is a function of the resistance of the medium between those plans. Thus if the difference in temperature between the two couples which have fire brick between them is greater than the difference in temperature between those which have an air space included in the space between them, the resistance to the flow of heat must be greater in the fire brick than

*From Bulletin No. 8 of the United States Bureau of Mines, Dept. of the Interior.

in the air space. It is thus possible to depend upon temperature difference as being an indicator of the high or low resistance of materials to heat flow, and to make roughly quantitative comparison of such materials.

Series of observations of temperatures were taken from all sets of couples in walls and roof of the furnace after equilibrium was established, and curves

represent the gradients before equilibrium, and are not significant in showing comparative resistances, but numbers 3 were taken during equilibrium and give the information sought very definitely.

In the lower curves, representing the drops through the furnace walls, it is shown in a striking manner that whereas the drop through the fire brick is very great, and through the common brick, almost equally so, the drop through the air space is very small, showing that the air space offers much less resistance to heat flow than either kind of brick. On the other hand, the number 3 curve representing conditions in the roof with its 1-in. asbestos layer shows just as strikingly that the resistance to heat flow is much greater on the part of the asbestos than that offered by either kind of brick. These curves show convincingly that 1 in. of asbestos is far more efficient as a heat insulator than a 2-in. air space.

Upon the entirely correct assumption that the conductivities of similar brick work in the walls and roof are the same, and as the thicknesses are the same, then the quantities of heat flowing through walls and roof per unit of surface in a given time are proportional to the temperature differences or drops. Calculating on this basis results show that about twice as much heat is lost per unit of area, through the side walls as through the roof.

The results of the investigation as outlined justify the following conclusion:

"In furnace construction a solid wall is a better heat insulator than a wall of the same total thickness containing an air space. This statement is particularly true if the air space is close to the furnace side of the wall, and if the furnace is operated at high temperatures. If it is desirable in furnace construction to build the walls in two parts so as to prevent cracks being formed by the expansion of the brick work on the furnace side of the walls it is preferable to fill the space between the two walls with some "solid" (not firm, but loose) insulating material. Any such easily obtainable materials as ash, crushed brick, or sand offer higher resistance to heat flow than an air space. Furthermore, any such loose material by its plasticity reduces air leakage, which is an important factor deserving consideration."

The explanation of the results above presented is found in well demonstrated physical laws, i. e., the laws of radiation and the laws of conduction. Although the laws of radiation are well established, they are of comparatively recent date and are not generally known, in spite of the commonness of radiation phenomena. The quantity of heat conducted through a portion of a solid wall depends upon the difference between the temperatures of the planes limiting the portion of the wall.

On the other hand, the quantity of heat radiated through an air space depends upon the difference between the fourth powers of the absolute temperatures of the surface enclosing the air space.

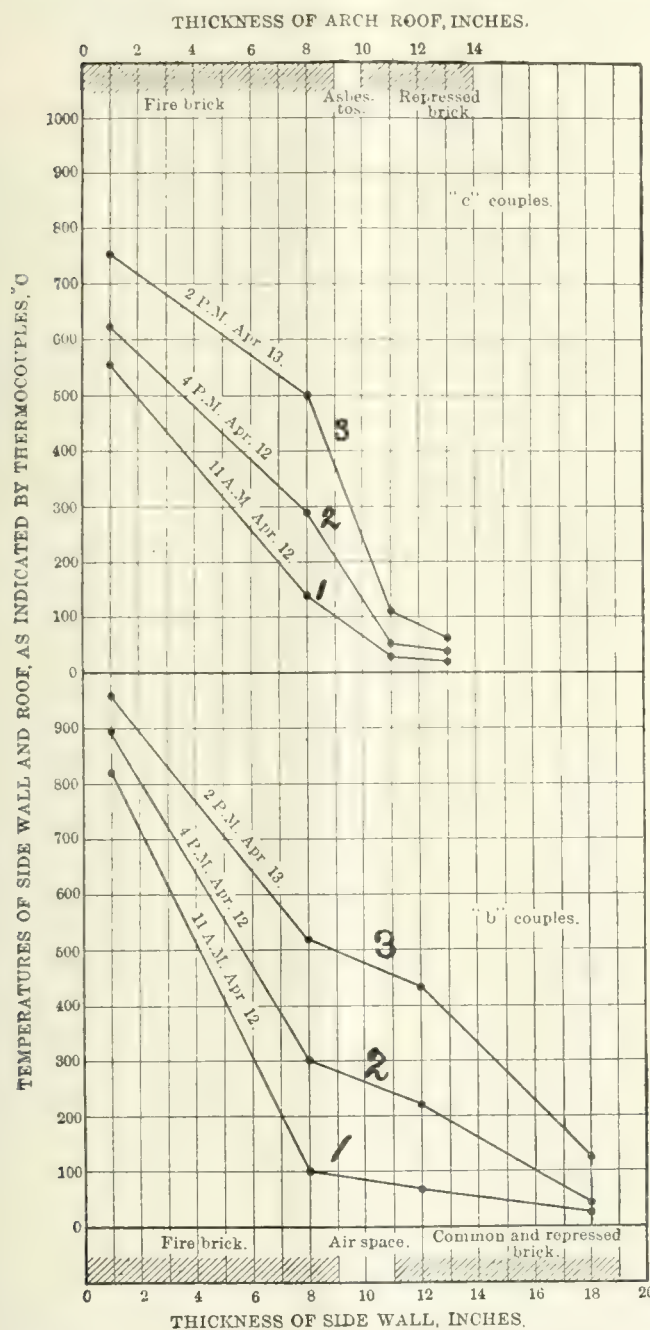


Fig. 2—Curves Showing Temperature Drops Through Walls.

were plotted with temperatures as ordinates against thickness of wall as abscissas.

Figure 2 is typical of the observations made at the various sets of couples. In it is shown the temperature gradients or drops through the walls, with their air space, below, and the roof with its layer of asbestos, above. Number 1 and 2 curves in both cases rep-

In the light of these laws, it is seen that no matter how high the temperature, so long as the temperatures between surfaces remain constant, the loss of heat through a solid wall will be constant. On the other hand, the quantity of heat radiated through an air space, being a function of a difference in fourth powers, increases rapidly with increased temperatures even though the difference between these temperatures be constant.

This also throws light on the fact that an air space is a splendid insulator for refrigerators, for there the temperatures are low, though it is objectionable where high temperatures are in order.

Upon the basis of the laws mentioned, calculations are made which check fairly well with the calculations made from experimental data, and further calculations of the total heat lost per unit of time through the walls of the furnace are presented.

The authors of the bulletin here reviewed give special thanks to "John K. Clement, Physicist, and his assistant, L. H. Adams, both men of the Bureau of Mines, for their aid in devising methods for taking many of the observations, and assisting the authors in studying the data."

COAL COMBUSTION RECORDERS.*

By AUGUSTUS H. GILL.†

By the simple determination of the carbonic acid content and temperature of the gases from a boiler furnace, its efficiency can be closely determined. By the investment of \$10 in premiums to keep the CO₂ at 12 per cent, an electric company in a Massachusetts city saved 40 tons of coal. This would seem to demand the attention of every manufacturer using coal.

The first analyses of chimney gases were made in 1827 by Peclet; they were sampled by emptying a bottle of water in the gases and he found that in ordinary combustion only about half of the air was used. Bunsen in 1839 analyzed the gases from a blast furnace in Veckershagen and found that over 40 per cent of the fuel was wasted; attempts were made to use this waste heat for raising steam. The analyses in those days were tedious and troublesome, a whole day being employed in taking a sample. Scheurer Kestner used 10 gals. of mercury and complicated trains of combustion and absorption apparatus. Now a result is obtained in five minutes and recorded—all automatically.

The first portable apparatus for gas analysis was devised in 1872 by Clemens Winkler, the discoverer of the rare element Germanium. This apparatus, however, is troublesome to manipulate, is not jacketed and hence is ill adapted to the drafty boiler room. It however served a useful purpose in preparing the way for a better. This was the Orsat, patented by M. H. Orsat of Paris in 1873, and notwithstanding its price of 15/,

in October, 1874, found its way into more than 50 factories in all parts of Europe and even in America. It was used wherever carbon dioxide was generated—with all sorts of furnaces—puddling, melting, Bessemer, Siemens-Martin, boilers, gas producers, fermentation industries, beet sugar factories, sulphuric acid and alkali works.

The chimney gas is collected and measured over water or brine in a burette and successively forced into various absorbents contained in pipettes, and the diminution in volume represents the percentage of the different constituents. This is done by hand and requires nearly a half hour for its completion. Three gases are determined, one of them carbonic oxide, indicative of imperfect combustion which is shown by no other apparatus. It, however, has the disadvantage that it tells what is transpiring in the furnace only at the time at which the sample was taken—a very brief interval two or three minutes at most. Furthermore it requires an attendant to operate it. Its indications were so valuable as to create the desire for an automatic device which should indicate the condition of the boiler furnace the same as the steam gage tells the pressure. The first of these automatic devices was patented in England in July, 1892, by Custodis and Duerr of Munich. This consisted of a balanced glass globe suspended in a case into which the gas to be tested was sucked—a modification of the Lux gas balance. This invention was quickly followed by a variation by Arndt in September, 1892. Custodis and Duerr, in 1895, then improved their first balance by adding another globe and making it recording.

This balance type of apparatus had the disadvantage that it was constantly in motion, with the result that the knife edged and planes wore, rendering it less sensitive, requiring frequent repair and adjustment. This type was superseded by the absorption apparatus of Arndt in Aachen, who received a patent in England in December, 1896, for his "Ados" or heating effect meter. Other patentees are Cederborg of Denver, Simmance Abady of London, J. C. Eckhardt of Stuttgart, Sarco Fuel Saving & Eng. Co. of New York, Uhling & Steinhart of Newark, Salan & Birkholz of Essen, G. Van Gildern & Co. of Dusseldorf, and Hugh K. Moore, Berling, N. H. With these automatic devices usually a small stream of water furnishes the power to draw a current of gas from the uptake or chimney into a floating gasometer and force it through potassium hydrate into another gasometer which is connected with a recording device which shows the amount of carbonic acid absorbed.

This should be about 13 per cent, as if a greater percentage be obtained, the loss by the formation of carbonic oxide usually more than compensates for the increase due to carbon dioxide. A greater percentage than thirteen means that the fires are too thick. A less percentage than 10 means either that the fires are too thin or there is leakage either through the bricks themselves which form the setting or through cracks

*Paper presented before the Congress of Technology at the 50th Anniversary of the Granting of the Charter of the Massachusetts Institute of Technology.

†Professor of Technical Analysis, Massachusetts Institute of Technology, Boston.

therein. In many cases this is sufficient to reduce the CO_2 percentage to 5 or 7, which means a total loss of 36 to 26 per cent of the fuel, 22 to 12 percent more than should be lost, or otherwise expressed of nearly 5 to 2 tons of coal every twenty.

This table shows the percentage excess of air and percentage loss heat with the gases escaping at 518°F . for various percentages of CO_2 in the gases:

Per cent CO_2	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
Excess air...	850	530	370	280	220	170	140	110	90	70	60	50	40	30	20
Loss of heat...	90	60	45	36	30	26	23	20	18	16	15	14	13	12	10

Nor are these apparatus solely applicable to the regulation of combustion; wherever an absorbable gas as sulphurous or hydrochloric acids or chlorine is evolved they, modified to suit the circumstances, can be installed. This will permit of increased control of chemical operations and consequently increased economy.

THE POLLUTION OF STREAMS BY MANUFACTURING WASTES.*

William S. Johnson, sanitary and hydraulic engineer of Boston, discussed the growing tendency on the part of the public to resist all pollution of streams, and the measures open to the manufacturers in dealing with his part of the problem. Mr. Johnson said in part:

In the case of streams the public is beginning to demand something more than protection to health. It requires that the stream shall become, as they will if kept clean, the most attractive features in the neighborhood. It is at the same time becoming better understood that it is possible to have clean streams, even in a region devoted to manufacturing, and that the expense of securing this condition is not such as to drive the manufacturer from the district. In fact, this movement for clean streams is so widespread that the manufacturer would have difficulty in finding a location where he would feel certain that this demand will not arise either now or in the near future.

It is certain that this public demand, enforced as it has already been by the legislatures and the courts, and the demand of the manufacturers themselves who require clean water in the manufacturing processes, will result in the necessity of keeping out of many of our streams the most objectionable of the manufacturing wastes, and this has become one of the serious problems to be met by the manufacturer.

The simplest solution of this problem, so far as the manufacturer is concerned, is to make where possible a connection with the public sewers, transferring the problem of the disposition of the wastes to the public authorities, but this method of disposal is, in many cases, impossible, and its feasibility in other cases questionable.

In this country the tendency is rather to prevent the discharge of any considerable quantity of untreated

manufacturing wastes into the sewers and to oblige the manufacturer to construct independent works, or at least to treat the wastes so that they will not cause trouble in the sewers or at the disposal works.

The quantity of liquid wastes produced at a single factory is frequently greater than the total quantity of domestic sewage flowing in the sewers of the town in which the factory is located, and the character of the wastes may be such that they cannot be purified in connection with the sewage, except at a very great increase in the cost. The cost of the sewers and of sewage disposal is largely assessed upon those benefiting by the sewers and in proportion to the benefits received. If a proper portion of the cost should be assessed on the manufacturers, the assessment would be in many cases enormous and enough to make such disposal practically prohibitive.

Some wastes can undoubtedly be discharged into the sewers without causing trouble, depending on the character of the wastes and their volume as compared with the volume of domestic sewage flowing in the sewers. Other wastes can be discharged into the sewers after some simple preliminary treatment without causing trouble, but in perhaps the majority of cases, where wastes cause trouble in the stream, they are likely to cause trouble in the sewerage system, especially if the sewage is purified.

Manufacturing wastes vary very greatly in their composition, and generally have quite different characteristics from domestic sewage. In some cases the wastes are very much more readily disposed of by dilution than is domestic sewage, while in other cases the effect of dilution is very much less. No fixed rule can be applied to the amount of manufacturing wastes of any given kind which can be discharged into a stream of given volume, for the seriousness of the pollution depends on the use to which the water is put even more in the case of manufacturing wastes than in the case of domestic sewage.

Corrosion of rails exposed to tunnel gases has been found to reduce the life materially in the case of the long tunnel at Sand Patch, Pa., on the Baltimore & Ohio R. R., according to the Engineering Record. This tunnel is 4,775 ft. long and is operated with double-track traffic on a single-track line. The interior condition is one of practically continual dampness, with locomotive gases present. Plain Bessemer rails have a life of about 18 months in the tunnel. Deterioration proceeds in the flaking of a scale from the rail until the edges of the base become quite sharp and removal is necessary. Chrome alloy rails, substituted for the Bessemer rail at the last renewal, have now been in service nearly three years. In addition to the increased length of life resulting from the ability to resist corrosion, the breakages in this latter class of rail have been less than one-fourth of the number for the plain rail, according to M. J. Corrigan, general inspector of tunnels of the Baltimore & Ohio.

*From a paper presented before the Congress of Technology at the 50th Anniversary of the Granting of the Charter of the Massachusetts Institute of Technology.

METALLURGY.

PRODUCER GAS IN STEEL WORKS: A SCOTTISH VIEW.

The following abstract of a paper by Dr. C. Bone, F. R. S., read at the recent meeting of the West of Scotland Iron and Steel Institute is taken from *The Industrial World*. In his paper Dr. Bone advanced as his main principle that gas producer plant for a steel works should be so designed and operated as to gasify the fuel as rapidly as possible, consistent with a high thermal efficiency and the production of gas of a quality best suited to the steel furnace. Moreover, the plant should be capable of maintaining a uniform quality of gas under considerable variation of load, and of quickly responding to sudden increase in the demand for gas.

It is the prevalent and the well-founded opinion of steel-works managers that the most suitable gas for a melting furnace is one containing not more than about 12 per cent of hydrogen, together with the highest possible percentage of carbon monoxide. The reasons why carbon monoxide is more efficient, as a furnace gas than hydrogen, which, volume for volume, has nearly the same calorific value, are still somewhat obscure and the subject of contention.

Dr. Bone next dealt at length with the chemistry of gasification in a producer, quoting among others Karl Wendt's experiments with an air-blown producer, and also with an air-steam producer.

The influence of successive additions of steam to the blast of a producer was, adds Dr. Bone, the subject of prolonged investigation by Dr. Wheeler and myself some few years ago, the results of which were reported to the Iron and Steel Institute in 1908 and 1909. The plant placed at our disposal for these experiments was of a modified Mond type with arrangements for "superheating" the blast and saturating it with steam at any desired temperature up to 80 degrees C. We thus had command of a wide range of conditions, and were able to establish a good many points which, it is hoped, will be of guidance to persons controlling producer plants. The fuel used throughout the experiments was a Lancashire "wash-nut" coal, containing on the average: carbon = 78.0, hydrogen = 5.4, sulphur = 1.0, nitrogen = 1.4, oxygen = 10.0, ash = 4.2 per cent volatile matter (at 900 degrees C.) = 36.0 per cent.

The principal conclusions established by this investigation may be summarized as follows:

1—That with ordinary bituminous producer coal, such as is usually employed in the North of England, some 92.25 per cent of the total carbon in the fuel is converted into permanent gas, i. e., into oxides of carbon and methane. Another 6.5 per cent leaves the

producer either as dense, tarry hydrocarbons or as soot, and the remaining 1.25 per cent is lost in the ashes. These proportions are independent of the steam saturation temperature of the blast.

2—That with steam saturation temperatures of blast up to and including 55 degrees C, the whole of the steam introduced may be decomposed in passing through the incandescent fuel bed, the depth of which need not exceed 3 ft. 6 ins. above the grate or blast dome. As the blast saturation temperature is gradually raised beyond the limit of 55 degrees C., a decreasing proportion of the steam introduced is decomposed, although the absolute amount decomposed may increase. This is shown in the following table:

Steam saturation temp of blast Deg C.	Lbs. of steam in blast per lbs. coal gasified.	Pct. steam decomposed, approx.	Lbs. of steam decomposed.
45	0.20		0.20
50	0.21	100	0.21
55	0.32		0.32
60	0.45	about 80	0.36
65	0.55		0.44
70	0.80	60	0.48
75	1.10	50	0.55
80	1.55	40	0.62

3—That the best "furnace gas is obtained with a blast steam saturation temperature of 50 degrees C. Lowering the steam saturation temperature below this limit has a detrimental influence upon the general working conditions, without any compensating improvement in the quality of the gas, while on raising the saturation temperature beyond 50 degrees, the quality of the gas, from the furnace standpoint, at first gradually, and afterwards rapidly, deteriorates owing to the increasing production of hydrogen at the expense of the more effective carbon monoxide. Moreover, what is of great importance, the proportions of carbon dioxide, carbon monoxide, hydrogen and steam in the gas generated with a steam saturation temperature of 50 degrees C. will correspond to those in equilibrium at 1,200 degrees C., provided that the dew-point of the gas leaving the producer is about 20 degrees C., which will be the case unless the fuel charged into the producer is very wet. Hence the gas as generated under such conditions will not alter in composition during its passage through the furnace regenerators.

4—That the thermal efficiency of the gasification does not vary very much, if indeed at all, with increasing steam saturation temperature up to about 60 degrees C., after which limit there is an appreciable and steady falling off. Hence it would seem that the conditions requisite for the generation of the best quality of gas, from the furnace point of view, are also compatible with the maximum thermal efficiency.

5—That, given a suitable construction of grate and blast arrangements, a very high rate of gasification—

exceeding 20 cwt. of dry coal per hour per producer of 10-ft. internal diameter (or, say about 300 pounds per hour per square foot of fire area)—can be continuously maintained, night and day, without detriment to the quality of the gas or the general working conditions. Moreover, it is quite possible to vary the load within very wide limits without appreciably altering quality of gas or impairing the efficiency of the process. A producer which, with a suitable quality of coal—i. e., not strongly caking or clinkering—will not permit such rapid gasification or variation in load without impairing the quality of gas, must be regarded as in some way structurally defective.

I have already stated that, given systematic and intelligent control, together with a judicious selection of the proper grade and quality of fuel, structural considerations are of secondary importance. I do not wish it to be inferred, however, that I underrate the importance of a proper design of the producer and its accessories. A good design, although in itself and apart from intelligent control no guarantee of good gasification, is assuredly a powerful aid to efficiency. May I, therefore, conclude my lecture with some brief reference to points of design which experience has shown to be advantageous to good working. If, in illustrating my remarks, I refer to certain features of value in types of producers now on the market, I wish it to be understood that I have selected these particular types for the convenience of illustration only, and not in any way as wishing to advertise them. There are other equally good designs which could be mentioned, for it is no part of my purpose to draw comparisons.

1—First of all, in designing an installation of producers, it is of prime importance to insure that the blowing and steam saturating arrangements are so disposed as to be easy of adjustment. There can be no doubt, I think, of the superiority of some form of "positive" blower over the usual type of "steam air" injector, and it is absolutely essential to good and continuous working that the mode of introducing the steam should permit of an accurate adjustment of the steam saturation temperature. Uniform composition of the gas cannot be maintained if the steam saturation temperature be not perfectly controlled. Moreover, preheating of the mixed steam-air blast before it enters the producer is advantageous, in that it accelerates the gasification, and at the same time improves the quality of the gas. Such preheating can be effected by some simple device permitting of a heat exchange between the hot gases leaving the producer and the ingoing steam air blast.

2. Having satisfactorily provided for a suitably regulated steam-air blast, the next point is to employ some form of grate which will insure its uniform distribution and rapid permeation through the lower parts of the incandescent fuel bed. Many of the "dome" grate arrangements in use are ineffective in this respect, and personally I do not favor them. The simple "Duff" grate is a better arrangement, but for

rapid gasification I know of nothing to equal the inverted and truncated conical bar grate of the Mond producer, such as was employed by Dr. Wheeler and myself during the Warrington trials in 1906, and which, in a modified and improved form, we have incorporated in a producer patented in 1908. This form of grate, with blast saturated with steam, and preheated at 250° C., enabled us to gasify an ordinary Lancashire nut coal in a producer of 10-ft. internal diameter at the rate of between 20 cwt. and 25 cwt. per hour continuously night and day over long periods of time—a performance which has not been surpassed, if, indeed, equaled, with any other form of grate.

3. From the point of view of labor saving, as well as of uniform quality of gas and general temperature conditions in the producer, some form of mechanical charging, insuring a continuous feed of the fuel and its uniform distribution over the fuel bed, is to be recommended. As an example of such a device may be cited the well-known George feed of the Morgan producer. This, if properly adjusted, is a valuable adjunct to a producer. With intermittent charging, the dumping into the producer of several hundred-weights of raw coal has a rapid and considerable cooling influence upon the upper layers of the fuel bed, which is instantly reflected in the temperature and composition of the gas evolved. If a recording pyrometer be fixed in the gas outlet of an intermittently charged producer and simultaneously records of the CO₂ content of the issuing gas be taken over a long period, the temperature of the gas will rapidly fall, and its CO₂ content rise after each fresh charge of fuel has been added, and good working conditions are only slowly re-established. Such irregularities disappear when continuous charging is adopted.

4. Within recent years there has been a tendency to depart from the former simplicity of grate construction, owing to the introduction of mechanical contrivances for the automatic removal of ash and the diminution of clinker troubles. The quantity of ashes to be withdrawn from a gas producer and the frequency of their removal depend, of course, upon the ash content of the fuel and the rate of its gasification. There is something to be said from the labor-saving point of view in favor of the continuous automatic removal of ash, but the chief advantage to be derived from the use of such contrivances probably lies in the constant movement imparted to the fuel bed, which diminishes the tendency to clinker, and facilitates the proper settling down of the charge. If the rate of gasification of the fuel be fairly constant hour by hour, it should be possible (with a continuous and automatic system) so to adjust the rate of ash removal that it corresponds exactly with its rate of formation, and so keeps the lowest layer of the actual fire at a constant height in the producer. But where the rate of gasification is subject to much variation, this advantage could only with difficulty be maintained.

Producers with revolving grates must be regarded as still in the experimental stage, although it is claimed that good results are being obtained with the Kerperley producer which has lately been introduced into this country from the continent. The chief features about this producer (which may be cited merely in illustration of the trend of recent constructional enterprise) are its lower water-cooled casing around the ash zone and its slowly revolving eccentric grate. The cold walls in the ash zone of the producer are said to prevent the sticking of fused clinker to them, and the continuously revolving grate crushes any clinkers that may form, and removes them automatically with the ash. Such an arrangement enables a producer to be worked with blast saturated with steam at a lower temperature than would be feasible in the case of an ordinary type of producer, and figures have been published respecting regular working with as low a steam saturation temperature as 35°C . But if the arguments advanced in the earlier part of my lecture are valid, there is no advantage at all, from the furnace point of view, in a steam saturation temperature lower than 45° to 50°C .

The rate of gasification of a 10-ft. Kerperley producer is apparently about 16 cwt. to 17 cwt. per hour, which although an advance on many other forms of producers, is nothing exceptional, and it has been surpassed even with fixed-grate producers. With a steam saturation temperature 35°C the composition of the gas produced is given as follows:

$\text{CO}_2 = 3.8$, $\text{CO} = 27.2$, $\text{H}_2 = 9.7$, $\text{CH}_4 = 3.1$, and $\text{N}_2 = 56.2$ per cent. Total combustibles = 40.0 per cent.

Calorific value in B. T. U.'s per cubic foot at 0° and 760 mm., gross = 159.0, net = 150.0.

It has sometimes been claimed that the employment of revolving grates and the like for the automatic removal of ash means higher rates of gasification and greater thermal efficiency than with a good type of fixed grate. It is doubtful whether any such claim can be sustained, and, judging from such figures as has come under my notice, it seems inadmissible.

The City Council of Bradford, England, is reported to be the owner of a new process of dyeing, which is expected to be of the greatest commercial value. Three years ago Bradford Technical College appointed Dr. H. H. Hodgson as lecturer on chemistry with the specific duty of carrying on investigations in applied chemistry which should keep Bradford as an industrial community abreast of modern developments. Professor Hodgson is understood to have discovered a new dye stuff, which has been provisionally protected under the patent laws, and meanwhile the City Council has been called upon to decide what shall be done with this new municipal asset.

CONTAMINATION OF LABORATORY SAMPLES BY IRON DERIVED FROM CRUSHING MACHINERY.*

By GEORGE A. JAMES.

Very few chemists realize to what extent iron is introduced into pulp by the abrasion of the faces of the crushing machinery. Even experienced mining men whom one would expect to be conversant with the fact that stamp batteries are heavy consumers of iron, frequently overlook the fact that iron is a necessary constituent of battery pulp, and when in panning down a sample they come across a heavy black substance in the concentrate, they at once jump to the conclusion that they have tungsten, tellurium, iridium, or some other rare metal present and at once call for an analysis to classify it. It might be said to be an almost daily experience in a large assay office to be called upon to make a determination of an unknown for some one who imagines he has discovered a rare compound in his ore, when in reality he has merely concentrated out some of the iron derived from his crushing machinery.

Carbon is a constituent of iron, and in crushing rock, the carbon is to a certain extent liberated and its presence is frequently made known by the formation of a scum on the surface of the water in panning. It is this carbonaceous scum that is frequently mistaken for silver sulphide or some other dark colored mineral notorious for its sliming propensities.

It is the purpose of this paper to draw attention to the fact that the amount of iron introduced into laboratory samples by fine grinding has an important bearing on the accuracy of the work, whether it be of an analytical nature or the determination of a method for treating the ore.

The extent to which abrasion takes place is determined principally by the character of the material crushed and the hardness of the grinding faces. Soft, talc-like material, as one would expect, causes very little wear on the iron, while tough, flinty quartz produces an abnormal loss. From carefully conducted experiments I have found that in reducing an ore to eighty mesh in a disk grinder, from 0.25 to 3 per cent of iron is added to the sample. In stamp mills the consumption of iron averages from 1 to 2 lbs. per ton of ore crushed. In regrinding in pans the wear amounts to from 3 to 8 lbs. per ton, clearly showing the heavy increase in the consumption of iron when attempting to produce a fine product.

The introduction of iron into a sample is frequently productive of serious results. In smelter work, where the practice is to reduce the sample to 120 mesh or finer, I have frequently seen charges which were made up according to analysis of such samples give great trouble, and no one apparently seemed to be able to assign a reason for the failure of the charge to work.

*From Transactions American Metallurgical Society as printed in The Pacific Miner, the official organ of the Society.

I recall a case where a chemist in charge of a smelter, weighed his charges out to the nearest pound, and had all of the laboratory pulps put through an excessively fine screen. Until his attention was called to the matter, he failed to understand the reason why his furnaces gave so much trouble. In smelter work, the error, it will be noticed, is twofold; it increases the apparent contents in iron, and decreases the seeming amount of silica to be fluxed.

As a rule dry ores received by ore purchasers possess the greatest value and consequently bear the highest treatment charges. It has been noticed that these ores are the most abrasive ones, a point which possesses considerable importance when it is remembered that in order to get it through a 120-mesh sieve it is necessary to salt it with 2 per cent iron. This fact is well known to smelters and when the seller insists on having the sample representing his shipment put through an extra fine screen he never meets with opposition. The profit which the smelter derives from samples innocently salted by fine grinding must be considerable in the aggregate.

In preparing large samples of commercial products, such as limestone, magnesite, clays and glass sands, for analysis, the difficulty of preventing contamination with iron, and the serious results which may follow should the iron find its way in, may be readily appreciated. The introduction of a very small fractional percentage of iron into a clay or glass-sand analysis may result in the rejection of what might be actually an excellent grade of material. In carbonic acid determinations, the effect of iron would be to generate hydrogen during the acid treatment, which would of course lead to erroneous conclusions.

In conclusion I would like to say that the introduction of iron into laboratory samples by the abrasion of grinding faces is a matter deserving of serious consideration on the part of chemists. The trouble might be overcome by substituting silex grinders for iron ones, but it is quite likely that an error of equal if not greater magnitude would be caused by their use because of a greater tendency to disintegrate. There is no data available on the subject, and it might be well to look into it. Aside from silex grinders, there seems to be little choice of any other material outside of iron or one of its alloys. It is quite possible that by using manganese steel faces, the abrasion might be reduced, and it would be well to investigate the use of this material.

Experiments recently made in the iron works at Lithgow with certain ore taken from the Mudgee district of New South Wales have proved extremely satisfactory, the ore averaging 42 per cent pure iron. The quality is so good that the owners of the works have made application to the railroad for a daily train to convey the ore to Lithgow. It is stated that there is sufficient ore in sight to keep the Lithgow works going for years.

USE OF PIG IRON MIXERS IN IRON AND STEEL PLANTS.*

In the issue of *Stahl und Eisen* for March 9, 1911, appeared the last of a series of articles on pig iron mixers by Prof. O. Simmersbach of Breslau. The previous articles gave details of design and construction, but the last article is especially interesting because it presents the metallurgical results obtained.

DESULPHURIZATION.

The most important of these is probably the desulphurization, which is effected equally well in heated or unheated mixers. A considerable removal of sulphur, however, is brought about in the ladle, while coming from the blast furnace. For example, in a 750-ton mixer, that was not heated, to which good basic pig iron containing 1.1 to 1.3 per cent manganese was brought over a considerable length of track, the following results were shown:

	Per cent.
Average sulphur at blast furnace.....	0.254
Average sulphur entering mixer.....	0.100
Average sulphur entering converter.....	0.084
Average sulphur of finished steel.....	0.052

The average analysis of the mixer metal for over four days was:

Silicon	0.49	Manganese ..	1.66
Sulphur	0.085	Phosphorus ..	1.98

It was used in the basic Bessemer converter. This shows that 76.24 per cent of the sulphur removed was taken out during transportation; 7.29 per cent in the mixer, and the rest in the vessel. Another test gave 0.25 per cent in the original iron, 0.122 on entering the mixer, 0.113 on entering the converter, and 0.058 in the finished steel. When the iron contained less manganese, less sulphur was removed during transportation, the same amount in the mixer, and therefore more had to be removed in the vessel.

HEATED MIXERS.

In the case of an English 200-ton mixer, in which preliminary refining was done, the following results were obtained:

The mixer was heated with producer gas and air, the air alone being preheated. The incoming pig iron had about:

Silicon	1.4	Manganese ..	1.00
Sulphur	0.14 to 0.17	Phosphorus ..	0.80

Ore and lime were added; slightly less than $\frac{1}{2}$ per cent ore and $1\frac{1}{2}$ per cent lime. The average temperature on entering was 1245 deg. C, and on leaving 1350 deg. The average removal of silicon over a period of five weeks was 30.29 per cent and of sulphur 50.54 per cent. The desulphurization was brought about by the high temperature and also 3 per cent manganese contained in the ore. The manganese in the iron remained the same, or slightly increased, which was also true of the phosphorus.

In the case of another heated mixer, so much ore, roll scale, and lime were added that the silicon was

*From *The Iron Age*

completely removed, and the manganese and phosphorus to a large extent. The pig iron and mixer metal had the following composition:

	Pig iron	Mixer metal
Carbon	3.50	3.97
Manganese	2.60	0.54
Silicon	0.36	...
Sulphur	0.10	0.04
Phosphorus	0.125	0.062

After a discussion of the part played by manganese in the removal of sulphur comes a series of curves giving the results obtained for a year with a mixer heated with blast furnace gas and hot air. No slag-making materials whatever were added. The phosphorus in the pig iron varied from 1.92 to 2.13 per cent. Only a very small amount was removed, and sometimes a slight increase could be noticed. The silicon varied from 0.56 to 0.77 per cent in the pig iron. A considerable part was removed, amounting in some cases to 15 per cent. The manganese in the pig iron varied from 1.48 to 2.11 per cent. It showed a consistent decrease, amounting in some cases to 9 per cent. Finally, the sulphur in the pig iron varied from 0.059 to 0.085 per cent, and was lowered in every case. The removal varied from 21.4 to 38.1 per cent.

The thorough mixing and preliminary refining that may be brought about in a mixer make it applicable to all steel making processes and also to foundry work. For acid Bessemer work its chief usefulness is as a collector, and often iron is needed to recarburize a charge. The silicon in the iron is necessary for the operation, and so should not be lowered.

OPEN-HEARTH WORK.

There is a wider sphere of usefulness in the acid open hearth. Here the excess of silicon may be removed in the mixer which prevents too stiff a slag in the later process. The phosphorus also can be lowered. In these ways the time of the operation is shortened, and less iron is lost in the slag. In the basic Bessemer its importance has been widely recognized and is well known. In the basic open hearth the mixer could serve as a desiliconizing apparatus, and at the same time large amounts of sulphur, phosphorus and manganese would be removed. This would furnish a very suitable material for the open-hearth furnace, shortening the time of the operation.

If the mixer process was so carried on that only a small amount of phosphorus was removed, then the open-hearth slag would be correspondingly richer in this element and more valuable as a fertilizer. This also follows because of the absence of the other metaloids. If, on the other hand, most of the phosphorus was removed in the mixer, then the pure open-hearth slag could be used in the mixer together with the lime and ore necessary. The greatest thing, however, is that the life of the open-hearth furnace would be increased, for the foaming and boiling of the metal would not be so lively or continuous. In this way much less basic slag would be thrown up and carried by the gases against the acid walls, roof and ports, and into the slag pockets and regenerators.

THE RESEARCH AND CHEMICAL LABORATORIES OF FRIED. KRUPP.

The new laboratories of Fried. Krupp, at Essen, Germany, embrace not only those for the ordinary chemical work of the plant but very well equipped ones for chemical, metallurgical and physical research. The laboratories were started as long ago as 1863, were very largely extended in 1883, and in 1895 a special building was found necessary for physical testing. In connection with this latter laboratory, four years later, the first microscopic laboratory in Germany was instituted. The chemical laboratories in 1910 made 476,728 estimations. The present new structure, which houses all these departments, consists of a main building with two large side wings, covering a ground surface of 39,127 sq. ft. The total floor space is 118,400 sq. ft. The northern wing is entirely occupied by the chemical laboratories, in which the following divisions are observed: A—steel analysis; B—special products, ore, etc.; C—gas, water analysis, etc.; D—oil testing; E—powder testing. The chemical-physical testing laboratories are divided into chemical and physical divisions. The main chemical laboratories are for the routine work, while these are for the special steels, etc. The physical division embraces metallographic testing, metallurgical testing, and physical testing proper.

According to the American Gas Light Journal, recent investigation of present day practice in gas houses reveals the fact that the successful operation of water gas sets largely depends primarily upon constancy of volume of air supplied the generator, not upon constancy of pressure. By furnishing a constant volume at high pressure the blast period is cut down, permitting a longer "making period," which results in a material increase in the capacity of the gas generator. To maintain this constant volume, regardless of the varying resistances of the fire, a gate or ture pierced disk will serve, although the Venturi tube is coming into general use. The constant volume is a result of maintaining a differential of pressure through the tube; in other words, if the drop of pressure is kept constant, the volume per any given unit of time will be a constant; therefore, a valve is placed in the tube to open or choke the pipe as much as is necessary to maintain the constancy of the pressure drop. To operate the water gas blowing set for these conditions, the speed is maintained to deliver the maximum volume required at a pressure which will drive the desired volume through the fire, regardless of the existing back pressure due to clinkering, etc. To run a blower at such high speed requires, in the case of a steam driven set, that the turbine be developed for this special service, capable of giving a blast pressure considerably over 30 ins. W.G. In some recent tests with very large blowers, driven by Terry turbines, over 40 ins. W.G. was maintained.



CORRECTIONS FOR THE EFFECTS OF ATMOSPHERIC CONDITIONS ON PHOTOMETRIC FLAME STANDARDS.*

The effects of variation of the pressure, degree of humidity and vitiation (deficiency of oxygen and excess of carbon dioxide) of the atmosphere on the luminous intensity of flames cannot be determined from the variations naturally occurring in an ordinary photometric testing room, except perhaps by deduction from the data obtained in a very prolonged series of observations made with exceptional care concurrently with extremely accurate determinations of the composition of the air. But in ordinary circumstances large variations, especially of one condition independently of the others, are necessarily rare, and the majority of the observations thus made will be of little or no utility in the investigation. Hence the ultimate conclusions even in this case will rest chiefly on a comparatively small proportion of the observations, and, therefore, unless the work has been extended over a very long period, on only a small number of them. The investigation may be carried out in a much shorter space of time and with greater trustworthiness if the observations are made in a room or chamber provided with means by which one set of conditions may be varied at will to any desired degree almost or wholly independently of the other two sets of conditions. The authors have made the investigations of which the results are reported in this communication in such a room or chamber.

THE STEEL COMPRESSION CHAMBER AND ITS EQUIPMENT.

The effects of variations in the atmospheric pressure on the light afforded by the Harcourt 10-candle pentane lamp, and by the Hefner amyl-acetate lamp, were first studied. For this portion of the work, and for some of the later investigations also, use was made of a steel compression chamber, designed primarily for physiological research on "caisson" disease or diver's palsy and mountain sickness. This chamber, which had been presented to the Lister Institute of Preventive Medicine by Dr. Ludwig Mond, F. R. S., was kindly placed at the disposal of the authors by Dr. Martin, the Director of that Institute. It is a short boiler of $\frac{5}{8}$ -in. (16 mm.) plate resting on its side; the ends are slightly dished steel plates $\frac{1}{2}$ -in. (13 mm.) thick. Inside it measures $7\frac{1}{2}$ ft. (2.29 meters) in length by 7 ft. in diameter (2.13 meters)

and has a capacity of 336 cu. ft. (9,500 liters). In one end is an elliptical manhole 24 by 15 ins. (609 by 381 mm.), the cover of which is secured by large bolts and can be readily fixed or removed from outside the chamber. In addition to four spring valves, the chamber is provided with three simple valves by means of which the pressure can be completely controlled either from inside or outside. The pressure is raised or reduced by a simple compressor driven by a gas engine, and it can be raised when required at the rate of about 2 pounds per square inch (about 0.133 kilo. per sq. cm.) per minute. The circulation of air maintained by the pump was sufficiently rapid to prevent serious vitiation of the air of the chamber when two observers were present in it. The Harcourt lamp, when used, was placed below a hood directly connected with the outlet air-way from the chamber, so that the greater part of the products of combustion of the pentane did not mix with the air in the chamber before they were extracted from it. The efficiency of the ventilation of the chamber was checked by a determination of the proportion of carbon dioxide in a sample of the air taken while each set of photometric observations was in progress. These determinations were made by means of the portable apparatus for the determination of carbon dioxide in air devised by J. S. Haldane (*vide Journal of Hygiene*, 1901, I, 109). In the subsequent investigation of the effects of atmospheric vitiation on the light afforded by the Harcourt and Hefner lamps, the proportion of oxygen in the air was determined by means of the portable form of gas analysis apparatus devised by J. S. Haldane (*Journal of Hygiene*, 1906, VI, 74). When necessary an electric fan was used in the chamber to keep the air well mixed.

The chamber was provided with electric light and electric heaters, and a telephone communicating with outside. There were small, stout glass windows in the ends. Two benches were arranged in the chamber. On one of them the photometer, with an electric standard lamp and the flame standard, were set up, while the other was used for the air analysis apparatus and accessories. The degree of humidity of the air was ascertained by means of an Assmann hygrometer (*vide Zeits. für Instrumentenkunde*, 1892, XII, 1) which consists broadly of wet and dry bulb thermometers, over which a regulated current of air is drawn by means of a clockwork fan. A simple U mercury manometer, one limb of which communicated with outside, was used for measuring pressures above normal, and the pressures below normal were read on a standard mercury barometer inside the chamber.

*A paper read at the meeting of the International Photometric Commission, Zürich, July, 1911, by W. J. A. Butterfield, F.I.C.; J. S. Haldane, M.D., F.R.S., and A. P. Trotter, M. Inst. C. E.

DESCRIPTION OF THE ELECTRICAL APPARATUS USED.

A small electric lamp was used as a standard. It was mounted on a carriage guided by V notches. A rod attached to the carriage, as in the Gas Referees' photometer, was used to adjust the position of the lamp. A scale was carried on the rod, and an index was fixed to the photoped.

For setting the filament of the electric lamp as exactly as possible at a measured distance from the photoped, metal plates pierced with vertical slits were fixed on each side of the lamp. The lamp having been removed, a thin steel rule was passed through these slits to form a stop for a measuring rod 30 cm. long; the other end of the latter rested against the face of the photoped. The steel rule and the measuring rod having been removed, the lamp was set up between the slits and its position was adjusted until the filament was as nearly as possible in the plane of the slits. This setting was inspected from time to time, but after about twenty sets of observations had been made it was found that the filament had warped, and that the warping was introducing an uncertainty of about 2 per cent in the later sets of observations of the first series.

As no large battery of accumulators was available during the first series of experiments for giving a steady supply to an electric lamp, portable accumulators were used, and an osram lamp intended to give 10-candlepower at 10 volts was tried, and was supplied by six accumulators controlled by an adjustable resistance. But a comparison with the pentane lamp showed that the colors of the lights differed too widely to make accurate photometry possible until the light of the electric lamp had been reduced to about 1 candle. It was hoped that a metallic filament lamp thus used would have an enhanced "life," or, in other words, would vary less by ageing than if worked at 10 candles. But the result was disappointing. It might have been better if the lamp had been worked at constant volts; but with the view of eliminating the effects of imperfect contacts, adjustment was made for working at a constant current. Another defect in the arrangement was the use of the electric lamp with the filament erect instead of hanging downwards. Notwithstanding the low candle power at which it was worked, the filament warped as has been mentioned.

The lamp took about 0.66 ampere. The exact value of the current was of no importance, but when once settled it was carefully maintained.

The arrangement used is shown in Fig. 1. The lamp *L* was supplied with current from the 5-cell battery *B*, the circuit was completed through an adjustable rheostat of carbon plates *R*, a manganin wire *W* of about 1.54 ohms resistance, and an ammeter *A*. In parallel with the wire *W*, a circuit contained a reflecting galvanometer *G*, a Weston cadmium cell *C*, and a double key *K*₁. A light pressure of this key completed the circuit through the high resistance *R*₂, which saved the cell from injury in case the lamp circuit was not com-

pleted. When an approximate adjustment of the current had been made, the key was pressed hard down and kept closed. Another circuit contained a resistance of 1,000 ohms, *R*₃, and a key *K*₂. When the current was adjusted and the galvanometer *G* indicated zero, closing *K*₂, gave a deflection corresponding to 0.35 per cent difference of current. The current as a rule was kept steady within about one-tenth of this amount.

In the second series of experiments, a large battery being available, a Fleming-Ediswan photometric lamp standard was used, giving 10 candles at 60.05 volts with 0.7577 ampere. This gave no difficulty. The lamp holder was double-wired, and the volts were measured with a potentiometer.

SCOPE OF THE INVESTIGATIONS.

The investigations were undertaken primarily to establish the corrections applicable to the Harcourt pentane 10-candle lamp when the atmospheric conditions were other than normal. It seemed desirable, however, while the arrangements for making such investigations were available to extend them to the Hefner amyl-acetate lamp. Also a few observations

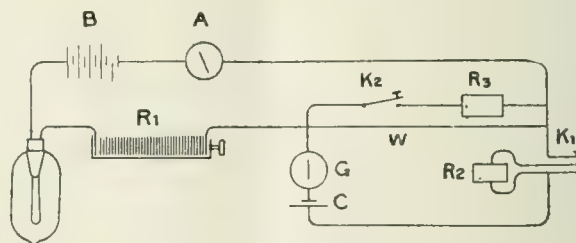


Fig. 1—Arrangement of Electrical Apparatus.

were made on the effect of variations in the atmospheric conditions on the light afforded by the Metropolitan Argand No. 2 gas burner, by a flat flame (slit or batswing) gas burner, and by the English standard spermaceti candle. The results of these investigations, which were not so exhaustive as those on the Harcourt and Hefner lamps, will only be briefly mentioned at the close of this paper. Also the publication of detailed figures of the observations, and of certain deductions on the relations subsisting between the light afforded by flames and the composition of the atmosphere, must be reserved for a subsequent communication.

PROCEDURE.

The procedure followed in making the testings was first to take photometric readings in the normal (or prevailing) conditions, then to modify the particular atmospheric condition, the effect of variations in which it was desired to study, and to take further readings. After several modifications of the same atmospheric condition had been made and readings taken, the original or normal conditions were, whenever possible, reverted to, and final readings taken thereon as a check on the initial readings. In the earliest sets of observations the readings thus obtained were corrected to normal in respect of variations in atmospheric conditions, other than those intentionally introduced, by

employing Liebenthal's or Paterson's formulæ, if available, or otherwise presumptive formulæ. The readings thus corrected were platted on squared paper; and from the curve obtained therefrom the correction applicable to variations from the normal of the atmospheric condition which had been intentionally varied in the set of observations was deduced. The correction was applied in respect of the accidental variations in

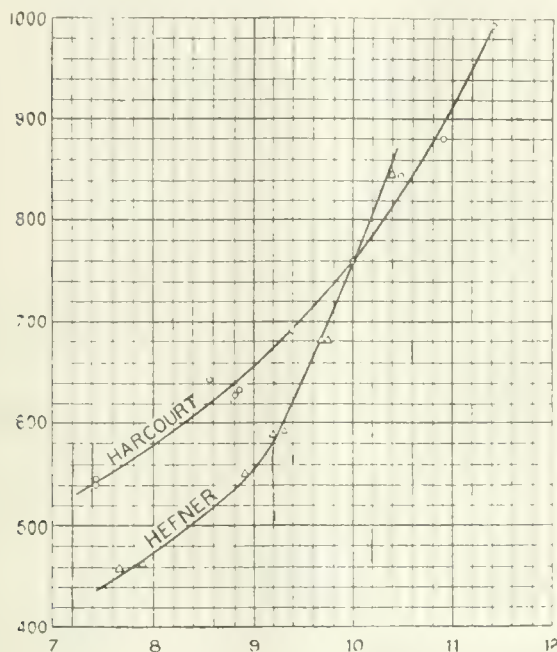


Fig. 2—Curves Showing Relation Between Atmospheric Pressure and Yield of Light.

that atmospheric condition in subsequent sets of observations, in which other atmospheric conditions were intentionally varied. By pursuing this procedure through a number of sets of observations, in which one and then another atmospheric condition was intentionally varied, the corrections applicable in each were gradually deduced with greater and greater freedom from uncertainty. Finally, the results of the whole of the sets of observations were re-calculated with the aid of the corrections so deduced, and the figures thus obtained were re-plotted and any discrepancies thereby disclosed were examined. This procedure involved much tedious re-calculating of the results of the authors' observations, but it led ultimately to corrections being deduced, all of which were independent of the work or formulæ of other observers.

CORRECTIONS FOR VARIATIONS OF ATMOSPHERIC PRESSURE.

The experiments made to determine the corrections for variations of atmospheric pressure were all carried out in the steel compression chamber already described. The range of variation of pressure for the observations was from about 450 to 1,000 mm. (or from about $17\frac{3}{4}$ to $39\frac{1}{2}$ inches). The results of the observations were ultimately corrected for the variations in the proportions of aqueous vapor and of carbon dioxide in the air (according to the formulæ referred to later), and the corrected figures thus obtained have been

plotted and give the curves shown in Fig. 2. The curves show that the relation between atmospheric pressure and the yield of light is not constant or expressed by a straight line. The falling off in the light with reduction of pressure is much greater at the lower than at the higher pressures.

HARCOURT LAMP.

The curve for the Harcourt lamp is, however, nearly a straight line for the range of 700 to 850 mm., and shows that within these limits the light increases or decreases by 1 per cent, with an increase or decrease respectively in pressure of 12.5 mm., or that a variation of 10 mm. in pressure is attended by a variation in the same sense of 0.8 per cent in the light afforded by the lamp. These figures agree exactly with the correction deduced by C. C. Paterson (*Journ. Proc. Inst. of Elect. Engineers*, 1907, XXXVIII, p. 280), which may, therefore, be accepted as correct for variations of pressure at about the neighborhood of normal. In regard, however, to the effect of variations of pressure elsewhere, the authors have failed to find a rational equation to the pressure curve for the Harcourt lamp, and prefer to give the curve of the differential coefficient (shown in Fig. 3) rather than an empirical formula. By the use of this curve, the correction which should be applied to the Harcourt lamp for barometric variations at any place at which the mean barometric height is considerably below 760 mm. may be deduced.

HEFNER LAMP.

The curve for the Hefner lamp is likewise nearly a straight line for the range of 700 to 850 mm., and shows that within these limits the light increases or decreases by 1 per cent with an increase or decrease respectively in pressure of 25 mm., or that a variation of 10 mm. in pressure is attended by a variation in the

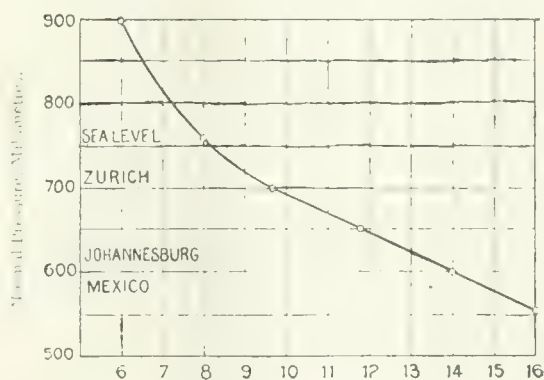


Fig. 3—Curve of Differential Coefficient.

same sense of 0.4 per cent in the light afforded by the lamp. These figures differ from the correction deduced by Liebenthal from observations of the variations of the light of the Hefner lamp, caused by the natural fluctuations of a barometer between 735 and 775 mm. Liebenthal has adopted a formula according to which a variation of 10 mm. in pressure is attended by a variation of only 0.1 per cent in the light. (*Zeits. für Instrumentenkunde*, 1895, XV, 157, and *Praktische Photometrie*, p. 121). Paterson mentions

that he found for 10 mm. variation in pressure 0.2 per cent variation in the light. The authors have therefore found that, about the neighborhood of normal, variation of pressure has only half the effect on the light of the Hefner lamp which it has on that of the Harcourt; but, on the other hand, that it has for the Hefner lamp four times the effect shown by Lieben-thal's formula, which has been very generally accepted. The facts that the effect of small barometric changes, such as Lieben-thal had available, is small compared with the effect of the unavoidable changes in the proportions of aqueous vapor and carbon dioxide in the air, and that the proportion of the latter was not ascertained by Lieben-thal, indicate that his formula for the correction to be applied to the Hefner lamp for variations of barometric height does not rest on acceptable data. The authors ascertained the proportions both of aqueous vapor and carbon dioxide in the air during the observations, and applied corrections to eliminate their effects, while the greater range of barometric change which they had at their disposal reduced the effect of unavoidable errors of observation on the final result. For these reasons, they believe that their figure of 0.4 per cent should now be accepted in preference to Lieben-thal's 0.1 per cent for the effect on the light of the Hefner lamp of a change of 10 mm. in barometric pressure in the neighborhood of 760 mm. In regard to the effect of variations of pressure in other regions of the barometric scale, the curve found by the authors coincides fairly well with the results of observations made by Lieben-thal in pneumatic chamber, in which results, however, he appears to have placed but little reliance because of the presence of a number of disturbing factors.

CORRECTIONS FOR VARIATIONS IN ATMOSPHERIC HUMIDITY.

The experiments made to determine the effect of variations in the humidity of the air on the light afforded by flames were carried out partly in the steel chamber already described and partly in a small room at the Ethical Standards Laboratory of the Board of Trade at No. 8 Richmond Terrace, Whitehall, London. This room was 4.03 meters (13 ft. $2\frac{3}{4}$ ins.) long, 1.975 meters (6 ft. $5\frac{3}{4}$ ins.) wide, and 2.45 meters (8 ft. $\frac{1}{2}$ in.) high. There was no pump for renewing the air of this room, but a hood attached to a flue pipe passing through the wall of the room was placed over the lamp, the heated products from which created a strong draught through the flue pipe. The proportion of aqueous vapor in the air was increased as required by passing steam into the chamber or room from a small boiler placed outside it. A large range in the proportion of aqueous vapor in the air was secured by choosing dry, cold days, on which observations of the light of the lamps when the humidity was low, could be made. The chamber or room having been heated by electric heaters, the air was then nearly saturated with steam at the high temperature. The range of humidity thus secured was from about 1 per

cent to over 4 per cent by volume of aqueous vapor in the air. Sets of observations on the Harcourt lamp showed that a change in the temperature *per se* of the air from 8° C. to over 20° C. had no definite effect on the light afforded. The steam introduced was well disseminated in the air of the chamber or room by means of an electric fan. In one set of observations, greater initial dryness of the air was secured by exposing trays of granulated calcium chloride in the room for 12 hours, while circulation of the air over the trays was maintained by the fan. The results of observations were in all cases corrected for variations in the barometric height and in the proportion of carbon dioxide in the air. The results showed that the light of both the Harcourt and Hefner lamps decreases by 1 per cent for an increase of 0.16 per cent in the aqueous vapor in the air or that an increase of 1 per cent in the aqueous vapor in the air is attended by a decrease of 6.25 per cent in the light of the lamps. Lieben-thal found 5.5 per cent for both lamps (*Zeits. für Instrumentenkunde*, 1895, XV, p. 157, and *Praktische Photometrie*, pp. 118 and 128), while Paterson has found 6.6 per cent for the Harcourt and 6.0 per cent for the Hefner lamp (*Journ. Proc. Inst. Elect. Engineers*, 1907, XXXVIII, pp. 276, 277) as the decrease in the light caused by 1 per cent increase in the aqueous vapor in the air.

CORRECTIONS FOR VITIATION OF THE ATMOSPHERE BY COMBUSTION.

Experiments made to determine the effect of different degrees of vitiation of the atmosphere on the light afforded by flames were made on air vitiated, partially by breathing and partially by the combustion of pentane or illuminating gas. The ratio of carbon dioxide produced to oxygen consumed differs according to the composition of the fuel consumed, and the composition of vitiated air will, therefore, differ according as the vitiation arises wholly or mainly from breathing, from the combustion of a hydrocarbon body such as pentane, or of the mixture of a number of combustible gases, in which hydrogen predominates, which constitutes ordinary town illuminating gas. In the experiments made by the authors, the proportions of both carbon dioxide and oxygen in the vitiated air were determined by analysis, and the somewhat complex results obtained will be dealt with more fully in a subsequent communication. But taking the proportion of carbon dioxide in the air as the sole measure of vitiation caused partially by breathing, but mainly by combustion of paraffine hydrocarbons or town gas, it was found that 1 per cent decrease in the light of the Harcourt lamp was caused when approximately 0.035 per cent of carbon dioxide was present in the air, and in the light of the Hefner lamp with 0.045 per cent. Or the presence of 0.01 per cent of carbon dioxide in the air was attended by a reduction of 0.29 per cent in the light of the Harcourt, or of 0.22 per cent in the light of the Hefner lamp. In the experiments on which these figures for the Harcourt lamp

are based, the proportion of carbon dioxide in the air ranged from 0.09 per cent to 2.54 per cent, and of oxygen from 20.80 per cent to 17.40 per cent. With 2.54 per cent of carbon dioxide and 17.40 per cent of oxygen, the flame of the Harcourt lamp was blue and practically non-illuminating. The diminution in light is fairly uniformly proportional to the increase in the amount of carbon dioxide up to about 2 per cent of the latter, i. e., the curve given by the figures is nearly a straight line to that point. The experiments on the Hefner lamp were not made over so wide a range as those on the Harcourt lamp.

Some observations were made in an atmosphere in which the carbon dioxide was gradually increased by combustion to 1.87 per cent while the oxygen in the air was maintained approximately at the normal percentage by a supply from a cylinder of compressed oxygen. These observations have no direct bearing on the effect of an ordinary vitiated atmosphere on the light of the lamps; but the results are interesting from a scientific standpoint, and will be published later.

GENERAL FORMULÆ FOR CORRECTIONS.

The corrections found for the Harcourt and Hefner lamps may be applied thus:

Let A = the accepted normal percentage of aqueous vapor in the air.

Let C = the accepted normal percentage of carbon dioxide in the air.

Let P = the accepted normal pressure of the air in millimeters.

Let a = the prevailing percentage of aqueous vapor in the air when the lamp is in use.

Let c = the prevailing percentage of carbon dioxide in the air when the lamp is in use.

Let p = the prevailing pressure of the air when the lamp is in use, then the actual light (I') of the lamp at the time, expressed in terms of its light (I) in normal conditions will be found with sufficient accuracy for all practical purposes from the equations:

For the Harcourt lamp:

$$I' = \frac{100 - \left(\frac{a - A}{0.10} + \frac{c - C}{0.035} - \frac{p - P}{12.5} \right)}{100} I$$

For the Hefner lamp:

$$I' = \frac{100 - \left(\frac{a - A}{0.16} + \frac{c - C}{0.045} - \frac{p - P}{25.0} \right)}{100} I$$

EFFECT OF VARIATIONS IN THE ATMOSPHERIC CONDITIONS ON THE LIGHT OF GAS AND CANDLE FLAMES.

The results obtained in the observations made on

the flames of the No. 2 "Metropolitan" argand, and the "G 15" — 5 feet — slit flat flame gas burners, L 10

when consuming the gas supplied in Chelsea and Westminster by the Gas Light and Coke Company, and on the flame of the English parliamentary standard spermaceti candle, must not be accepted as so trustworthy as those given for the Harcourt and Hefner lamps because they are based on fewer observations. These results are, therefore, given with reservation in the following tabular statement:

One per cent decrease of light is caused approximately by the following changes from normal.

	Pressure. mm.	Aqueous Vapor Per cent.	Vitiation ex- pressed by per cent CO ₂ .
No. 2 Metropolitan argand gas	25	+ 0.20	+ 0.03
Flat flame gas	5	+ 0.11	+ 0.035
Spermaceti candle	20	+ 0.50	+ 0.01

These results indicate that small variations in the atmospheric conditions of a gas testing room will not appreciably affect the results of photometric comparisons in which the Harcourt or Hefner lamp is used as the standard of light, and that these standards will give as accurate results as are anyhow practically obtainable in determinations of the illuminating power of gas, if they are used in all ordinary circumstances without correction for any divergence from normal atmospheric conditions. In other circumstances, however, as in the testing of electric lamps by a flame standard, corrections are called for.

THE DIVISION OF THE WORK OF OBSERVATION.

The whole of the work involved in the investigations has been carried out by the authors personally. The work of making the observations as shared in this way: W. J. A. Butterfield remained in the experimental chamber or room and made all the photometric readings and a few air analyses, etc.; J. S. Haldane was also in the chamber or room except on a few occasions when the ventilation was inadequate and the presence of a second observer would have caused undesirable vitiation of the air; he made nearly all the analyses of air and readings of the hygrometer and other accessory apparatus in the chamber or room, controlled the atmospheric conditions therein and generally directed the course of the investigations; while A. P. Trotter had entire charge of the electric standard lamps used and controlled them from outside the chamber or room. When variations in the light of gas flames were being investigated, A. P. Trotter also made, simultaneously with the observations proceeding in the experimental room, observations of the illuminating power of the gas on a photometer which had been set up outside the room, in order that due allowance might be made for fluctuations in the illuminating power of the gas. The electrical and some of the photometrical apparatus was provided and arranged by him.

VOLUMETRIC DETERMINATION OF POTASSIUM BY THE COBALTI-NITRITE METHOD.*

By O. M. SHEDD.*

The writer, in making some determinations on the fertilizer samples sent this year by the referee on potash of the Association of Official Agricultural Chemists, obtained such interesting results in using the cobalti-nitrite method as modified by Drushel¹ for the determination of potassium that it was decided to publish a summary of them, since they throw much light upon the best conditions for applying this method.

The accuracy of this method has been considerably discussed of late and it has been considered of sufficient importance to have been recommended to the above-named association for co-operative work by the referee on potash and also by the referee on soils.

Drushel in his original article had published some results obtained with this method of fertilizers and recommended it for this purpose, and his method, with a slight modification by Bowser,² is the one proposed for study this year. On the other hand, the writer³ has proposed it in connection with the J. L. Smith method for the determination of total potassium in soils and has obtained very good results in this manner. Later on, further work⁴ by the writer and others was presented to the association at its annual meeting last year with the result that this method has been taken up by the referee on soils. Incidentally, it might be of interest to mention that the writer has obtained very good results in using this method on the soils sent this year by the referee on soils for co-operative work.

In the method, as proposed for soils, a change was made in the outline as given by Drushel which, while it may seem of small importance, is very essential in the application of the method. Drushel had recommended that the solution (apparently not restricting the volume) at the proper point was to be slightly acidified with acetic acid and a liberal excess of the sodium cobalti-nitrite reagent added and evaporated to a pasty condition on the water bath. On the other hand, the writer in his article had recommended that the volume should be concentrated to about 5-10 c.c. before adding the nitrite reagent. That this is the best condition under which to work will be shown later, and it is not necessary to discuss it here, except in a general way. The nitrite reagent being unstable at room temperature and especially so on heating, the method that completely precipitates the potassium when the reagent is added with the minimum amount of heating, of course, is the one to be preferred. Again, when the dilution is too large, the nitrite reagent is all decomposed on heating before the potassium salt is precipitated and consequently low results

will be obtained. Finally, the potassium salt itself is acted on to some extent by the acetic acid and water present, and while it may precipitate at ordinary temperature, on heating the precipitate will disappear and the solution become pink. Therefore, if at the end of the evaporation there is not an excess of the nitrite reagent present, low results will be obtained.

To illustrate the above points, a few qualitative tests might be given. The potassium salt was made by precipitating C. P. KCl solution cold with an excess of the reagent and washed with cold water by decantation. A portion suspended in water and heated in the boiling water bath dissolved slowly, giving a clear pink solution after about two hours' heating. This solution evaporated on the water bath in a dish left a blue residue. On taking up with cold water acidified with acetic acid, a notable quantity of the yellow potassium salt remained, perhaps about half of the original quantity, and the solution was pink. On repeating the evaporation and re-solution a second and a third time, only traces of the yellow salt remained.

Another portion was suspended in water and divided equally between two test tubes. To one, a few drops of acetic acid were added and the two heated side by side in the water bath. The potassium salt disappeared much quicker in the one containing the acid than in the pure water. This experiment was repeated with the same result.

Another portion was allowed to stand at room temperature in contact with 2 or 3 c.c. of water, shaking occasionally. After several days, the water became distinctly pink, showing decomposition of the yellow compound.

Another portion was let stand at room temperature in about 50 c.c. of water, with occasional shaking. At the end of a month, all but a trace of the yellow compound had disappeared. On evaporating part of the clear liquid at room temperature over sulphuric acid, there remained a yellow residue, apparently consisting mostly of the potassium double salt, but mixed with pink, needle-shaped crystals, having the appearance of a cobalt salt.

A little of the reagent diluted with 10 or 15 times its volume of water and heated on the water bath soon lost its brown color and became pink, showing complete decomposition of the reagent. An equal portion treated in the same manner, except that several drops of acetic acid were added, underwent the change more quickly. The greater the dilution, the quicker this change takes place.

Upon evaporating the reagent to dryness after diluting with water, as in the above experiment, the mass is distinctly purple in color, and upon adding water there remains an insoluble greenish residue containing cobalt while the solution is pink. The characteristic color of the reagent has entirely disappeared which indicates complete decomposition.

To prove that this greenish substance will have the same effect as the potassium salt, if it should be pres-

*From the Journal of Industrial and Engineering Chemistry.

¹Kentucky Agricultural Experiment Station.

²Chem. News, 97, 124 (1908).

³Journal of Industrial & Engineering Chemistry, 1, 791.

⁴Ibid., 1, 302.

⁵A. O. A. C. Proceedings, 1909.

ent, the following tests were made: two 10 c.c. portions of the reagent were diluted to 15 times their volume and then evaporated until the contents of one was pasty on cooling. The other was further evaporated until the contents were just dry and firm on cooling. They were then taken up with water, filtered, and treated with permanganate and oxalic acid in the same manner as the potassium salt in the method. The following results were obtained:

	Al. KMnO_4 required in titration
Contents on cooling	
Pasty	1.71 cc.
Dry	3.08 cc.

The above results show that if these same amounts of this substance were present in two determinations worked like the above on 0.10 gram aliquots of a potassium salt, the results would be too high to the extent of 1.50 per cent and 2.64 per cent K_2O in the respective determinations.

These tests show that too great a dilution should be avoided, because it decomposes the reagent, and also continued heating in the evaporation should be avoided because in the final stage of evaporation, the last traces of acetic acid may be expelled and as soon as this takes place, the greenish substance is formed more rapidly than it would be if there still remained some free acid.

To show that the above holds true, two 5 c.c. portions of the reagent were diluted to 15 times their volume and evaporated until the volume of one was about 5 c.c. and liquid on cooling. The other was evaporated until pasty on cooling. When water was added to both, there was a small amount of the greenish residue in the first, but considerably more in the other.

This source of error could be avoided in the work, if a few drops of acetic acid were added to the solution before filtering, as this readily dissolves this substance and would leave the potassium salt practically pure.

The above experiments show that both the cobalti-nitrite reagent and the yellow potassium compound are decomposed by heating with water and that the change is hastened in the presence of acetic acid.

The samples sent by the referee on potash were three in number, namely, commercial muriate, kainit and a complete mixed fertilizer. The directions called for a study of the cobalti-nitrite method in comparison with the official platinum method. The aliquots recommended for the volumetric method were 0.10 gram for the muriate, 0.5 gram for the kainit and 1 gram for the mixed fertilizer. These aliquots were used in all of the determinations. The directions also called for the volume of the solution to be about 25 c.c., when 1 c.c. glacial acetic acid and 10 c.c. recently prepared nitrite reagent were to be added and the evaporation made.

The referee's directions were followed in the determinations below, the aliquots for the two methods being taken from the same solution. In the volumetric method, the solutions were heated until the contents were just dry and firm on cooling.

	OFFICIAL METHOD	
	Grams K_2O found	Percentage K_2O found
Muriate	0.26025	52.05
	0.25980	51.96
Average	0.26003	52.01
Kainit	0.06380	12.76
	0.06435	12.87
Average	0.06416	12.81
Fertilizer	0.04660	4.66
	0.04660	4.66

COBALTI-NITRITE METHOD		
Muriate	0.04928	49.28
Kainit	0.05970	11.86
Fertilizer	not determined	

It was apparent that the referee's directions had a tendency to give low results, probably due to the volume being too large when the nitrite reagent was added. To see if an increase of the amount of reagent added at that volume would recover the potassium present, the following determinations were made on the muriate.

	Grams K_2O found	Percentage K_2O found
10 cc. nitrite added	0.04928	49.28
15 cc. nitrite added	0.05022	50.22
20 cc. nitrite added	0.05125	51.25

The following determinations were made by evaporating the solution to about 5 c.c. before adding the acetic acid and nitrite reagent. The reagent was added in various amounts and dishes heated until residues were of the consistency as noted on cooling.

MURIATE			
	Contents on cooling	Grams K_2O found	Percentage K_2O found
Nitrite.			
10 cc.	dry	0.05159	51.59
10 cc.	dry	0.05151	51.51
15 cc.	dry	0.05223	52.23
15 cc.	dry	0.05155	51.55
15 cc.	dry	0.05169	51.69
15 cc.	very dry	0.05142	51.42
15 cc.	pasty	0.05142	51.42
15 cc.	thick paste	0.05159	51.59
15 cc.	very thin paste	0.05134	51.34
25 cc.	dry	0.05095	50.95
25 cc.	pasty	0.05108	51.08
KAINIT			
15 cc.	dry	0.06280	12.56
15 cc.	dry	0.06360	12.72
15 cc.	very dry	0.06230	12.46
15 cc.	very thin paste	0.06315	12.63
25 cc.	dry	0.06355	12.71
25 cc.	pasty	0.06345	12.69

C. P. K_2SO_4 —THEORY, 54.05 PER CENT, K_2O			
	Contents on cooling	Grams K_2O found	Percentage K_2O found
Nitrite			
15 cc.	dry	0.05382	53.82
15 cc.	dry	0.05347	53.47
15 cc.	dry	0.05458	54.58
15 cc.	dry	0.05453	54.53
15 cc.	pasty	0.05390	53.90
15 cc.	pasty	0.05399	53.99
15 cc.	pasty	0.05399	53.99

MIXED FERTILIZER			
	Contents on cooling	Grams K_2O found	Percentage K_2O found
Nitrite			
25 cc.	pasty	0.04680	4.68
25 cc.	pasty	0.04630	4.63

In following the direction of the referee in the above work, the dishes were heated until the residues were syrupy while hot and just dry and firm on cooling. According to Drushel, this is the best plan to follow, for it thus avoids errors in the continued heating of the residues which will vitiate the results. This was done regardless of the time consumed which was noticed varied in similar determinations and very often with different salts.

After some correspondence with the referee, who

stated that the method as outlined by him generally gave satisfactory results in his hands, the writer made some further determinations on the samples and also on C. P. salts. In this work, the referee's directions were followed and also the modification as stated above, whereby the solution was concentrated to about 5 c.c. before adding the nitrite reagent. The time consumed in the evaporation after adding the nitrite reagent and the consistency of the contents on cooling after the evaporation was made were also noted. The evaporations were made in casseroles of 750 c.c. capacity, except in the case of the kainit, when 400 c.c. casseroles were used.

REFEREE'S DIRECTIONS.

MURIATE.			
Time	Contents on cooling	Grams K ₂ O found	Percentage K ₂ O found.
15 min.	thin syrup	0.05187	51.87
45 min.	thin syrup	0.05226	52.26
45 min.	thin syrup	0.05179	51.79
45 min.	thin syrup	0.05145	51.45
		Average,	51.84
40 min.	thin paste	0.05145	51.45
40 min.	thin paste	0.05136	51.36
45 min.	thin paste	0.05104	51.04
45 min.	thin paste	0.05159	51.59
		Average,	51.36
1 hr.	dry	0.05119	51.19
1 hr.	dry	0.05110	51.10
		Average,	51.15
KAINIT.			
50 min.	pasty	0.06300	12.60
1 hr.	pasty	0.06345	12.69
1 hr.	pasty	0.06285	12.57
		Average,	12.62
1½ hr.	dry	0.06275	12.55
C. P. KCl—THEORY, 63.17 PER CENT K ₂ O.			
20 min.	liquid about 15 cc		
	vol.	0.05093	50.93
45 min.	thin syrup	0.06108	61.08
45 min.	thin syrup	0.06358	63.58
45 min.	thin syrup	0.06283	62.83
1¼ hrs.	pasty	0.06342	63.42
1½ hrs.	pasty	0.06325	63.25

The determinations below were made by having the volume about 5 c.c. when the nitrite reagent was added and 15 c.c. of this were used in each case.

C. P. KCl.			
Time.	Contents on cooling.	Grams K ₂ O found.	Percentage K ₂ O found.
20 min.	liquid	0.06383	63.83
30 min.	syrup	0.06366	63.66
50 min.	pasty	0.06383	63.83
C. P. K ₂ SO ₄ .			
20 min.	liquid	0.05401	54.01
30 min.	thin paste	0.05435	54.35
50 min.	thick paste	0.05467	54.67

No reason can be easily assigned why the higher results were obtained on the samples when worked by the referee's directions than those given before, unless it is that the time given to the evaporation is a factor to be considered. In this connection, it might be of interest to note that in the low results obtained before, while following the referee's directions, and in some of the higher results obtained at the same time when the volume was reduced to about 5 c.c. before adding the reagent, the same stock of reagent was used. Evidently, these variations were not due to the reagent, but rather to the different volumes of the solution on adding the same. In support of this, reference might be made to the more uniform results which

have been obtained where this change has been made in the referee's directions.

From the above work, one must recognize that the evaporation is a point that is to be watched in the working of the method and, in fact, an important point which needs further attention before definite conclusions can be drawn.

To determine what other factors influence the results, first the effect of precipitating in larger volumes with the reagent; second, using old nitrite reagent and finally using more or less acetic acid was tried. Also, the plan was tried of adding the nitrite in two portions at different times in the evaporation in order to more completely precipitate the potassium by maintaining the excess of fresh reagent in the solution.

Effect of Volume.—10 c.c. of new nitrite reagent were added to 0.10 gram of the sample in the volume indicated, which included the volume of the reagent added. The evaporations were then made in the usual manner. In the case of KCl, the residue after evaporation consisted mostly of the greenish substance already mentioned and only a small amount of the potassium salt.

C. P. KCl.			
Volume Including reagent.	Grams K ₂ O found.	Percentage K ₂ O found.	
200 cc.	0.00377	3.77	
A. O. A. C. MURIATE.			
100 cc.	0.03533	35.33	

To illustrate further what effect the reducing of the volume of the solution before adding the reagent has on the results obtained, those determinations made on the A. O. A. C. muriate according to the referee's directions and also where the volume was further concentrated to 5 c.c. before adding the reagent are given for comparison.

Volume of solution on adding reagent.	Grams K ₂ O found.	Percentage K ₂ O found.
25 cc.	0.04928	49.28
25 cc.	0.05022	50.22
25 cc.	0.05125	51.25
25 cc.	0.05187	51.87
25 cc.	0.05226	52.26
25 cc.	0.05179	51.79
25 cc.	0.05145	51.45
25 cc.	0.05145	51.45
25 cc.	0.05136	51.36
25 cc.	0.05104	51.04
25 cc.	0.05159	51.59
25 cc.	0.05119	51.19
25 cc.	0.05110	51.10
		Average, 51.22
5 cc.	0.05159	51.59
5 cc.	0.05151	51.51
5 cc.	0.05223	52.23
5 cc.	0.05155	51.55
5 cc.	0.05169	51.69
5 cc.	0.05112	51.12
5 cc.	0.05142	51.42
5 cc.	0.05159	51.59
5 cc.	0.05134	51.34
5 cc.	0.05095	50.95
5 cc.	0.05108	51.08
		Average, 51.49

The plan was also tried of adding the nitrite reagent to the dry residue of the salt after evaporation, but on account of the potassium salt being precipitated so finely divided, that it was impossible to filter it, the determination was discarded. This was repeated with the same result.

Effect of Old Reagent.—The reagent used here had

been made for about eight months and kept in the dark. The referee's directions were followed in the work, except in the last two determinations on the muriate. In these, the volume of the solution was about 5 c.c. on adding the reagent.

A. O. A. C. MURIATE.			
Reagent used.	Contents on cooling.	Grams K_2O found.	Percentage K_2O found.
10 cc.....	dry	0.03531	35.31
10 cc.....	dry	0.03655	36.55
20 cc.....	dry	0.05050	50.50
20 cc.....	dry	0.05076	50.76
KAINIT.			
20 cc.....	dry	0.06085	12.17
20 cc.....	dry	0.06095	12.19

In connection with the above work, the following experiments were tried to see if it was possible to determine colorimetrically how much the old reagent had decomposed on standing. Two c.c. each of the new and of the old reagents were diluted to 50 c.c. in Nessler tubes, and, by comparing the two, it was found that, judging from the color, two-fifths of the old reagent had decomposed.

When the color test was made on the old reagent in the above, the solution had the characteristic brown color of the reagent. After the test was made, the tubes were left standing in the light for about six hours, and during that time about one hour in the sunlight. At the end of the time, the solution in the tube containing the old reagent had changed to pink, showing complete decomposition, while the new reagent solution was still faint brown, but much lighter in color than before, showing a decided decomposition had taken place. After standing about $1\frac{1}{2}$ hours longer in the sunlight, this solution also became pink in color.

Evidently, the reagent must be kept in the dark to prevent decomposition.

Effect of More or Less Acetic Acid.—The plan followed was to evaporate the solution to about 5 c.c. volume, 15 c.c. new reagent added and the amounts of acetic acid stated. The contents of the dishes were evaporated until the volume was about 5 cc., when they were taken from the bath. This was done in order to eliminate any error caused by further evaporation.

A. O. A. C. MURIATE			
Acetic acid.	Contents on cooling.	Grams K_2O found.	Percentage K_2O found.
None.....	thin paste	0.05142	51.42
1 cc.....	thin paste	0.05134	51.34
5 cc.....	pasty	0.05246	52.46
A. O. A. C. KAINIT.			
None.....	thin paste	0.06330	12.66
1 cc.....	thin paste	0.06315	12.63
5 cc.....	pasty	0.06450	12.90

Addition of the Reagent in Two Portions.—The determinations below were made according to the referee's directions, that is, by adding 10 c.c. new reagent to the volume at 25 c.c. The solution was then evaporated to about 5 c.c. volume and 5 c.c. more of reagent were added. The evaporation was continued until the contents were dry on cooling.

Muriate.....	50.99 per cent K_2O
Kainit.....	12.41 per cent K_2O

In the following experiments, an attempt was made

to estimate the potassium by adding the salts which compose the nitrite reagent in separate solutions at the time the evaporation was to be made. In other words, it simply consisted of preparing the reagent in the presence of the potassium salt to be estimated. If found to work, it was thought that this would improve the method, because it would be possible in this way to regulate the volume of the solution, which is important, and at the same time avoid the use of the prepared reagent which decomposes slowly on standing.

For the work 220 grams $NaNO_2$ were dissolved in H_2O and made to 500 c.c. and designated the sodium nitrite solution. 113 grams of cobalt acetate were dissolved in H_2O and 100 c.c. 50 per cent acetic acid added (one-half of the amount prescribed) and the solution made to 500 c.c. and designated as the acetate solution.

A. O. A. C. Muriate.—The solution containing 0.10 gram salt was evaporated to dryness and taken up with 5 c.c. of the sodium nitrite solution and the 5 c.c. of the acetate solution were added, but no acetic acid, and the whole evaporated until the contents were pasty on cooling. Results, 50.59 per cent K_2O .

Another trial was made in same manner, except the acetate solution was added first and then the sodium nitrite. Result, 51.53 per cent K_2O .

C. P. K_2SO_4 .—The same method as followed in the second determination above was then tried on a solution containing 0.10 gram of the salt, and the evaporation continued until the contents were dry on cooling. Result, 52.43 per cent K_2O .

Some further determinations were made on 0.10 gram aliquots of C. P. K_2SO_4 and C. P. KCl by adding the acetate to the dry residue after evaporation and increasing the amount of acetic acid present to that prescribed in the regular method. In this work, 1.5 c.c. of glacial acetic acid and 7.5 c.c. each of cobalt acetate and sodium nitrite solutions were added and the evaporations continued until the contents were pasty on cooling. The results obtained were:

C. P. K_2SO_4 —55.10 per cent and 54.84 per cent K_2O .

C. P. KCl —63.39 per cent and 63.52 per cent K_2O .

The plan of adding the acetate solution first seems to be the most promising one, and further work along this line may prove that this is a safe plan to follow, although as yet there has not been sufficient work done to draw definite conclusions. It is hardly necessary to add here that for the above work, it is better to have chemicals free from potassium, or a blank is all the more essential for work of this kind.

The blank used in all the other work had remained constant and never varied from 0.20-0.23 c.c. $N/10$ $KMnO_4$ solution but the blank here contained too much potassium from the chemicals used, it not being removed as it is in the ordinary nitrite reagent. As no purer chemicals were at hand, this work was discontinued, but it merits further investigation.

According to Bowser,¹ the cobalti-nitrite method for qualitative tests for potassium is much more satisfactory when an equal volume of strong alcohol is added

to the solution to be tested. In this manner, he claims that very small amounts of potassium are precipitated in much less time than from an aqueous solution.

From his work, the writer was led into a series of experiments to see if the above could be used for quantitative purposes. If such would prove to be the case, then the evaporation in the method would be eliminated, which would be a great improvement in the work.

C. P. K₂SO₄.—The plan followed at first was to evaporate the solution containing 0.10 gram of the salt to dryness. To the residue was added 5 c.c. of the acetate solution and then 5 c.c. of the sodium nitrite solution already described. 10 c.c. of alcohol were then added and allowed to stand at ordinary temperature for the time indicated.

Time.	Percentage K ₂ O found.
15 minutes.....	52.20
60 minutes.....	51.99

The next plan tried was to evaporate the solution to about 5 c.c. volume and then add 10 c.c. of the ordinary nitrite reagent and 15 c.c. alcohol. In the first two determinations below, 0.1 c.c. glacial acetic acid was added and in the third, no acetic acid. The solution stood at room temperature for the time indicated.

Time.	Percentage K ₂ O found.
Filtered immediately.....	50.78
5 minutes.....	50.80
15 minutes.....	45.85

In the next experiments, the addition of the alcohol was omitted and the work pursued along a different line. It had been found in working on the A. O. A. C. samples that the addition of more acetic acid than the directions called for had a tendency to give higher results in the regular method.

The plan was then followed of evaporating the solution containing 0.10 gram aliquots to about 5 c.c. volume, cooling, and adding 10 c.c. of nitrite reagent and the amounts of glacial acetic acid indicated. The solution was then allowed to stand for 30 minutes at room temperature.

Acetic acid added.	A. O. A. C. MURIATE.	Percentage K ₂ O found.
1 cc.....		48.98
5 cc.....		50.38
15 cc.....		Could not filter

Evidently here, as in the regular method, an increase of acetic acid seems to give a higher result, but whether this increase is due to more potassium precipitated or to other causes is not easy to determine.

It was thought that perhaps the increase of acetates present in the solution influenced the precipitation of the potassium by the nitrite reagent and to prove this, if possible, the following experiments were made on C. P. KCl. Two 0.10 gram aliquots of the salt were taken and to one, after evaporating to dryness, was added 10 c.c. of a saturated solution of sodium acetate and 10 c.c. nitrite reagent. To the other aliquot, after evaporating to about 5 c.c. volume, was added 10 c.c.

reagent and then 10 grams of sodium nitrate. Both were allowed to stand 1 hour at room temperature and gave respectively 55.02 per cent K₂O and 48.34 per cent K₂O.

To determine if it was possible to completely precipitate the potassium at room temperature by this method, if the volume of the solution was reduced, the following experiment was tried on C. P. KCl.

Two 0.1 gram aliquots were evaporated to about 5 c.c. volume, 1 c.c. glacial acetic acid and 10 c.c. nitrite reagent added and the whole put in a desiccator over H₂SO₄ and the air exhausted as much as possible by means of the suction pump. It was then allowed to stand about 24 hours and at the end of that time, the solution had evaporated and the residue was nearly pasty. Result, 65.74 per cent and 67.52 per cent K₂O.

In every determination that has been made without heating the solution, it has been impossible to wash the potassium sodium cobalti-nitrite with water until the filtrate was colorless. While the washings did not contain any of the precipitate in suspension, they had a yellow color which seemed to indicate that either the salt itself was dissolving, or, which is more probably the case, it was contaminated with some substance which made it difficult to wash clean. While it is easy to understand how long results are obtained while working in this procedure, the high results can only be explained on the supposition above, that the potassium salt contains some other compound that has been formed which is difficult to wash out and reacts with permanganate.

From this experience with the Drushel modification of the cobalti-nitrite method, the writer believes that it is very accurate if properly handled, but an inexperienced worker not knowing its weak points may not have this opinion with his first use of it. Briefly stated, the outline of the method which has given the best and most uniform results in this work is as follows:

The solution containing the potassium salt, after the preliminary work has been done to get it at this stage, is evaporated in a 500 c.c. casserole to a small volume of about 5 c.c., slightly acidified with acetic acid and 15 c.c. fresh nitrite reagent added. The larger amount of nitrite reagent makes possible a better filtration and a good excess of reagent after the evaporation is made. The solution is evaporated on the water bath for about 45-60 minutes, or until the contents become a thick syrup while hot and pasty on cooling. Continued heating is to be avoided, and this is important to obtain good results. After the filtration is made and the casserole washed, the Gooch crucible and contents can be put in the same casserole and treated with permanganate, as prescribed in the method.

In conclusion, the writer desires to express his sincere thanks to Dr. A. M. Peter of the Kentucky Agricultural Experiment Station for the valuable advice he has rendered during the progress of the work.

¹Jour. Am. Chem. Soc., 32, 78; Chem. News, 101, 100.

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NOTES AND COMMENTS.

Promotion of Commerce by the Federal Government.

The Bureau of Manufactures of the Department of Commerce and Labor has just issued a pamphlet of 15 pages, called "Promotion of Commerce," which every manufacturer and business man would find useful on his desk or in his files.

After a rather full description of the work of the Bureau of Manufactures, the pamphlet outlines briefly in short paragraphs, somewhat after the manner of the Congressional Directory, the duties and functions of various branches of the government which are carrying on service related to the promotion and development of trade and manufacture. Over 30 bureaus and branches of the government are listed from the Departments of Commerce and Labor, State, Interior, Agriculture, Treasury, War, etc.

The Pennsylvania Railroad Department of Tests.

While wide publicity is being given to statistics showing, or purporting to show, how the railroads waste millions every year through unscientific methods, some railroads are endeavoring to show what they are doing to promote economy.

The Pennsylvania Railroad has departments for the sole purpose of insuring it against loss through the use of poor materials. These departments do not bring in a dollar of direct revenue, but every year they save the company hundreds of thousands of dollars above the cost of their maintenance. Over two hundred men are employed in these departments, and among them are some of the most skillful engineers and chemists in the country. The departments are known as the Test Department and the Chemical Laboratory. The first is in charge of the Engineer of Tests and the second is operated under the direction of the Chemist of the Pennsylvania Railroad. Both are situated at Altoona, Pa.

As an instance of the saving possibilities of this department, a short time ago the engineers in charge of the Locomotive Testing Plant noticed that a certain coal was not producing the right amount of steam per pound. Upon investigation it was found that the coal was coming from an outcrop, a vein lying outside the ground, where it was damaged by exposure to the weather. Shipments from this source were immediately stopped when the attention of the company furnishing the coal was directed to the matter. On the road it would probably not have been noticed, and it is estimated that the discovery saved enough to pay a month's expense of operating the Testing Plant.

Scientific management to Nth power is practiced by the Pennsylvania Railroad in the purchase of supplies, and by supplies is meant everything from a rubber band to steam locomotives. Inspectors are stationed so as to be available at every manufactory from which materials or equipment are bought. Not only are the finished products subjected to a rigid examination before they are accepted, but the material which is bought by outside concerns must pass the scrutiny of these inspectors if it is destined for equipment ordered by the Pennsylvania.

These outside inspectors are under the direction of three resident inspectors with headquarters at Altoona, Pittsburgh and Philadelphia. Some things, such as car couplers, axles, etc., they test themselves at the works where they are made. Usually, however, specimens are sent to the laboratories at Altoona, where they are put through physical and chemical tests, the results of which are compared with the specifications.

The company relies on its experts to protect it and will accept nothing that does not come up to the specifications prepared by them. These cover practically everything that is used by a railroad. The wide range of articles they embrace is shown by a glance down the list, from automatic couplers, tin, and lumber for ties and telegraph poles, to caustic soda, Tuscan red, soap, passenger car thermometers, and sponges. Every year new specifications are added and the old ones are constantly being revised to conform with more complete knowledge or more stringent requirements.

The thoroughness and independence of this branch of the work is illustrated by some special experiments that are now being made in the chemical laboratories. At present there are no chemical specifications for rubber, except those prepared by the government. Instead of accepting these, as others have done, the Pennsylvania's experts are making an exhaustive study of the subject in order to have specifications that they *know* are adequate.

Thirty-five thousand eight hundred and seventy-two samples of various materials were examined in the chemical laboratories in 1910, and 121,970 determinations made. Among the special subjects investigated were drinking water, disinfectant, paint and varnish removers, steel wheels, rails, etc., with studies on smoke-preventing devices.

Theoretically every bolt and rivet, every piece of wood or steel that goes out on the Pennsylvania Railroad is competent to do the work allotted to it, with a store of reserve strength for any extraordinary strain.

If the theory on which the Test Department works could be carried out perfectly in practice there would be no breakage. But, naturally, this is impossible where the work is done by men who are bound to make mistakes sometimes. Broken parts are always sent to the laboratories, and a large part of the business consists of investigating the causes of such breakage in order to prevent repetition. Almost incalculable loss is avoided in this way. If the break is due to a miscalculation the department can lay its finger on every other piece of equipment that could have been affected before more damage is done. Or, if it is due simply to wear from age, it offers a clue for the investigation of similar parts that were put in use at the same time.

The Pennsylvania Railroad began the testing of materials in 1875. The establishment of the laboratory took place in 1879, with a force of only four men, two on chemical and two on physical tests. From that time the history of the department has been one of rapid growth both in size and importance, until today it is one of the vital parts of the great railroad system. It is a sort of bureau of scientific management where problems arising daily are studied out, and where infinite time and pains are expended in working towards the perfection that means the least waste and the highest efficiency.

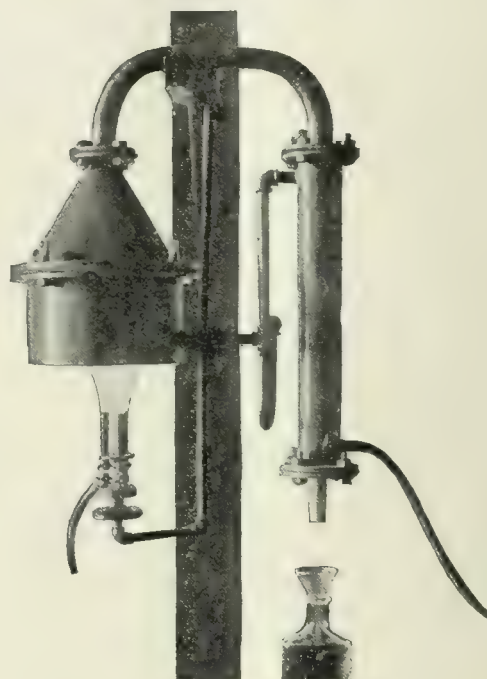
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GALVANIZING WIRE IN ZINC-DUST.*

By ALFRED SANG.

Ever since I presented before the American Electrochemical Society my first paper on galvanizing steel wire by heating it electrically in zinc oxide or zinc dust, I have realized that a perfectly clear conception of the forces at work was essential for successfully operating the process. Once the nature of zinc was well understood and the action which different temperatures may have on it was clearly demonstrated, the application of the process was found to be, after all, not nearly as complicated as might have been feared, in fact it was less complicated than the other two processes in general use.

It must be recalled that zinc dust does not melt when heated to the temperature of fusion of zinc, but at higher temperatures reducing agents will "precipitate" the metal, as it were, and it can also be squeezed out under mechanical pressure. This property of resisting fusion is supposed and is probably due to the surface of each particle being formed of zinc oxide or zinc carbonate.

If zinc dust is heated evenly, the temperature is found to rise steadily until around 390° C. when the rise becomes much more rapid, attaining its maximum rate at about 420° , then falling gradually until in the neighborhood of 470° it again becomes normal. When the dust is cooled down an opposite effect is noted between 450° and 260° . If the operations of alternately heating and cooling the same dust are repeated, the exothermic and endothermic effects are manifested each time between the same temperatures, with their apices at about 420° on the rise and about 350° on the fall.

We cannot help but notice that the exothermic effect on the rise occurs at the melting point of zinc metal, but when zinc is melted the effect is the very reverse, heat being absorbed at the moment of fusion to the extent of 28 calories per kilogram.

We are therefore in presence of an anomalous critical point, and I will now consider separately the operations of galvanizing in zinc dust below and above that critical point.

If a clean piece of steel wire is immersed in zinc

dust which is heated to a temperature below 420° , or at what is known as "sherardizing temperatures," a slow cementation takes place and zinc is absorbed from the very moment that the wire reaches the temperature of the dust; there is at the same time a slow sublimation of zinc at the surface increasing with time but sufficient for all practical purposes after a few hours of treatment; in the presence of reducing gases the time of treatment can be considerably lessened but the penetration of zinc is of course less. This process is sherardizing, unsurpassed for many purposes, unsuitable for many others. There are two varieties of sherardized coatings: the gray, porous and brittle variety, mostly composed of zinc carbonate which darkens with age; other variety has a high metallic luster and is compact and of a finely crystalline structure. If this were the class of sherardizing produced by all existing plants, there would not be much hot-galvanized work produced outside of wire and sheets.

When we work with the dust at a temperature above the critical 420° , results are entirely different; galvanizing is *instantaneous*, taking place the very instant that the wire reaches the temperature of the dust. There is very little penetration of zinc in the iron; there may be vaporization and condensation of zinc as in sherardizing, although I doubt it. A wire heated to 800 or 900° in a non-oxidizing flame and quickly thrust into cold zinc dust will condense around itself a considerable thickness of zinc. The best working temperature ranges between 500 and 600° C. The galvanizing is bright and more pleasing in appearance than that obtained by any other means.

I will not go into details of the furnace used, which merely consists of a zinc-dust trough between two sand troughs provided with a set of cast-iron cover-plates over the zinc trough and power-driven rolls. As high a speed as 850 ft. per minute can be used for securing good merchant galvanizing. It is found preferable to run at 35 ft. only, running a large number of strands and attaching the coils end to end.

The process is economical as regards current consumption which, outside of zinc and labor with a very small amount of power for driving the rolls, represents the bulk of direct cost. Figures are readily arrived at, for as soon as the dust has attained a fairly constant temperature, as it very rapidly does, the con-

*Paper presented at the Twentieth General Meeting of the American Electrochemical Society, in Toronto, Canada, Sept. 21-23, 1911.

sumption of electricity in watt-hours is represented by the formula

$$\text{Watt hours} = \frac{w \cdot s \cdot t}{u} \cdot \frac{7}{6}$$

where

w = weight of wire treated, say 1,000 kilograms.
 s = specific heat of steel, say 0.18 at 550°.
 t = temperature to be maintained, say 550° C.
 $w s t$ = calories needed, which, multiplied by the ratio 7/6, gives the watt-hours.

Figuring 75 per cent efficiency for the dynamo or transformer supplying the current, the above formula gives 154 kilowatt-hours per long ton of wire of any diameter.

While the wire is softened by the process, a full annealing can only be secured by using a higher temperature, the consumption of electrical energy being then increased from 40 to 50 per cent and the sand trough through which the wire passes after leaving the zinc dust must be appreciably lengthened. If a slight scale is to be removed, the high temperature must be used in connection with a reducing agent. Sand must be used with the zinc dust to prevent caking and cavitation in the path of the wire. Aluminum contact rolls and special mercury contacts are the only ones suitable for this kind of plant.

The wire can be brought out into the air immediately after coating, but it is preferable to let it cool by passing it through sand which brightens it at the same time.

The instantaneous galvanizing of steel in zinc dust above 420° is readily explained by completing the theory which I developed as far back as 1907; I will now state it in its complete form.

Zinc dust is produced by the sudden cooling of the vapor of zinc and the transition from the gaseous to the solid state is so rapid that for all practical purposes, the liquid state is skipped. The setting of the zinc, brought about by sudden chilling, prevents regular arrangement of the molecules along determined stereochemical lines; it forbids the formation of crystals for which time is required. The heat of solution of matter in an amorphous state is always higher than when in a crystalline state; the same is true of a strained condition which likewise presents an amorphous appearance. At the time of formation, considerable heat is retained which would have been radiated by slow cooling. The condition is one of physical instability resulting in readier and, oftentimes, explosive decomposition.

The physical instability of zinc dust may be explained thus: in the prolongs of zinc furnaces the uncondensed vapor is suddenly chilled and collects in minute drops which are instantly congealed at the surface and oxidized, forming a rigid spherical skin or crust. Within this crust the liquid zinc cools and contracts more slowly but the rigidity of the crust does not permit the drawing in of the surface to form the facets of a crystal and compensate this internal contraction. Thus, voids must exist within the crust and

we may presume a structure which might be described as "piped radially" or as negatively crystalline.

Below the melting point of zinc this condition persists but affinities and thermal inducements find a readier response from zinc dust than from any other form of zinc, both by reason of the enormous amount of free surface and because of this unstable condition.

At about 415°, which is the melting point of zinc, the strain within the zinc dust particle is totally or almost totally relieved as indicated by the loss of heat, although some of the latent heat must be retained for fusion.

At high temperatures zinc dust is pasty, because formed of innumerable little bags of melted zinc, but these will not coalesce on account of their small size and closely packed condition, unless considerable pressure is used. These minute bladders might be compared to soft-shell turtle's eggs; they form a plastic and cellular mass.

If, however, a reducing agent, say a drop of coal tar, is introduced, reduction of the skin takes place and we get a small mass of more or less spongy metallic zinc. If a metal which is electro-negative to zinc, such as iron, is put into contact with these bags of zinc, reduction and an exchange of metals take place, the bag is punctured at the point of contact and the zinc spreads itself on the iron.

Hence it is that wire can be galvanized in this manner by means of a plant closely resembling a hot galvanizing plant, for the process is, in fact, one of hot galvanizing.

EXTRACTION OF GASOLINE FROM NATURAL GAS.

The Honolulu Consolidated Oil Co. some months ago completed what is stated to be the largest plant in the world for the extraction of gasoline from gas—an industry which is yet in its infancy. The Honolulu company's plant is located in the Buena Vista hills in California, almost on the top of the range in which has been developed an extensive natural gas belt. The following description of the gas belt and the plant is taken from the *California World*:

"Not all of the gas in the Buena Vista hills contains gasoline to an extent that it can be profitably handled. The gas found in the upper strata is practically free from it, and that of the second strata contains but a small amount. But with greater depth the gas, which evidently has passed through or been generated in the formations which contain petroleum, is quite heavily saturated. In a strictly gas well the tests conducted in the small test plant on the Honolulu showed about 5 quarts of gasoline to the 1,000 ft. of gas. In oil wells the casing-head holds gas as high as 25½ gals., where the product of the wells is 26 gravity.

"The Honolulu plant was purchased from the Bessemer Gas Engine Co. It comprises four units each consisting of two 80-hp. Bessemer air compressors. The capacity of the entire plant is 4,000,000 cu. ft. of

gas in 24 hours, but the plant will probably not be run above 85 per cent of its capacity, in order to get the best results.

"The gas is taken in at zero, or a slight vacuum, through a 6-in. main through a low pressure machine and is discharged from that machine at a high pressure of 40 lbs. through a single header, where any water in the gas is condensed and taken off. From there it is taken through a 3-in. main into a high pressure machine, from which it is discharged, through a double header at a pressure of 300 lbs., through an expansion coil of 160 ft., where residue gas from a high pressure tank finishes the cooling process. The condensed liquid then passes through a series of brass tubes, where it is purified, and from there goes into receiving tanks of 2,000 gals. capacity, from where the gasoline is drawn off under high pressure to stock tanks.

"The distillation of the gasoline consumes from 10 to 15 per cent of the volume of gas and the compressors, which are operated by gas, another 5 per cent. The residue gas is discharged at a low pressure into the mains and is ready for use as a fuel. It is claimed that not only is the gas still suitable for fuel, but that it is superior for domestic use, as it is clean from soot and dirt.

"The Honolulu plant is complete in installation and lacks but the connection with the mains from the wells and the building of a structure in which to house it.

"The Menges Oil Co. has given a lease to the Pacific Gasoline Co., on the gas from the rich Menges well, for a period of 20 years, 25 per cent royalty. The Pacific Gasoline Co., of which Dr. A. H. Liscomb, of the Producers' Refining Co., of Bakersfield, is president, has installed a single unit gasoline plant for the extraction of the gasoline, which was put in operation Sept. 1.

"The oil from the Menges well tests 32 gravity and the gas is correspondingly prolific of gasoline, containing 3 gals. to the 1,000 ft. The production is a little under 1,000,000 cu. ft. in 24 hours. The plant installed will have a capacity of 750,000 ft. and will turn out not less than 2,000 gals. of gasoline each day. At 20 cts. per gallon, a low price, this means a revenue of \$400 per day, or \$12,000 per month.

"The share of the Menges company in this new source of revenue, which heretofore has gone to waste, it is stated, will be \$3,000 per month."

The production of phosphate rock in the United States in 1910 was considerably larger than in 1909. In 1910 the total marketed production was 2,654,988 long tons, valued at \$10,917,000, as compared with 2,330,152 tons, valued at \$10,772,120, in 1909, an increase in quantity of 324,836 tons, or 13.9 per cent, and in value of \$144,880, or less than 2 per cent. The increase was mainly in Florida rock, although the Tennessee production also showed a considerable gain. The average price of the rock was \$4.11 in 1910 as against \$4.62 in 1909, a decrease in 1910 of 11 per cent.

ALLOYS.*

By W. R. WHITNEY.†

This is not intended as a paper describing accomplishments, but is aimed at contributing, if possible, to increase the interest of experimenters in new alloys. It is the result of reading Mr. Stephenson's paper on "The Value of the Association to Its Members," and a letter from the secretary, asking for a contribution to the papers of the American Brass Founders' Association.

I want to call attention to the possibilities in the way of useful discoveries, which may well lie more nearly within reach of some of the members than they realize, because of their particular knowledge or possession of special materials. For example, it frequently happens that one manufacturing company produces a new metal or alloy for some particular use, which, owing to lack of general study, may remain the sole use for a long period of time. Probably those who produce the special cerium-lanthanum-iron alloy used in the various types of friction cigar lighters, have not been able to study very thoroughly the application of this alloy to other uses. So also, those who use the very modern metals, tantalum, tungsten and molybdenum, have been engrossed with applications to a single field, and have hardly been able to look carefully into others. It is often through the interchange of information across the gap between quite remote fields that useful advances are made.

It occurred to me that even an imperfect review of some of the relatively recent metals might not come amiss. Probably few realize the rapidity with which new metals are coming into use, particularly in alloys. There are in all approximately 50 metallic elements, though most of our important industries employ but a few of them. Some of these, in the metallic state, have market prices which are not yet controlled by the cost of production, nor by the infrequency of occurrence, but rather by the lack of development of a utility. Beginning with gold, which we may assume is the one element whose exchange value depends upon its commonness in nature plus its cost of production; and passing over iron, copper, lead and zinc, whose values may be said to be well fixed by occurrence and costs of production, we soon reach other metals, for which a new demand might well greatly reduce the cost. Among these are many whose ores occur in abundance. In the case of this type of element, the interest attached to research work is doubly great.

It is highly improbable that the cost of copper will ever be greatly changed by the discovery of new uses. This is because the world's supply of the ore is pretty well known; the demands are high and the cost of production of metal from ore have been so studied that further reduction will probably only be of what I like to call the second order of magnitude. This was not

*Paper read May 24, 1911, at the Convention of the American Brass Founders' Association.

†Research Laboratory, General Electric Co., Schenectady, N. Y.

true of the metal aluminum a few years ago, and it is still possible that considerably wider uses and reduction of production costs may develop in its future. There is apparently a much wider divergence between the occurrence of the aluminum in nature and its price in metallic state than in the case of copper. In case of aluminum, only selected and purified ores are used at present, while other compounds of it occur everywhere in nature. On the other hand, in the case of copper, ores containing even less than two per cent of copper are worked for the metal. Aluminum may thus still be considered in the transition state, a state long ago passed by copper and iron, and not reached by some of the metals considered below. We have all been witnesses of the interesting advance of aluminum. From 1869-74, its properties were becoming generally known. The inefficient process by which it was then reduced from its ores made it impossible to sell the metal below \$10 per pound. Advance was then made so that in the 80's the price was about \$5. There was not a great demand for it at this price. In the year 1907, something like 26,000,000 lbs. of this metal were made in America alone. The price is now about 20 cts. per pound.

The element *calcium*, which a few years ago was listed only as museum specimens and at several dollars per gram, was sold in 1908 at \$1.50 per pound, and could certainly be sold for a small part of this price if a greater use could be found for the metal. It slowly decomposes water, giving hydrogen, and it differs from the alkali metals in producing such a feeble alkaline solution, that it is generally harmless. It ought to serve as a good deoxidizer, and should be a very cheap metal. It is not fair to relegate it to a list of useless metals. History of the metallic arts points to there being no such list.

Thallium is an element quite similar to lead, but probably possessing some property which will some day warrant its exploitation. It is softer and heavier, and could be obtained in quantity, if a demand were created.

The elements *chromium*, *molybdenum*, *tungsten* and *tantalum*, the three latter now obtainable in wire-form, are tempting elements to study in mixtures with others. Who knows the useful properties of a chromium bronze, for example?

Tellurium has long been an apparently useless metal, and any market price is fictitious, as there is but little isolated in metallic state. It is not necessary that a great use, such as a substitute for zinc in brass, should be found for it. Our industries are so great, that if a pound of tellurium added to the ton of aluminum was of benefit to the latter, the production of the necessary tellurium would be real industry.

Consider *cobalt* a moment. The world's rate of supply of ore has been greatly augmented. It may take time to actually realize a greatly reduced cost of metallic cobalt, but we ought, notwithstanding, to realize it when uses have been developed. Our natural im-

pulse in such a case, is to try direct substitution of one metal for another in some well developed use. Cobalt, for example, might replace nickel in most uses when the cost fell below that of nickel, but this is a second order use. A first order use would be the supplying of a want which no metal previously supplied, or supplied distinctly less perfectly.

In this connection, an interesting alloy of cobalt and chromium has just been described by Elwood Haynes in the October number of the *Journal of Industrial and Chemical Engineering*, and it is altogether probable that technical use will soon be made of it.

Many tons of metals are annually consumed as resistance wire for electrical purposes. At one time iron was the element most used. German silver replaced it in some cases, where a lower temperature co-efficient was needed and the increased cost was permissible. Now there are a dozen or more special alloys for this particular electrical use. The new ones have far out-classed the old in most of those properties for which the electrical engineer uses them. In such alloys, nickel, chromium, manganese and others are now used by the ton.

Silicon, which in 1900 was a curiosity and sold for 40 cts. per gram, is now a necessary component of special iron alloys and of high-grade transformer iron, and the world uses thousands of tons of the alloy annually. Silicon is now sold at about five cents per pound. The use of this metal in other alloys is still quite limited. In the case of iron, it greatly decreases hysteresis loss and increases electrical resistance.

Boron, still a quite expensive material in metallic state, is coming into commercial use in assisting the making of solid copper castings of high electrical conductivity.

Vanadium seems to be a young wonder working metal. Its use has increased very rapidly in the past few years, but the quantities consumed are not known to us. As several companies are producing the iron alloy, it is safe to assume that it is being sold by the ton. The price for the metal in the alloy is not far from \$5.00 per pound.

Cadmium is a beautiful metal in many respects, and it is certainly awaiting use. It is whiter and less crystalline than zinc, and doubtless the high price of nearly a dollar a pound keeps practical workers from trying it in their experiments. It should be produced as cheaply as aluminum, if there were a good demand for it.

Titanium is an element long the subject of criminal negligence. It is a high-melting ductile white metal, which, at present, is only separable from its ores at high cost. It exists in many cheap ores widely distributed in nature. It is now apparently coming into use in steel manufacture, particularly for railroad rails, and for this purpose it is fortunately unnecessary to isolate the pure titanium from its ores, an iron titanium alloy being produced directly. What will happen when the pure element has been tried in special fields, can

only be surmised. The optimist sees great chances. The pessimist feels himself busy living with the optimist.

If one omits the common alloys, brass, bronze, solder, etc., and considers only possible alloys of two metals, and still confines himself to 20 of the common metals, like vanadium, manganese, chromium, boron, etc., he is interested at once to recognize that there must be one hundred and ninety different pairs of binary alloys. When, in addition, the effect of varying proportions in these alloys is considered, it becomes evident that the field of alloy-research is truly a large one. Many of the alloys apparently unstudied, are those which melt at extremely high temperatures.

The brass founder who knows the upper limiting temperature of his melting furnaces may at once point out that this temperature is fixed both by the life of crucibles and the particular coke or oil heating schemes with which he is familiar. If he thought that a molybdenum bronze of 80 per cent molybdenum, would have useful properties compared with all other alloys, he might at once conclude that he must give up his alloy because of the difficulty of melting it. If it were not for the advances in our available temperatures, there would seem to be little more than amusement in considering alloys high in tantalum, in chromium, in titanium, in molybdenum, in silicon, in uranium, in vanadium, and a number of other high melting metals, but hand in hand with the discoveries leading to isolation of such metals go also discoveries of aluminothermics, oxyacetylene and oxyhydrogen temperatures and electric furnace processes. The time is always ripe for the study of new alloys with new tensile strengths, elasticities, colors and wearing powers. The automobile and the aeroplane have forced the aluminum and iron alloys to make rapid strides, and it is natural that we should want to inventory our possibilities. The physical chemist has started along the way of a systematic co-ordination of certain properties of binary, and in a few cases tertiary alloys. He has shown how to plot a few freezing points of two-metal and three-metal mixtures and to construct therefrom curves showing not only all possible alloys but what may be expected in the way of segregation and structure and such effects as caused by annealing or quenching.

He has found that there is a solubility of metals in one another which varies just about as the solubility of substances in water varies. Metals may be melted together and well mixed, but the quality and permanency of the mixture is determined by just such solubility laws as control ordinary solutions. We know that in some cases well-mixed melted metals will separate into two layers if allowed to remain even a few moments in molten condition at low temperatures. They act like a mixture of water and ether. The two separated layers contain both metals, no matter what the temperature, but the quantitative compositions depend on the temperature.

The other extreme of metal solubility is found in such a case as zinc-cadmium, which acts much like a mixture of alcohol and water, the two components

going into solution in all proportions and remaining in solution at all temperatures. Having seen this analogy between the facts of solubility of substances in water, it is natural to search among the metal mixtures for all the peculiar kinds of solution observed in aqueous solutions. Two such classes interest us at once. They are those corresponding to aqueous solubilities where temperature widely influences the quantities dissolved, and those in which the solvent (as water) combines with the dissolved substance more intimately than by simple solutions, as by chemical combination. In the case of zinc and lead we have one of the metal alloys of limited solubility. If these two metals are well mixed in liquid state, they separate into layers—one, the zinc, carrying a few per cent of dissolved lead floats on an alloy made up of lead carrying a few per cent of dissolved zinc. In general, the quantity of the one metal dissolved and held in solution by the other, depends on the temperature, and the higher the temperature the greater the solubility. Between 900 and 1000° C., they are apparently completely soluble. It follows from this that when a dissolved pair of metals is cooled slowly, one of them may separate on cooling, if the limiting solubility is reached, and the extent of effective separation may depend on the rate of cooling.

Our second case, that of chemical combination between the metals, is made most evident by the form of the freezing point curve of the possible alloys. A compound of two metals which is stable at a temperature above the melting point of one or both of the metals, shows very clearly on the melting-point curve and acts towards each of the elements just as a new, or third element. Its melting-point cannot yet be predicted from any knowledge of the component metals. It may even melt higher than either of the components. Such cases are seen in alloys of aluminum-antimony, in lead-tellurium, etc.

Man first used the metals as he found them; then, as he reduced them from the ores, and finally, when specified requirements became more and more exacting, he not only brought into use previously unused metals, but also greatly modified the old familiar ones. For a harder iron he used steel, a carbon alloy; for a harder steel, or one capable of cutting iron more readily, he added tungsten, nickel, chromium and other metals. For permanent magnets molybdenum was added, for high electrical resistance nickel, chromium, etc., were added, for low electrical resistance and low hysteresis, silicon and aluminum were added; for toughness in springs a little vanadium was used, and for wearing qualities titanium is now introduced. These are only a few of the successful alloying experiments with iron. They will probably be repeated with other metals, such as copper, zinc and aluminum, where the cost of the base metal is not high.

On the other hand, the study of those metals which have not yet advanced to a stage where first order cost-reduction is impossible, is equally interesting. Consider again, the element chromium. What do we know about it? Is it a workable metal? Can it be ham-

mered or cast? Is it permanent in the air? Is there a considerable possible ore supply? Has the cost of obtaining the metal been reduced to what seems a reasonable rate? etc., etc. As it is unlikely that such an element will suggest itself for use by men as did copper and iron, it is probable that its properties must first be determined and made known. As a metal, it is only about fifteen years old. It is made in the metallic state by reduction of the oxide by metallic aluminum and also by electrolysis of its salt solutions. It cannot yet be produced at a lower cost than that of the aluminum required, and it now sells at about 80 cts. per pound. In the oxide from which it was made, it may be had for less than half this cost, and in its alloys with iron, which are made by direct reduction with carbon; it is sold for 29 cts. per pound. This gives a rough idea that ultimately, by perfection of metallurgical processes, etc., we may possibly obtain the metal much below 80 cts. per pound. It withstands heat exceedingly well. When pure it melts at very high temperature (Ostwald, about 3000° C.) and it does not scale when heated red hot in air, as copper and iron do. It is for this reason that it is used in resistance alloys for electric heating devices. It has been plated onto metals, and then looks and acts like nickel plate. Doubtless its use will quite rapidly increase in special alloys, as it has already come into use in tool steel.

POTASH SALTS SUPPLIES.

The new organization of the German Kali-Syndikat has been formally incorporated. It is stipulated that the companies holding American contracts can come in provided they terminate those contracts at the end of a year. If they decline to do so, a new syndicate will be formed.

The London correspondent of the *American Fertilizer* writes: "Until lately it has been believed that Germany was the only country in the world in which deposits of potassium salts existed. Now, however, the news of the discovery of potassium deposits in other countries is announced in the German press. According to the *Brunswick Landes-Zeitung*, the latest German discoveries are in Alsace, close to the French boundary, and the potassium deposits there will probably extend across into France, as has been the case with the salt deposits. It has also been established that deposits occur in Hungary, Russia, Holland and Persia. It is further announced that potassium salts have been found in China. Although the fact has been known for some time, it has been kept a secret, and the whereabouts of the deposits has not been disclosed. Samples of these Chinese salts have been tested by a chemist connected with the Potassium Syndicate, and the results have shown a large percentage of chloride of potassium."

TRANSFORMATION OF OTHER FORMS OF CARBON INTO GRAPHITE.*

By W. C. ARSEM.†

INTRODUCTION.

The conditions under which "Amorphous Carbon" is transformed into graphite have been the subject of much discussion, and many statements have found their way into the literature which are not supported by experimental evidence. The following theories are commonly held:

1. A high temperature alone will convert "amorphous carbon" into graphite (Moissan).
2. Pure carbon is not converted to graphite by heat alone (Berthelot).
3. Graphite is the result of intermediate formation and decomposition of carbides, due to the presence of mineral matter as an original constituent, or purposely added (Acheson).

I have collected all available references relating to this problem, and have given briefly the evidence in favor of each theory.

Despretz¹ heated several varieties of carbon in an arc. He states that "Any carbon which is submitted for a long time to a high temperature becomes proportionately softer. Finally it is transformed into graphite." No physical or chemical tests for graphite are described.

Berthelot² was first to recognize the ambiguity in the use of the term "graphite," and proposed the Brodie test as a criterion.

He was not able to convert amorphous carbon to graphite at a *white heat* in hydrogen. A retort carbon rod which had been ignited, thrust into a stream of oxygen, and quenched in water when fully incandescent, was found to have been graphitized on the tip.

*Work of Moissan.*³—Moissan heated in the arc furnace different varieties of carbon enclosed in carbon crucibles, and, after heating, determined their specific gravity and their behavior toward nitric acid and potassium chlorate. He found that the products differed in specific gravity, but in all cases they yielded yellow graphitic acid with the oxidizing mixture. He concluded that heat alone will convert amorphous carbon into graphite. His results are given in Table I.

TABLE I.

Carbon.	Ash after firing.	Specific gravity after firing.
Sugar carbon.....	0.11%	2.19%
Carbon from Al ₂ Ca.....		2.11
Cryst. graphite from cast iron, 1150°.....	1.30	2.17
Cryst. graphite from cast iron, high temperature.....	0.17	2.18
Cryst. graphite from cast iron cooled in water.....	1.29	2.16
Graphite from cast iron by silicon.....	0.85	2.20
Graphite from platinum.....	1.10	2.06-2.18

He showed also that graphite separates during the

*Paper presented at the Twentieth General Meeting of the American Electrochemical Society, in Toronto, Canada, Sept. 21-23, 1911.

†Research Laboratory, General Electric Co., Schenectady, N. Y.

¹Comptes Rendus XXIX, 709-724.

²Ann. Chim. Phys. XIX, Series 3, 392.

³Le Four Electrique.

cooling of a saturated solution of carbon in various metals, and that graphitic carbon is a product of the decomposition of many carbides, but gives no specific gravity determinations except for Fe and Al.

Work of Acheson.—Acheson noted the formation of graphite by the decomposition of silicon carbide at high temperatures. He also noted that the furnace core of bituminous coal coke in the carborundum furnace became converted, to a considerable extent, into graphite. The following statements, made by Acheson, are to be found in the literature and in patents:

(a) The amount of graphite produced by highly heating pure carbon is insignificant and impracticable, but if the carbon is mixed with considerable mineral matter, the yield of graphite is enormously increased. (U. S. Patent 568,323, September 29, 1896.)

(b) A mixture of 97 parts coke, or charcoal, and 3 per cent iron oxide, can be changed to a greater, or less, extent into graphite by varying the time and temperature of heating. As the iron is insufficient to convert all the carbon to carbide, it is assumed to have a catalytic effect, first forming a carbide which decomposes, yielding graphite and setting free iron, which again forms carbide, and the cycle is repeated. (U. S. Patent 617,979, January 17, 1899.)

(c) A natural variety of carbon containing mineral matter, such as anthracite coal with 5.783 per cent ash, gives practically pure graphite with .033 per cent ash. This result is assumed to be due to the intimate admixture of "inherent impurities," it being stated that even with the best artificial mixing and distribution of carbon and mineral matter, the conversion to graphite will be more or less irregular and incomplete. (U. S. Patent 645,285, March 13, 1900.)

(d) Petroleum coke, to which has been added 5 per cent iron in the form of oxide, is heated to the vaporizing point of iron in a specially constructed electric furnace. The iron vapor is stated to cause conversion of the petroleum coke to graphite. (U. S. Patent 711,031, October 14, 1902.)

(e) Acheson also makes the following statements in an article presented before the Franklin Institute.⁴

⁴J. Frank Inst., 147, 475, 486.

1. Comparatively pure petroleum coke produces practically no graphite.

2. Impure bituminous coal coke produces large quantities.

3. The larger the known percentage of impurities in the bituminous coal coke, the greater the amount produced.

4. Only a part of the core (in the carborundum furnace, bituminous coal coke) is converted into graphite, this not being increased even by repeated use of the same grains in successive carborundum furnaces. He concludes: "The amount of graphite produced in the core of the carborundum furnace, and also in graphite articles I have made, is much too great to be accounted for by the theory that it is formed by the dissolution of the fixed carbides formed

by the contained impurities, and carbon sufficient to satisfy the chemical formula. The most probable and satisfactory explanation is that a catalytic action occurs—a progressive formation and dissolution of carbides. The temperature being much above the point of volatilization of silica and all other possible impurities, a rapid dissipation of the active agents takes place, and it is completed in this case before the conversion of all the amorphous carbon can occur."

No experimental evidence is given by Acheson to show that pure carbon is not graphitized by simple heating. Direct proof is also lacking to show that a small amount of mineral matter can cause, or facilitate, the conversion of a large amount of carbon to graphite.

The only experiment bearing on this point that I could find described in the literature was in an article by F. A. J. FitzGerald,⁵ from which I quote as follows:

"Two carbon rods, one composed of very pure *lamp black carbon*, and containing less than 0.2 per cent ash, the other made of *petroleum coke carbon* which had been intimately mixed with a certain amount of ferric oxide, were heated side by side in an electric furnace. At the end of the experiment, the rod that had contained the iron was found to be graphitic; could be easily cut with a knife, took a beautiful metallic luster on rubbing, and would mark paper like an ordinary pencil.

"The pure carbon rod showed little change, was dull black in color, nearly as hard as before heating, and would not leave a mark on paper. One end of this carbon rod, however, was clearly graphite, from the fact that it had been exposed to the action of carbide-forming elements. These vapors had even penetrated to a certain depth in the carbon rod, and in so far as this had occurred the rod showed a brilliant graphitic appearance, was soft, etc."

This experiment would seem to be inconclusive, because two different varieties of carbon were employed, one being heated with iron oxide and the other without. It should have been determined whether or not pure petroleum coke is graphitized when heated without admixture with iron oxide.

Borchers⁶ claims that a small amount of Al or Al₂O₃ can convert a considerable quantity of amorphous carbon to graphite.

He does not state what kind of amorphous carbon is used, or what happens if this carbon be heated without addition of mineral matter, but refers to work by Borchers and Mögenburg ("Graphit aus Amorpher Kohler in Borchers's Institut für Metallhüttenwesen und Elektrometallurgie"), which was not available.

Borchers claims priority for the theory that graphite is produced from amorphous carbon by the formation and decomposition of carbides, on the basis of

⁵J. Frank. Inst., 154., 338.

⁶Elektrometallurgie (3d. ed.), 564 (1909).

an article in Zeit. f. Elektrochem, 3, 394 (March 20, 1897), from which he quotes: "Metals which form carbides, alloys or more or less dissociable compounds can assist the crystallization of carbon." The article mentioned was a discussion of methods for the production of the diamond, no reference being made to graphite in this connection.

Moreover, Moissan⁷ had previously prepared graphite by crystallization of carbon from solution in metals and by decomposition of many carbides.

Other patents relating to the manufacture of graphite are:

E. G. Acheson, 542,982, July 23, 1895. Purifies carbon by volatilizing impurities in an electric furnace.

H. Y. Castner, 572,472, December 1, 1896. Heats rods of retort carbon or other carbon by current. Product has lower density and greater conductivity.

Herbert H. Wing, 598,549, February 8, 1898. Amorphous carbon converted into graphite by prolonged heating at high temperature. (Ordinary coke mentioned.)

DEFINITION OF GRAPHITE.

Much confusion has arisen from the uncertainty as to the exact significance of the term "graphite." Many different varieties of graphite have been described, and widely differing values for the various physical constants have been published.

Berthelot was the first to adopt the Brodie⁸ test as a criterion. He defined graphite as any variety of carbon which yields graphitic oxide when oxidized with nitric acid and potassium chlorate. Moissan seems to have felt the need of a further criterion, as he defined graphite as a variety of carbon, usually crystalline, having a specific gravity about 2.2, which yields graphitic acid when oxidized.

Some of the uncertainty has been removed by the recent work of Le Chatelier and Wologdine.⁹ They showed that a number of natural graphites, when freed from mineral matter and air, had practically the same specific gravity, 2.255, this value being also obtained for a sample of Acheson graphite.

Charpy¹⁰ considers the density a much better criterion than the Brodie test.

My own work indicates that any variety of carbon, after heating to 3,000°, will give a green or yellow oxidation product by the Brodie test, although many samples have a relatively low specific gravity and lack the physical characteristics of graphite. There is no proof that the yellow oxidation product has the same composition in every case. It may be that a variety of structurally related oxidation-products may exist.

It was found, however, on examination of a large number of carbon samples, that the graphitic properties become more pronounced as the specific gravity approaches 2.26. We have then either a series of carbons of varying molecular complexity, the end mem-

ber being graphite, or a series of mixtures of graphite with other forms of carbon. In either case it seems to me that there can be little objection to the following definition:

Graphite is that allotropic form of carbon having a specific gravity of 2.25 to 2.26.

Those varieties of carbon which have some of the physical properties of graphite, such as color, softness and streak, but a lower specific gravity, may perhaps be regarded as impure graphites; that is to say, mixtures of graphite with other forms of carbon.

PURPOSE OF THE INVESTIGATION.

The points at issue are the following:

1. Can a pure form of carbon be transformed into graphite by simply heating to a high temperature?

2. If this is not the case, is it possible to cause this transformation to occur by heating the carbon, well mixed with a quantity of mineral matter, *insufficient to form carbides* with all the carbon present.

There are two ways of attacking the problem.

(a) To determine the effect of heating various forms of pure carbon alone and with small amounts of added mineral matter.

(b) To remove the mineral matter from certain varieties of impure carbon, which ordinarily change to graphite when heated, and see if the change will still occur.

Both methods were tried.

EXPERIMENTAL METHODS.

All samples were ground to fine powder before firing, to facilitate the determination of specific gravity, and all samples were tested, after firing, by treatment with potassium chlorate and nitric acid.

Grinding.—To insure freedom from enclosed air which would vitiate the specific gravity results, all samples of carbon were ground in a roller-mill. This is a steel cylinder with a tightly fitting cover on each end, and contains four steel rolls. All the surfaces are case-hardened and polished, to insure a minimum of wear. As to the extent to which iron is introduced, it is found that after 50 hours grinding of a 20-gram sample of petroleum coke, the ash had increased only 0.09 per cent. The mill is rotated at 200 r. p. m., and from two to eight hours is sufficient to grind most substances, so that the largest particles are not over .005 mm. in diameter.

Firing.—The samples were packed in small Acheson graphite crucibles with tightly fitting covers. These, in turn, were placed in larger tubular graphite crucibles, closed at both ends, the space between the smaller and larger crucibles being packed with Acheson granular graphite, previously fired and containing only mere traces of ash. The samples thus protected were fired at approximately 3,000 to 3,300° C., in an experimental tube furnace for 15 minutes. This furnace consisted of a heater tube of Acheson graphite surrounded by an insulated carbon tube with an annular free space between them. The outer carbon tube, is well packed in granular petroleum coke, but no packing material is

⁷Ann. Phys. Chem., (7), 8, 463 (1896).

⁸B. C. Brodie, Liebig's Annalen, 111, 7, (1860).

⁹Comptes Rendus, 146, 49 (1908).

¹⁰Comptes Rendus, 118, 320 (1909).

in contact with the heater. With the exception of the graphite heater tube, which was renewed for each run, the furnace has been used for a great many runs and is practically free from metallic impurities.

When making tests of purified and unpurified samples of the same kind of carbon, the purified samples were always fired in a tube which had not previously been used for firing impure carbon.

Specific Gravity.—Specific gravity determinations were made on samples pulverized as above stated, using a pyknometer of the form shown in Fig. 1, holding about 7 cc. The sample was weighed in the pyknometer, which was then half filled with Kahlbaum's C. P. benzene, and heated to the boiling point of the benzene to remove air and fill the pores of the carbon. Owing to the fineness of the powder, this can be accomplished in a few minutes. The pyknometer was then cooled to 20° C., filled to the mark with benzene, and weighed. Results are referred to water at 4° C. The limit of accuracy is about .005.

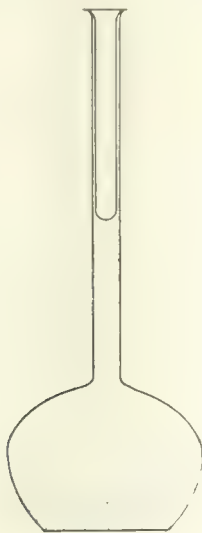


Fig. 1.

Brodie's Test.—All samples before and after firing were tested by Brodie's method, which consists in heating the sample with fuming nitric acid and potassium chlorate. "Amorphous Carbon" is converted into brown soluble substances, while graphite is changed to green graphitic oxide, or yellow graphitic acid, according to the concentration of the nitric acid, or the duration of the treatment.

In my experiments I used in some cases nitric acid, distilled from a mixture of concentrated nitric and sulphuric acids. In other cases the nitric acid was made by distilling well dried potassium nitrate with concentrated sulphuric acid, as recommended by Moissan.

EXPERIMENTAL RESULTS.

Petroleum Coke.—This was obtained from the Standard Oil Company in lump form, and contained 0.10 per cent ash. These lumps, when fired, had all the physical properties of graphite, being soft, and of a silver gray color. They marked paper easily. A sample of this graphite, pulverized before firing, had a specific gravity of 2.26. This value was checked a

number of times. A sample of the coke ground with 5 per cent ferric oxide had a specific gravity after firing, of only 2.22.

It is thus seen that quite pure petroleum coke, when fired without additions, yields a very good grade of graphite and that the addition of iron oxide is, if anything, disadvantageous.

Bituminous Coal Coke.—This was ordinary foundry coke, containing about 10 per cent of red ash, mostly iron and silica.

Lumps of this coke, when fired, became bright gray, and gave the characteristic gray streak on paper. This material seemed harder than petroleum coke graphite. Some of the powdered coke, when fired, had a specific gravity of 2.192.

A sample was treated with fused sodium hydroxide, then with water, and finally with hydrochloric acid. The ash was now reduced to 5.70. After firing, its specific gravity was 2.225.

The results of these experiments are given in Table II.

TABLE II.

	Ash before firing.	Ash after firing.	Specific gravity after firing.
Crude coke.....	10.00	2.07	2.192
Treated with fused NaOH, then with HCl.....	5.70	0.78	2.225

Here we see again the *smaller* the amount of mineral matter present, the *higher* the specific gravity reached on firing.

Anthracite Coal.—A sample of Lackawanna coal, containing 6.46 per cent ash when fired in lump form came out bright gray, rather hard, and with a tendency to cleave into plates, and angular fragments, like the unfired coal. It marked paper with some difficulty, and its feel and appearance were much less graphitic than in the case of the fired foundry coke and petroleum coke.

A pulverized sample, when fired, had a specific gravity of 2.133.

Some of the pulverized coal, when treated with fused NaOH then washed and treated with HCl, had its ash reduced to 0.20 per cent, and this purified coal, when fired had a specific gravity of 2.149, a value slightly higher than that given by the crude coal.

To check this result, some samples of coal obtained from the U. S. Geological Survey were treated in the same way. The results were even more striking, as may be seen from Table III.

TABLE III.—RESULTS OF FIRING ANTHRACITE COAL ABOVE 3,000°.

Coal samples.	Ash before firing.	Ash after firing.	Specific gravity after firing.
LackawannaCrude ...	6.46	..	2.133
LackawannaPurified .	0.20	..	2.149
U. S. G. S. "Buck- wheat No. 1".....Crude ...	17.68	..	2.125
U. S. G. S. "Buck- wheat No. 1".....Purified .	1.73	..	2.170
U. S. G. S. "Buck- wheat No. 5".....Crude ...	13.30	..	2.138
U. S. G. S. "Buck- wheat No. 5".....Purified .	0.93	..	2.180

Anthracite coal, therefore, is a form of carbon which graphitizes only imperfectly, in spite of the high per-

centage of ash which is well distributed throughout the material. We have here, therefore, excellent conditions for catalytic action without a corresponding effect. The results show that the presence of ash *hinders*, rather than *assists* graphitization. Some samples of coal seem to yield purer graphite than others, especially after the ash has been removed.

Lampblack.—A commercial variety of lampblack, known as "Patton's Sunproof," containing about 0.2 per cent ash, principally iron oxide, when fired in the tube furnace reached a specific gravity of 2.090. Some of the lampblack, unfired, was ground with 5 per cent ferric oxide and fired. This had a specific gravity of 2.094. Another sample of lampblack was impregnated with ferric oxide, so as to obtain as good a distribution as possible in the following way:

The lampblack was stirred into a solution of ferric chloride and ammonia was added to precipitate ferric oxide, when the lampblack precipitated with the ferric oxide, forming an apparent homogeneous mixture. This mixture was well dried and divided into two portions; the first was heated one hour in vacuo at 1,600°, and the other one hour at 2,000°, to reduce the iron oxide to metal, and then both were fired in the graphite tube furnace at about 3,300°. After firing, the specific gravity of the first portion was 2.122, and of the second portion 2.109. All these fired samples were indistinguishable from each other by physical or chemical tests. We have here a pure form of carbon, which reaches a limiting density of about 2.10, and this limit is not appreciably raised by the presence of even 5 per cent ferric oxide, although the distribution of the latter was about as perfect as possible and the conditions especially favorable for a catalytic transformation into a more graphitic variety of carbon if this were possible. They are all grayish-black non-crystalline powders. They all yield by Brodie's test, a yellow graphitic acid, but have none of the characteristic properties of graphite.

Another sample of lampblack, Reichard's No. 7, was also fired, and reached a density of 2.074.

Patton's sun-proof lampblack was also heated with small percentages of different substances, but no catalytic effect was noticeable.

The results are summarized in the table.

	Density.
Patton's lampblack 0.2% ash.....	2.090
Patton's lampblack + 5% Fe ₂ O ₃ ground together.....	2.094
Patton's lampblack + 5% Fe ₂ O ₃ by pp n. (a).....	2.122
Patton's lampblack + 5% Fe ₂ O ₃ by pp n. (b).....	2.109
Patton's lampblack + 5% Al ₂ O ₃	2.080
Patton's lampblack + 1% Si.....	2.076
Patton's lampblack + 5% MnO ₂	2.091
Patton's lampblack + 5% NiO.....	2.099
Reichard's lampblack.....	2.074

Retort Carbon.—The carbon obtained from the inside walls of gas retorts occurs in three varieties, black, gray and white. These differ in ash content and in behavior on heating, and show in a striking way that the ability to graphitize is independent of the amount of ash present. Both the white and the gray varieties yield excellent graphite, although the former is very low in ash and the latter high.

The black variety which is also in ash does not graphitize but reaches a density of 2.11. Four different substances tried as catalyzers had no effect, the density after firing being practically the same as that of the same carbon fired without mineral additions.

TABLE IV.—EFFECT OF HEATING RETORT CARBON ABOVE 3,000°.

	Ash. before firing.	Ash after firing.	Sp. grav. after firing.
White retort carbon.....	0.34	0.10	2.265
Gray retort carbon.....	2.66	0.16	2.263
Black retort carbon.....	0.23	0.10	2.110
Black retort carbon + 1% CaO.....	..	0.31	2.123
Black retort carbon + 1% SiO ₂	..	0.12	2.117
Black retort carbon + 1% Li ₂ CO ₃	..	0.05	2.120
Black retort carbon + 2% Fe ₂ O ₃	..	0.09	2.178

Diamond.—Some white diamonds when fired became converted into a coke-like carbon, having a specific gravity of 1.915, which does not mark paper.

Parsons & Swinton¹¹ heated a diamond in the focus of a cathode ray tube at 1890°, and state that it was converted into coke. The specific gravity was not stated.

These results are especially interesting since it has been commonly supposed that the diamond is converted into graphite by heat, and statements to that effect appear in most text-books.

CONCLUSIONS.

All the pure forms of carbon which have been tested, when fired above 3000°, reach a limiting density which is not appreciably raised by the addition of small amounts of mineral matter. The end product is graphite in some cases and not in others. Pure petroleum coke, heated without addition of mineral matter, is converted into graphite of excellent quality, while lampblack, although it increases in density, does not reach the value corresponding to graphite, nor acquire any of its other physical properties, even when heated with various oxides.

The impure carbons show a similar behavior. The properties of these carbons after firing are characteristic for each variety of carbon and independent of the amount of ash present.

Anthracite coal is only imperfectly graphitized by heating. The specific gravity of the fired material was approximately the same for three samples having a range of ash content. Moreover, coal from which most of the ash has been previously removed graphitizes better than the crude material.

Bituminous coal coke, which graphitizes fairly well, yields an even better product if a part of its ash has been removed before firing.

It must therefore be concluded that a small amount of mineral matter exercises no beneficial effect in the manufacture of graphite by the heating of carbon, and that the quality of the product cannot be improved in this way. As to the effect of mineral matter on the rate of conversion, no data are yet available, although some experimental work along this line has been started. At 3000° the maximum density is reached in less than 15 minutes for any variety of carbon.

¹¹Proc. Roy. Soc. 80, 184 (1908).

We need some theory of the nature of graphite and of amorphous carbon which will permit a rational explanation of the changes which occur on heating. The following discussion is a tentative step in this direction.

THE NATURE OF GRAPHITE.

Graphite, in the most restricted sense of the term, is an allotropic form of carbon having a definite and not very complex molecular configuration. The molecule might, for example, be regarded as two benzene rings side by side and joined at all six angles, the extra bonds being satisfied as in the usual centric formula. Such a formula might be used to explain the formation of Mellitic acid and graphitic acid by oxidation of graphite.

Amorphous Carbon.—When an organic compound is decomposed, there results a mixture of substances constantly increasing in complexity until finally carbon is obtained. This carbon need not be regarded as a simple substance, but may be considered to be a mixture of many varieties of carbon, each with a different number and arrangement of atoms in the molecule. According to this view, it seems better not to regard "Amorphous Carbon" as the name of a distinct allotropic form, but rather as a general term covering all varieties of carbon except graphite and diamond. In speaking of amorphous carbon, therefore, the source of each sample should be stated.

The Mode of Transformation.—In a given sample of amorphous carbon, some of the molecules will be capable of easily undergoing rearrangement under the influence of heat to form graphite molecules, while others will not, and the proportion of molecules capable of such change will determine the character of the final product.

Petroleum coke would therefore consist almost wholly of graphitizable molecules, while anthracite coal contains a smaller proportion.

THE GERMAN POTASH INDUSTRY.

The United States consul at Brunswick, Germany, reports that at the conference held recently at Hamburg between the conflicting interests, German and American, in the potassium salts industry, an agreement was reached as to prices and rebates.

The agreement has a duration of $5\frac{1}{2}$ years. During this period the American producers of potash, that is to say, the North trust, the South trust, and the independents, obligate themselves to make all purchases from the potassium syndicate. The probability of the market being disturbed by outside mines is thought to be remote, as such outsiders would have to organize a sales agency in the United States similar to the German Kali works owned by the syndicate, which delivers salts directly to the buyer. Such an organization would be expensive and difficult to effect.

The basis of prices is practically the same for concentrated salts as that contained in the potassium law,

and the same as prevailed in America in 1909 before the syndicate was renewed and before the Schmidt-mann contracts went into force. The prices, aside from 40 per cent manure salts, which are sold to German agriculturists at low prices, are the same as the maximum prices laid down in the law for the domestic consumption. On the other hand the prices proposed for raw salts, especially for kainit, are higher than the prices formerly paid by Americans to the syndicates. Rebates for most sorts of salts are increased. Formerly a rebate of 9 per cent on the average was granted the American companies. Now it will be from 11 to 12 per cent.

In reference to certain expenses which may be incurred by the syndicate, the right is granted it during the duration of the agreement, five and one-half years, to increase prices 3 per cent.

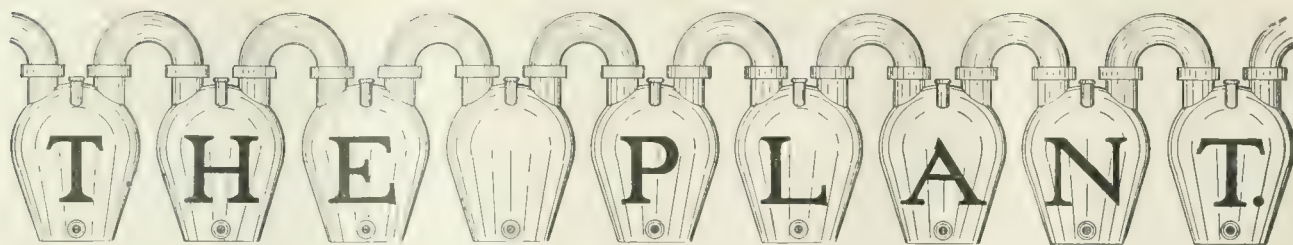
The agreement is incomplete, inasmuch as the Aschersleben and Sollstedt works, belonging to the Schmidtman interests and the Schmidtman International trust, have not united in it. An effort will be made by amicable means to bring the remaining parties into the agreement. If it is not successful, it is thought that their sales contracts will be attacked, as they are regarded as in some respects legally indefensible.

At a recent general meeting of stockholders of an important mining company held previous to the aforementioned conference at Hamburg, the president stated that the favorable result of the business year, namely, the distribution of the dividend of 15 per cent, had been attained in spite of the potassium law.

"The chief purposes of the law," he continued, "the prevention of the precipitate founding of new works and the rendering ineffective of the alleged ruinous American contracts, have not been accomplished. In both of these directions the law has proven abortive. Directly contrary to the law a much more active founding of new companies has taken place and the American (Schmidtman) contracts are in full force."

This opinion seems to be shared by persons in Brunswick acquainted with the condition of the potassium industry; it is said that the industry has been sustained by the steady increase of sales in old markets and the introduction of the salts into new markets.

According to figures of the U. S. Geological Survey the production of crude barytes in the United States in 1910 amounted to 42,975 short tons, compared with 61,945 in 1909. The average price per ton as reported to the Survey was \$2.83 last year, as compared with \$3.39 in 1909, representing the amount paid the miner for his crude ore, hand-cobbed, sorted, and ready for shipment to the mills, exclusive of transportation costs. The wholesale prices per ton for refined barytes at the close of 1910 were \$12 to \$15 for "American ground," \$17 to \$19 for "floated" and \$20 to \$23 for "foreign floated."



REFINING STEEL FROM BESSEMER CONVERTERS BY A COMPOSITE ELECTRODE ARC PROCESS.

By FRANK C. PERKINS.

There is a vast amount of available iron ore in the United States which is high in phosphorus and sulphur and titanium; there is also a large supply of carbonate iron ores.

The electric smelting process has been shown to be entirely successful in eliminating both phosphorus and sulphur from steel, the percentage remaining in the ingots depending entirely upon the time and amount of electric current used in refining.

The cost of refining steel, as it comes from a Bessemer converter by the electric process depends upon the cost of the electric energy which is extremely low in places where generated by hydro-electric power plants.

The electric arc refining process may be considered commercially successful for making high grade tool steel where the molten steel is electrically treated as it comes from the converter and it is generally considered that there is every reason to believe that the process can be developed for ordinary grades of steels so that it may be used for producing steel rails and structural steel with phosphorus at .05 or lower without great expense or interfering with the present rate of production in existing rolling mills. If this can be accomplished it will place the present Bessemer plants on an equal or possibly a more advantageous footing than the large new open hearth installations.

A new composite electrode arc process for producing and refining steel has recently been patented in the United States and the construction and arrangement of electric ladles or furnaces provided with those special composite electrodes may be seen in Figs. 1 and 8. This process consists in treating molten iron from a blast furnace or molten steel, *a*, from a Bessemer converter or open-hearth furnace, with electric heat produced by arcs formed between the slag, *d*, and specially designed electrodes, *D*, *D*.

Several forms of composite electrodes are indicated in Figs. 3 to 7. It will be noted that one consists of an iron rod *E*, a cored carbon *F* and a mixture of carbon, lime or oxide of iron or other slag producing materials *G*, surrounding same. Another shows the slag material, *g*, Fig. 8, packed in an iron or steel tube, *e*, and still another surrounding a carbon or iron rod or rods, *F*, with or without projections for support-

ing the slag materials with the usual binder employed in carbon electrodes.

The use of the composite or combination electrode, instead of the ordinary carbon electrode, or an electrode of pure iron, introduces a large part of the slag material in a highly fluid state at the hottest points

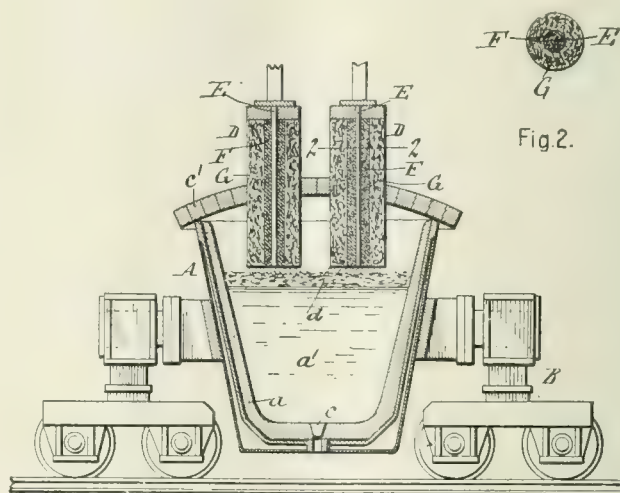


Fig. 1—Electric Furnace of Double Pole Type.

which are at the arcs or at the points of contact of the electrodes and the layer of slag floating on the bath of molten steel, the remainder being added in the usual manner.

When two of these electrodes, *D*, *D*, are used in an electric furnace of the double pole type as in Fig. 1, the current passes from one electrode, *D*, into the slag, *d*, through the slag as a resistance and out of the other electrode, *D*, the two arcs operating at about 100 volts pressure. One electrode may extend into the slag or down into the molten steel below the slag, a single arc only being employed at this time formed at the other electrode, with a pressure of approximately 50 volts, the one moving up and down for agitating the metal if a large ladle of 10 tons capacity or more is used requiring other movements of the metal that naturally provided by the action of the arc.

The double pole electric furnace is well adapted for practising this composite electrode process similar to the Heroult method because when the two electrodes are introduced into the refining slag, the latter acts as the resistance and forms one or two arcs according to whether one or both electrodes are out of contact with the bath and producing the arc or arcs.

An ordinary basic lined ladle, crucible or pot may be used as noted in the drawings without cover which

may be supplied in the same manner as a basic lined two pole electric furnace. This ladle having received the charge of molten steel from a Bessemer converter is electrically refined by eliminating the phosphorus and sulphur, as far as desired and the steel deoxidized and recarbonized. The steel is then poured into the ingot moulds after an electric treatment of from a

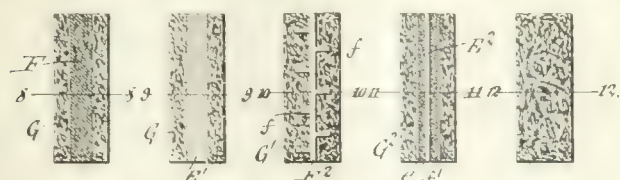


Fig. 3. Fig. 4. Fig. 5. Fig. 6. Fig. 7.

few minutes to an hour or more in the electric ladle according to the degree of refining desired.

When only one of these electrodes is used in an electric furnace, this electrode extends into the refining slag forming the arc at this point the current being conducted through the molten bath and out through the bottom of the ladle similar to the Girod process.

Another construction shows double arc or single arc action taking place when a ring or cylindrical electrode is arranged on a level with the slag. A single arc action only at center may be employed, the outer electrode dipping into the slag or molten steel.

By this electric arc process the molten metal continually circulates, all particles of the bath coming into contact with the refining slag at the arc or arcs and elsewhere and being rapidly refined, remaining at the highest temperature only a short time, then replaced by other particles of molten steel reaching the slag near the arcs.

It is held that in the Heroult arc steel furnace the circulation in the bath is always active and the average temperature may be kept as low as any other furnace and all parts of the bath come rapidly into contact with the slag. In case a deep bath of say 10 or 12 tons or more is used in the electric ladle and additional circulation is desired; one electrode may be plunged up and down in the metal by an electric hoist mechanism while the other electrode produces the single arc then used with the slag.

By the use of these special electrodes made with slag producing materials, the highly fluid slag at the arcs causes rapid circulation to take place and the gases are removed which are often retained in the metal and cause bad steel when poured into the ingot moulds directly from the Bessemer converter without electric refining.

Furthermore, in this electric refining process segregation of sulphur and phosphorus are largely avoided while any quality of steel may be produced regardless of the quality of the raw materials.

It is well known that the Bessemer converter process for making steel is far cheaper than the open-hearth process, but the quality very largely depends

on the metal, ore and other material available, and it is not possible to test the metal during the operation. The resulting steel therefore from high phosphorus and high sulphur charges contains an injurious quantity of sulphur and phosphorus which cannot be eliminated by the existing acid Bessemer process. Too much or not enough carbon, manganese, silicon or other elements can readily be corrected and gauged by the charge introduced into the converter and subsequent treatment but phosphorus and sulphur less than .09 and preferably below .05 is desired and this is not attainable by the existing present Bessemer process.

With these special electrodes and the use of an electric arc for refining purposes, materials can be used in the electrodes or added to the bath for a neutral slag, as it is maintained that a thorough deoxidization of the steel is not possible where there are iron oxides in the slag as they will react to a certain extent with the molten bath of steel.

It is held that adding carbon under ordinary conditions will not result in complete deoxidization as

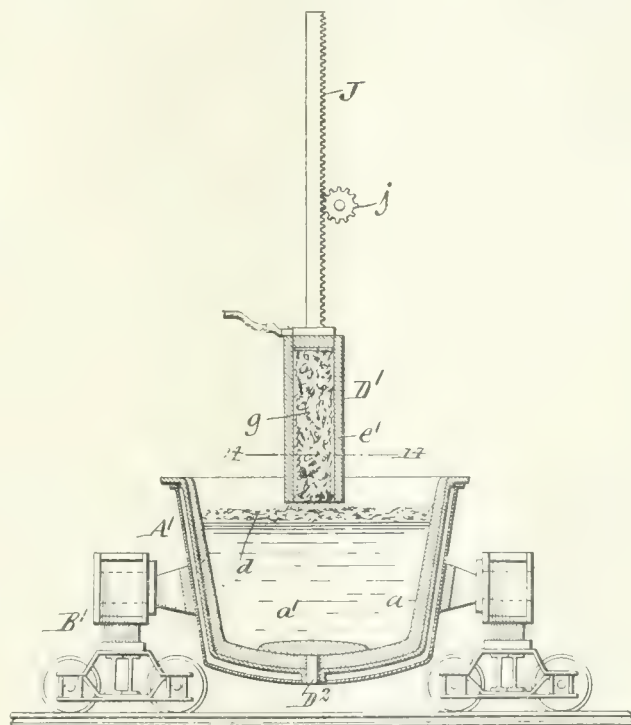


Fig. 8—Electric Furnace Provided with Composite Electrodes.

both iron carbide and ferrous oxide readily exist together. By means of the electric process of steel refining with these combination or composite electrodes, however, carbon or a mixture of carbon and iron may be added to the slag as an auxiliary to the carbon and iron may be added to the slag as an auxiliary to the carbon mixture of the electrode, forming calcium carbide and resulting in deoxidation without the slightest difficulty at no great expense, and on any scale desired by providing electric arc apparatus for several electric furnace ladle cars arranged for serving various Bessemer converters or open hearth furnaces.

The desired amount of manganese can be added for counteracting the bad effects from the ferrous oxide, the carbon reducing the manganese ore which has been added to the slag and taking out the last traces of ferrous oxide leaving the steel bath protected by the layer of slag from oxygen in the air so that no further oxidization takes place.

The determining questions of the electric refining of steel are cost and output. By shortening the time or treatment with these special composite electrodes combined with slag producing mixtures, and refining only down to such percentage of phosphorus and sulphur as are essential for rails and structural steel, the electric power consumed is so low and the time required so short as to be practical for this work.

By this process cheap ore can be used having higher phosphorus and sulphur, these ores being abundant while high grade ores are nearly exhausted.

This new composite electrode process may be utilized as an auxiliary treatment to the Bessemer process for eliminating the phosphorus by employing an oxidizing slag in the bath as the molten metal is treated in the electric ladle with the arc, the auxiliary slag producing materials in the combination or composite electrode acting instantly on the metal and the slag being in a highly fluid state as it melts in the high temperature of the arc.

The highly oxidized metal from a Bessemer converter overblown may be utilized deoxidizing same in the electric ladle by this process. It may be stated that the desired end to be attained by this electric auxiliary process with the Bessemer converter for rail making particularly, is the reduction of phosphorus to .05 or thereabouts by the oxidizing slag and electric arc, the other elements being easily controlled by existing Bessemer methods.

It is well known that with the Heroult electric furnace having pure carbon electrodes and a treatment extending from 75 to 90 minutes or more taking molten metal from an open-hearth furnace and electrically treating same with oxidizing and neutral slags the phosphorus can be brought down to .003 per cent and the sulphur below .007 per cent according to time of treatment.

By the use of the composite electrodes containing slag producing mixtures it is possible to reduce the phosphorus to .03 or .05, which is 10 to 20 times as much as the above in a far shorter time and within practical limits for Bessemer working only a small part of the phosphorus being removed.

It is also well known that the carbon is always eliminated before the phosphorus and if it is attempted to carry the reduction far enough to lower the phosphorus, in an open-hearth furnace, the metal is highly oxidized and decarburized.

This composite electrode process provides a means of taking the highly oxidized metal from the open-hearth furnace, if carried far enough to eliminate as much phosphorus as desired and by electric treatment with the arcs and these special electrodes having

neutral slag mixtures it is also possible to eliminate, if desired, nearly the last traces of sulphur and metal being deoxidized and as much carbon being added as found desired, producing a finished steel of any degree of perfection desired according to the length of time the electric treatment is carried on.

It may be stated that in metal taken from an open-hearth furnace, if the reduction is carried on long enough to reduce the phosphorous to .01 per cent or less than this amount, the carbon is eliminated. It is of course true that such a highly oxidized metal would be very unsatisfactory in practice but by means of the electric furnace treatment with these composite or combination electrodes, the arc and proper slag mixtures, introduced in the molten bath, together with the proper mixture in the electrode themselves acting as auxiliary slag producers, will bring most satisfactory results, eliminating the sulphur, adding the proper amount of carbon and deoxidizing the steel completely.

The proper mixture can be introduced into the molten bath and also provided in the electrode mixture to remove the sulphur on account of the high temperature of the electric arc even though the slags selected could not be used by any other slag materials to lower the melting point and this would interfere with the efficiency of operation. The slag mixture is also melted to a highly fluid state at the arc largely from the mixture in the electrode itself, placed there for this purpose, in addition to that supplied in the ordinary way on the top of the bath.

Ordinarily there is great trouble and it is very costly to deoxidize the metal by existing processes and it is claimed that pipes and blow holes are produced in the ingot and other difficulties result from the presence of iron oxides. It is also maintained that when ferro-silicon and ferro-manganese are used to prevent these troubles, the oxides which result stay in the steel as an "Emulsion" in a finely divided state.

By this process of only taking the metal from the converter or open-hearth furnace when it is nearly finished steel and molten and merely completing the refining operation electrically only one-quarter of the electric power and less time is required than when electrically heating from cold metal. It is of value as an auxiliary process to the Bessemer converter and open-hearth furnace taking the molten metal with little loss of heat and refining same to any degree according to time taken, the mixtures of carbon and iron or steel combination electrodes used and nature of slag employed.

While this process is specially adapted to the refining of molten steel taken from the Bessemer converter and open-hearth furnaces and treated in an electric furnace or ladle, it is not confined to this as the identical process will convert iron into steel from the molten or cold state without being previously treated in a Bessemer converter. Any grade of steel may be made regardless of quality of raw materials and carbon may be eliminated from electrodes for special

steel making, the current being conducted to the arc by the iron rod instead of carbon and iron oxide, or composite electrode.

The neutral slag electrodes may work with a low current arc or dipping into slag and heating by resistance and the steel may be kept for hours under this molten neutral slag without changing its quality. The metal may be cast, remelted or worked over to a higher or lower grade or it may be cooled, chilled and melted a second time without injuring the quality of the steel.

The cost of production in this process is low as most of the work of steel making is done by the Bessemer converter or open-hearth furnace, only the removal of small percentages of the phosphorus or sulphur from the steel being attempted or the deoxidation of the metal, this being a good product for the larger rolling mills which supplies the demand of rail and structural steel interests. The partial removal of the injurious phosphorus and sulphur reduces the time to a few minutes instead of hours, hence reduces the current required per ton.

It may be stated that the shorter time also lowers cost of linings and refractory materials per ton of metal as many charges may be treated in same time required for treating one open-hearth charge by existing process with gases alone. This shorter time process only reducing phosphorus from .09 to .05 for rail steel instead of to .003, which is possible by longer treatment, makes it a practical process for rail mills dealing with Bessemer converters treating full charges of 15 tons in electric ladles, handled by cranes taking same to positions where electrodes are inserted.

After treatment the metal is drawn from the bottom of ladle or tilted and poured into the ingot moulds after slag has been removed. These simple methods are of vital importance as any new process for removing phosphorus and sulphur, should not greatly increase the cost of power; should not cause excessive cost for basic linings or electrodes, or slag producing materials and should be readily used as an auxiliary process in existing Bessemer plants.

The total value of the production of gold, silver and copper in Wyoming in 1910, according to C. W. Henderson, of the United States Geological Survey, was \$32,234 against \$69,028 in 1910. The gold yield was \$3,199 against \$4,086 in 1909; silver, 1,478 fine ounces, valued at \$798, against 1,754 ounces, valued at \$912; and copper, 222,339 lbs., valued at \$28,237, against 492,539 lbs., valued at \$64,030. The amount of ore sold or treated was 1,474 tons, against 2,393 tons in 1909. There were 12 producing mines in Wyoming in 1910, of which 4 were gold placer mines. Copper has been the most important metal produced in Wyoming, and the state is to be credited with a total output of 25,843,400 lbs. of copper from 1881 to 1910 inclusive.

MODERN CUPOLA PRACTICE.*

By JOHN C. KNOEPPPEL.

Foundry owners frequently wonder why they are having trouble with their iron, blaming almost everything but their management of the cupola. This has not received the attention that it should, and as a usual thing the cupola is considered foolproof, in fact, is expected to run itself. Good work with the cupola is first of all dependent upon a thorough knowledge of the cupola and its performance while working under the blast. The construction must next be such that the proper amount of air can actually get in, and in such a way that the temperature conditions are as uniform as possible, so that cold spots are avoided. The tuyere area must be of such a size and so arranged that the blast enters the cupola without undue friction.

Preparing for the Charge.—The ultimate shape of the cupola lining, if allowed to take its own course, will be found to be slightly built out above the tuyeres, and this is an argument that, if left that way, it is most desirable for efficient work. Hence in chipping the cupola and daubing up, it were best left so that only the refuse matter and iron above the tuyeres is taken away, but that this natural shape is left intact. The daubing can be very light, and cracks filled with small broken firebrick or flat brick, as the condition may require it. As the heat at this point becomes very intense, there is a tendency for the daubing to crack off long before melting is in progress. This material falls into the stock and retards melting afterward.

Bottom is put in so that the surface, while dipping from back and sides to the spout, does not fall too steep, as the melted material will run downward with too much force, causing trouble in tapping during the heat. It should be so made that the iron will lie quietly on it and not injure or break through it. The necessary shavings and kindling are now put in, the latter being laid by a man in such a way that when burning it will do so evenly and without injury to the bottom. Coke is now thrown in and allowed to burn through evenly. When up to proper height, as indicated by wire gauge or other suitable method, charging may begin.

Details of Charging.—Charging should commence at least 2 hours before the blast is put on, so that the stock may become well heated through, and the time for lighting up should be arranged accordingly. There should be neither too much nor too little coke on the bed. Oftentimes the cupola man is allowed to take all he wants, and the result is a prolonged heat, with dull iron instead of the expected hot material. The iron charged should be of medium size, especially in small diameter cupolas. There should not be too many openings for the free passage or rather escape of the blast. In large cupolas, while the stock can be larger, it

*Paper read before American Foundrymen's Association.

should not be charged in too compact a way, thus retarding the blast.

The amount of the bed of fuel for a cupola is determined by a number of factors: The diameter inside the lining, the height of the tuyeres, blast pressure, class of work, etc. For light work, with practically continuous melting and pouring, the tuyeres should be set low, thus saving fuel for the bed.

The bed should extend from 20 to 24 in. above the top of the tuyeres, whether one or two sets are used—above the top of the upper if the latter. On the other hand, the tuyeres must not be set too low, otherwise if any iron is to be held it will be chilled if remaining in too long. From 12 to 16 in. from the bottom plate to the bottom of the tuyeres would seem about right for ordinary practice, the bottom being from 4 to 6 in. thick.

Slagging.—Slagging when the tuyeres are very low often does more harm than good; it retards melting and is destructive to the cupola lining. If the heats are heavy, so that slagging must be resorted to, it is better to put the tuyeres up higher in the first place. The slag hole should not be opened until the slag is high enough to reach it; usually 10 tons or so of metal will have passed out by that time.

In large cupolas for medium heavy work the tuyeres are usually set from 18 to 24 in. from the bottom. The higher they are set the more fuel is required for the bed. More air must be forced through or else melting is unduly slow. High melting zones and blast pressures have a tendency to harden the iron, make more slag and give trouble generally.

The charges of iron should be uniform, the first being as large as the last. While undoubtedly more iron could be carried on the first charge, it would mean lowering the bed unduly with subsequent trouble, as the coke charge coming down will not restore the bed to its full height again. It is the best kind of practice to maintain the bed at its proper height at all times, and this can only be done by small and equal charges of metal and the proper proportion of coke between them.

In long heats of high tonnage, it is oftentimes a good practice to help build up the bed by adding additional coke at intervals. Experience will show this best. The writer has always had better results with such small charges. The melting was more uniform and more rapid, with hotter and better iron. With heavy charges the melting difficulties result in giving metal with compositions different from the expected, and the chemist get the blame, which really belongs to the foundry manager.

The Volume of Air Important.—The construction of the cupola should be such that the air entering has great volume at low pressure rather than vice versa. Good lining material should be used, preferably double up to the charging door and single from there upward. The clay-wash should be mixed a day previous to use, and the addition of a little salt helps to tighten the

joints. Space is left between the lining and the shell—about half an inch—and this filled up with a grout of old firebrick, ground up, and clay. The lining at the melting zone, as stated previously, is allowed to take its own shape during the heats, being originally straight like the rest of the construction. If this is done, the melting capacity will be found increased. While this is the experience of the writer, he does not wish to be understood as advocating a contracted hearth construction.

Fuel that burns up fast, such as coke, must necessarily be supplied with a sufficient amount of air in quick time. The design of the tuyeres, therefore, plays an important part in this. The writer has found that a continuous tuyere of proper design gives the most uniform distribution to all parts of the cupola furnace. The velocity of the air is reduced and the best kind of melting is obtained.

When it is considered that to melt 10 tons of iron about 300,000 cu. ft. of air must be admitted into the cupola for that hour, it will be seen that the tuyere conditions must be such that a minimum obstruction shall take place before the blast actually is doing its work in the charges.

The coke below the tuyeres serves simply to hold the iron and support the stock. The temperature is far below that of the iron, and hence every pound thus used, unless for special reasons, is a direct waste. If the iron is held in the cupola too long the blast strikes over the top and injures it. Hence, while on general principles low tuyeres are advisable, they should not be so much so that the metal is damaged when carrying out the daily practice.

The charging door should be as high as conditions will permit above the bottom, so that the fuel will receive the benefit of the heat otherwise wasted. Ten to 15 ft. is the usual custom, though the distance is often made greater. Where a very high stack and charging door is used it is sometimes advisable to put in an intermediate door, so that in charging the first part of the heat the bed is not damaged. Further, it allows a more even charging. This intermediate door should be sealed up when the charging has reached that point, and then the upper door is used. The charging floor would naturally have to be constructed accordingly.

In every cupola, under the same conditions, there is a fixed melting zone. Below and above this the metal cannot be melted successfully. In either case the metal would be dull and damaged. This zone is determined by the cupola conditions and the volume and pressure of the blast. If the iron comes down within 10 ins. of the putting on of the wind it would indicate that the bed was of proper height. If it takes longer, then the bed was too high. The excess of fuel must be burned away, and in doing this the iron is melted slowly and becomes dull instead of very hot. The melting zone is usually a space across the cupola about 4 to 8 ins. high. Where the blast is heavy this is

sometimes greater. A double set of tuyeres sends the melting zone up higher than is the case with one row.

The Fuel Ratio.—Regarding the fuel ratio, this much may be said. What is economy in one foundry becomes a waste in another. The high melting ratios published are oftentimes very misleading. For general purposes 7 to 8 lbs. of iron to 1 of fuel is very fair. With the same sized small metal charges to coke between charges generally works out at 1 to 10. This can often be cut a little on the coke end if the bed is ample, especially at the end of the heat. As this work should not be left to the tender mercies of the cupola man, it is a good plan to hang up a chart of sufficient size in the charging room on which every charge is marked plainly, so that it can be followed without difficulty. All stock should be weighed, as even the coke will not run uniform in weight if measured in a basket. This is not with the idea that the fuel should be skimmed, but simply to have the conditions in charging as perfect as possible, so that the best results may be expected and obtained.

For 20 years the writer has advocated the use of small charges and as mild a blast as possible, and he is glad to note that others are beginning to agree with him. The question of the tuyeres has been his special study, and he hopes that the system bearing his name may have helped to make the work about the cupola easier and the results surer. The cupola is not merely a vessel into which any old thing can be thrown and good results obtained, but it may become a money maker or loser just as one pleases. In these days of keen competition it becomes a problem not only to melt iron properly and economically, but to turn out a product that will give the smallest percentage of scrap castings.

The value of the mine output of gold, silver, copper, lead, and zinc in Montana in 1910, according to V. C. Heikes, of the United States Geological Survey, was \$48,358,253, against \$51,429,694 in 1909, a decrease of \$3,071,440, occasioned almost entirely by the lessened output of copper ores. The value of the gold, silver, and copper combined was \$4,326,868 less than in 1909, while the value of the lead and zinc yield was \$1,255,427 more than in 1909. The value of the production of Silver Bow County, which includes the Butte district, was 91 per cent of the total value of the metals from Montana. The decrease in value in this county was \$3,108,861, so that the combined production of the other counties in Montana was slightly greater in 1910 than in 1909. The production of gold in 1910 was only \$61,024 less than in 1909. The greater part of the gold, 115,261.06 fine ounces, or 64 per cent of the total, was from siliceous ores; and 31,441.93 ounces, or 17 per cent, came from copper ores. The placer gold in 1910 amounted to \$575,917, of which \$473,385 was obtained by dredging. The gold won from placers was \$32,545 greater than in 1909, but that taken from lode mines was \$93,569 less.

COKE OVEN GAS AS AN OPEN HEARTH FUEL.*

By ARTHUR P. SCOTT.†

Characteristic of more or less recent metallurgical literature has been an increasing interest in the disposition of by-product coke-oven gas, more especially with reference to its possibilities as a fuel in the open-hearth furnace. In this connection four articles, which seem to be of special importance, will be briefly abstracted by way of introduction to a short discussion of the subject. The first of these, "On the Present-Day Status of Basic Open Hearth Practice," which is to be found in Nos. 1 and 2 of *Stahl und Eisen* for 1910, was read by Dr. Otto Petersen before the Verein Deutscher Eisenhüttenleute on December 5, 1909, in Düsseldorf. In describing the Hubertushütte open hearth plant in Kattowitz, O. S., Dr. Petersen says in substance:

RESULTS AT HUBERTUSHÜTTE.

The steel works consist of two 20-ton and one 25-ton basic open hearth furnaces, the capacity of whose regenerators amounts to about 1.4 cu. m. per ton in the gas chambers and 1.6 cu. m. in the air chambers. The gas producer plant consists partly of old Siemens producers and partly of modern water-sealed producers. This open hearth plant has been using coke oven gas as fuel since June, 1907. The quantity of oven gas used was small at first, but was gradually increased until, taking the coal consumption at Hubertushütte for 1906 as normal under producer practice at 31.8 per cent in September, 1909, with an ingot production of 6,200 tons, the coal consumption amounted to only 14.9 per cent, so that 53.15 per cent of the total coal requirements was replaced by oven gas. The above fuel consumption includes, besides the open hearth furnaces, provisions for such allied requirements as test and hardening forges, ladle warming and a 6-ton acid open hearth in the steel foundry with its annealing and core ovens. Oven gas is in regular use at Hubertushütte, and the full complement of producers is only requisitioned to make good an occasional shortage. The life of the open hearth ends and roof has been reduced by about 8 to 10 per cent (being now 550 to 600 heats), while the life of the checkers has been increased 40 to 60 per cent (now 1,050 heats). For this reason, along with the saving in coal, unloading wages, producer labor and maintenance, this departure in open hearth practice appears to be of considerable economic importance.

EXPERIENCE AT SYDNEY, NOVA SCOTIA.

In the discussion following Dr. Petersen's address, E. von Maltitz, of Barmen, said in part: "Some years ago I had to run three of the open hearth furnaces on coke oven gas exclusively at the works of the Dominion Iron & Steel Company, at Sydney, C. B., Canada. It was not possible to preheat the oven gas in the regen-

*From *The Iron Age*.

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erators without considerable loss of CO and a corresponding increase of CO₂. We were compelled, therefore, to use the gas without regeneration and we followed the method employed in the Pittsburgh district for natural gas; that is, the gas was fed into the furnace through 6-in. pipes inserted close to the hearth on both sides of the ports. The heat obtained from this gas was so small that we were compelled to supplement it with tar from the coke oven plant. As a result of the use of tar the life of the roof was reduced to 90 heats. We were obliged after three years of fruitless effort to give up the use of coke oven gas with tar. For this reason it would interest me very much if Director Amende, of Hubertushütte, gave a few details concerning his methods, whereby not only is the time of heat cut down, but the life of the roof, considering the fuel employed, is so unusually high."

Director Amende in replying gave no information as to the method of introduction of the gas, merely adding the following facts: The coke oven plant at Hubertushütte consists of 90 Otto-Hoffman ovens, 10 m. long, 1.5 m. high and 0.55 m. wide. This plant handles 320 tons of coal in 24 hours, and the oven gas has the following composition:

	Per cent.
CO ₂	6.5
C ₂ H ₂	2.0
O	1.2
CH ₄	16.4
H	38.7
N	24.8
CO	10.4

The coal employed is of poor quality and 50 to 60 per cent or even 70 per cent of the gas is used for heating the ovens themselves. In general the excess gas amounts to 45 per cent—in all about 50,000 to 70,000 cu. m. in 24 hours. There has been no deleterious effect upon the open hearth product, which consists of shapes and bars, also sheets and band iron of the finest grade. In every respect the use of coke oven gas has proved satisfactory and he would gladly use more of it if he had it to use.

THE FIELDS FOR BLAST FURNACE AND COKE OVEN GAS.

The second article in question is a paper read before the Fifth International Congress for Mining, Metallurgy, Applied Mechanics and Practical Geology at Düsseldorf in June, 1910, by Chief Engineer Terpitz, who describes the use of oven gas as an open-hearth fuel at Hubertushütte, but adds nothing to the data given by Dr. Petersen and Director Amende. He calls attention, however, to the large additional surplus of tunnelhead gas that has become available by reason of the development of the gas engine, and emphasizes the increasing importance of studying the peculiar adaptability of blast furnace gas and coke oven gas, respectively, that each may be employed to the best economic advantage. He describes certain attempts to utilize blast furnace gas as an open hearth fuel, and, though results to date have not been definitely encouraging, he believes that in a few years blast furnace gas will be successfully used in that capacity, although it will probably have to be regenerated by

passing it through hot coke or coal. He adds, however, that in view of the ease with which low-grade dust coal may be used under boilers and of the fact that blast furnace gas is distinctly more suitable for gas engine consumption than is oven gas, the logical outlet for the latter is through the open hearth furnace, for use in which, as has been fully demonstrated at Hubertushütte, it is pre-eminently well adapted.

OVEN GAS AT THE COCKERILL STEEL FOUNDRY.

The third article, which is a communication made to the alumni of the Liege Engineering School on May 1, 1910, by Charles Wigny, chief engineer of the Cockerill Company, and published in the *Revue Universelle des Mines* for August, 1910, describes the use of oven gas in a 4-ton, open hearth furnace in the Cockerill foundry. This article was reviewed from the German in *The Iron Age* of February 2, 1911, p. 318, but has lost so much in the double translation that it will be well worth the reader's while to refer to the original. A few of the main facts are here recapitulated. The furnace in question manufactures soft steel for the foundry, making with producer gas 3½ heats in 24 hours from a charge of 30 per cent pig iron, 35 per cent steel scrap and 35 per cent steel turnings, with a yield of 95 per cent. It is of special design, the hearth, gas producers and regenerative chambers for the air constituting one solid block of masonry. The gas is not preheated, but arrives from the producer with practically all of its sensible heat and without losing any tar or carbon. The fuel consumption under producer practice is 40 per cent, which, in view of the small capacity of the furnace, seems excellent. The use of oven gas as a fuel was begun without any change in the charge or in the furnace design, the oven gas was led directly into the main of the two producers, it having been the original intention to mix oven and producer gases. This was found unnecessary, however, and for almost a year previous to the date of writing not a single kilogram of coal had been charged. The oven-gas consumption per ton of steel was 435 cu. m., the daily production being raised from 14 to 16 tons in 24 hours. The oven gas had the composition:

H	57 per cent
CO ₂	1.5 per cent
CO	6.0 per cent
CH ₄	22.5 per cent
Nitrogen	13.0 per cent
Caloric power, calories per M ³ (Net.)	3640

Mr. Wigny then proceeds to show by calculation that oven gas is actually a more efficient fuel in the open hearth than is producer gas; that it is better economy to employ oven gas in the open hearth than to use it for the production of power with either a steam engine or turbine or a gas motor; and that with even a relatively poor oven gas the flame temperature amply suffices for steel melting. On the other hand he points out that blast-furnace gas at the best produces a flame temperature that is very near the low line as far as open-hearth requirements are concerned,

and that a poor blast-furnace gas, even with air and gas both preheated to 900 deg., is quite inadequate.

THE HIGHER EFFICIENCY OF COKE OVEN GAS.

The fourth and last paper referred to is by M. Tarsenster in the *Revue Universelle des Mines*, November, 1910. This article has been reviewed from the German in *The Iron Age* for May 18, 1911, p. 1911, p. 1232; but, as C. C. Tufts very justly observes (*The Iron Age* for June 22, 1911, p. 1501), this article also suffered in the double translation.

Trasenster's contribution is best read as a sequel to Wigny. He states that the very favorable results described by Wigny for the 4-ton furnace in the Cockerill steel foundry have been confirmed by a like experience with the 7-ton open-hearth furnace in the same foundry, and on the basis of some very interesting calculations explains why coke-oven gas shows higher thermal efficiency in the steel furnace than producer gas and that although under some conditions the efficiency of oven gas in the gas engine may be greater than its efficiency as an open-hearth fuel, nevertheless the greater cost of producer coal as compared with the poorer grades that may be employed for steam raising may probably even then swing the actual balance of economy in favor of burning coke-oven gas in the open-hearth furnace.

THE RESULTS SUMMARIZED.

The gist of the evidence to date appears, therefore, to be:

1. That as regards steel works consumption, the most economical disposition of the surplus gas of the by-product coke oven is as an open-hearth fuel—a disposition which has been shown to be perfectly practical for the gas, either alone or mixed with producer gas.

2. That the furnace tonnage is thereby not diminished, but rather increased and the furnace life is not materially impaired. Recorded experience in this direction seems to be confined to Kattowitz and Seraing. It is unfortunate that the method employed at Hubertushütte for introducing the oven gas into the furnace has not been described, but the writer is informed¹ that mixture of the two gases is regarded as essential from the point of view of safety, the producer gas being lighted first and the oven gas introduced subsequently into the current of producer gas. At what point the introduction is effected is not stated.

Against these two successes must be scored the failure recorded by Herr von Maltitz at the works of the Dominion Iron & Steel Company, at Sydney, and as the writer has some knowledge of the manner in which the experiments at the latter place were carried on, a few notes regarding them may be of interest. The facts are as recorded by Herr von Maltitz, but other facts which seem to the writer to have a very important bearing on the case have been omitted.

FURTHER DATA ON SYDNEY PRACTICE.

In the first place, while it is true that natural gas is

ordinarily introduced at both sides of the ports, it is also true that the Pittsburgh furnace in vogue with this gas is characterized by its use of the open port with both chambers on air, which is distinctly different from the type of end employed in other districts for use with producer gas; and more especially does it differ from the type of end necessitated by the usual form of the Campbell type of tilting open-hearth furnace. It is a perfectly safe assumption that the furnace end in which it was attempted at Sydney to burn oven gas would not have proved successful with even natural gas, which is well known to be an almost ideal open-hearth fuel. The complaint was made by the Sydney melters that the oven gas "flew high to the roof"—which was true; but this manifestation was wrongly blamed upon the relatively high hydrogen content of oven gas. As will be shown later herein, the Sydney gas is of excellent quality. As a matter of fact, whenever we find the flame in an open-hearth furnace curling about the roof brick instead of sweeping rapidly over the bath, we may conclude with certainty, not that aught is amiss with the fuel, but that there is a pinch in the draft somewhere. Natural gas in a poorly drawing furnace will hug the roof just as closely as the oven gas ever did at Sydney. The same principle holds a *fortiori* when the congestion is aggravated by the introduction of a bountiful supply of steam-blown tar. The premature destruction of the roof recorded by Herr von Maltitz is simply an indication that a good fuel was misapplied. The writer firmly believes that under the same conditions fuel oil would have made an equally poor showing. Finally, the employment of coke-oven gas as a fuel at Sydney was only one of several very formidable metallurgical problems that confronted the management there at the outset and it was considered inexpedient at that juncture unnecessarily to complicate the situation by experimenting further with oven gas, but the ingenuity of Herr von Maltitz and of his associates had by no means been taxed to the utmost in this regard when the use of coke-oven gas was abandoned.²

A COMPARISON OF GASES AT BELGIAN AND GERMAN WORKS.

The following table (Table I) of analyses and of calorific values based thereon are for the Cockerill and Hubertushütte fuels.

²It is safe to say that if the management of the Dominion Iron & Steel Company had considered itself warranted by general economic consideration in resuming the attempt to utilize oven gas in this manner it would have done so without hesitation, because there is not, to the writer's knowledge, on this side of the Atlantic a single steel works where the utilization of the by-product is more carefully and effectually studied than is done at Sydney. For instance, the high phosphorus content of Dominion iron, which at first seemed a serious menace to production, has been turned to very material account, the procedure being to blow the molten pig in a basic vessel to low phosphorus and then to transfer to a basic open-hearth furnace, where, by mixing four or five pots of basic blown metal with perhaps one pot of molten pig, a 50-ton heat is finished in a very short space of time after the last pot has been blown, the "reboil" with molten pig counteracting entirely any superoxidation that may result from the after blow. In this manner during a recent month Dominion made nearly 12,000 tons of ingots from one 50-ton furnace, which must be conceded to be a remarkable performance. Under such circumstances, of course, fuel cost almost vanishes. The economy of the process has more recently been added to by marketing the basic slag as a fertilizer. The only other plant where this basic duplex process has been operated, so far as the writer is aware, is that of the Phoenix Company, at Hörde, Westphalia.

¹Private communication.

TABLE I.
Analyses

	Cokerill prod. gas.	Cokerill oven gas	Typical. Oberschlesien prod. gas	Typical. Hubertushütte oven gas.
CO	7.5	1.5	2.8	6.5
O ₂	..	..	..	1.2
C ₂ H ₄	..	..	..	2.0
CO	19.3	6.0	36.2	10.1
H ₂	12.3	57.0	14.1	38.7
CH ₄	1.3	22.5	..	16.4
N ₂	59.6	13.0	52.0	24.8

Calorific Values

	Net cal. per M ³	Air reqd. per M ³	Pr ducts of comb. per M ³	Eqv. vol.	Air Reqd.	Total gas and air.	Total waste gases.
Cokerill producer gas	1924	885	17.7	3.523	3.118	6.641	6.984
Cokerill oven gas	3608	3.678	1.363	1.000	3.678	1.678	4.363
An Oberschlesien prod. gas	1293	1.067	18.15	2.343	2.500	4.843	4.323
Hubertushütte oven gas	3629	2.986	3.711	1.000	2.986	3.986	3.741

The tables clearly show that while the products of combustion of oven gas are less voluminous than those of producer gas, they are, nevertheless, considerably more bulky than either an equivalent of producer gas or the air required to burn it. Wigny's furnace does not preheat the gas; therefore the air downtakes must be ample to accommodate freely all the products of combustion and insure proper draft. The Hubertushütte furnace, on the other hand, being of the usual producer-gas type, must needs keep both air and gas downtakes in commission, in order to insure good working draft. This has been effected in one of the two practical ways; viz., by keeping gas and air distinct and throwing the rich gas into a carrying medium of lean gas. The other way is to merge gas and air ports and utilize the whole regenerative system for air, introducing the rich gas cold at the neck of the furnace, as is done in the Pittsburgh district with natural gas and which is in principle the method followed in the 4-ton furnace at Seraing.

HOW FAR COKE-OVEN GAS CAN BE DEPENDED ON FOR
STEEL WORKS SUPPLY.

There is no reasonable doubt that scrubbed by-product, coke-oven gas of average quality can be substituted for natural gas and successfully burned unmixed with any other gas in the present Homestead type of open-hearth furnace without detriment either to furnace or to production, though there would be anticipated, other things being equal, a certain increase in the sulphur content of the product, dependent upon the character of the coking coal used. Nor can we doubt that with the same general type of furnace, successfully burning fuel oil, a tar tank could be coupled to the oil pumps, due regard being had to the fluidity of the tar, without prejudice either to the furnace or to production. Admitting this to be so, let us see to what extent in a self-contained steel plant, the by-product

coke oven installation may be depended upon for open-hearth fuel.

Let our blast-furnace plant, with a fuel ratio of 2,000 lb., deliver 1,000 tons of molten basic iron per day to our mixers. Let us use exclusively our own scrap which we shall suppose amounts to 20 per cent. Let our open hearth yield (ore not reckoned) be 98 per cent and let our fuel consumption be 8,000⁶ cu. ft. of natural gas per ton of steel. Let our natural gas have the following composition and calorific value:

Carbon dioxide	..
Oxygen	0.2
Ethylene	2.2
Carbon monoxide	0.3
Hydrogen	3.0
Methane	91.8
Nitrogen	2.5
Net calorific value (calculated), calories per cu. m.	8298
Volumes of air required to burn one volume of gas	9.13

Let our coke-oven gas have the following composition and calorific value:⁷

Carbon dioxide	3.2
Oxygen	0.4
Ethylene	2.8
Carbon monoxide	6.3
Hydrogen	41.6
Methane	29.6
Nitrogen	16.1
Net calorific value (calculated), calories per cu. m.	4240
Volumes of air required to burn one volume of gas	4.43

We can, therefore, figure roughly on an oven-gas consumption of 16,000 cu. ft. per ton of steel.⁸

The following is fairly typical of the charge and yield of a well-known type of by-product oven with a good average gas coal:

Charge: 5.65 gross tons of coal.

Yield—

Coke, net tons	4.43
Tar, U. S. gallons	57
Surplus gas, cu. ft.	122,600

On this basis we figure from our 1,000 net tons daily of coke a surplus of 5,100,000 cu. ft., or enough, none being diverted to other uses, to take care of 320 tons of steel daily; that is, about 25 per cent of our ingot production, which we may calculate as 1,219 tons. With a higher fuel ratio at the blast furnace we should, of course, have a correspondingly larger surplus of gas.

In addition to this gas surplus we have also $1,000 \times 57$

— = 12,867 gal. of tar as a by-product from

4.43

our daily quantum of coke. Figures on the calorific value of coal tar are by no means plentiful, but from an abstract of an address by President Godinet, of the Société Technique du Gaz en France, and which appears in the Journal of Gas Lighting, Water Supply,

⁶This is a most liberal allowance. Under standard conditions this figure should be well under 7,000 cu. ft. per ton for basic furnaces.

⁷The figures here given represent a typical analysis of the Sydney gas, and are taken from trustworthy notes in the writer's possession. This gas has at times been poorer, but was practically always at least of the excellence recorded for the Hubertushütte gas.

⁸This estimate is somewhat higher than the consumption given above for Wigny's 4-ton furnace. He reports the equivalent of 15,355 cu. ft. for a gas with a net calorific power of only 3640 calories, which, in view of the coal consumption of 40 per cent reported by him for the same furnace under producer practice, shows our forecast to be conservative in the extreme. Our estimate also takes no account of suspended tar and benzol, which quite appreciably enhance the actual heating value of oven gas.

⁹The figures given by Director Amende of Hubertushütte are equivalent to approximately 37,400 cu. ft. of surplus gas for the same oven charge. His figures seem to indicate a 36-hour coke.

¹Private communication.

²Wigny's value is 3640 calories.

³Amende's value, presumably allowing for benzol, etc., 3300 calories.

etc., July 13, 1909, p. 99, we select a heating value of 8,000 calories, that being the lowest of the several values there given. The highest value quoted by M. Godinet is 11,000 calories and some authors give 10,500 as a representative value. Using 8,000 calories and a specific gravity of 1.18, we calculate calories per gallons of tar as 35,736 virtually the same value as 35,961 per gallon of fuel oil of the usual 10,555 calories and specific gravity 0.900. Assuming like thermal efficiency in the furnace for both tar and oil, we find ourselves enabled with 12,867 gal. daily of tar to provide for the production of a further 250 tons of steel, on the basis of 50 gal. of fuel oil per ton of basic open-hearth steel—a reasonably liberal allowance.

We have thus substituted gas and tar for 47 per cent of our fuel consumption. In other words, figuring the coal consumption under producer practice as 700 lb. per ton, we effect a daily saving of about 180 tons of coal, which represents an absolute return of 14 per cent of the coal required to produce our daily coke supply, besides a certain small reduction in producer maintenance and wages; also, the life of the furnace should not be appreciably affected. When a melter states that fuel A is harder on the furnace than fuel B, he is simply unconsciously stating that he and his bricklayer have been accustomed to fuel B and favor furnace lines that suffer under fuel A. A melter accustomed to fuel A will aver contrariwise, and so on. If the Hubertushütte regenerators are situated immediately beneath the furnace and lack slag pockets; if the producer coal is relatively high in ash and the producers are fairly close to the furnace—all of which things are probable—the increase of checker life with oven gas, as recorded by Director Amende,¹⁰ is readily understood. Under normal conditions, however, the character, both physical and chemical, of the stock employed, has more to do with furnace life than has the fuel.

OBJECTIONS TO BY-PRODUCT GAS.

There are two objections that may be urged against the use of by-product gas:

1. A shop that depends upon tonnage for its livelihood must standardize in every possible way, and in the long run, other things being equal, an open-hearth mill burning the same fuel all along the line has the advantage of the mill which burns three fuels.

2. Not least among the advantages of a producer fired furnace is its absolute independence of every other part of the world, as far as fuel is concerned, except the coal yard. The furnace fired with oven gas or tar introduces an additional peradventure. If the normal fuel for the mill be natural gas or oil, this objection is negligible, since in the light of the above ventured opinions natural gas, fuel oil, oven gas and coal tar can be burned interchangeably in the oven-port type of furnace. If the normal fuel, however, be producer gas, two types of furnace are necessary; and in case of absolute failure of oven-gas supply, the open-port type will require at the very least one week's time to change it over to the producer-gas pattern. On the

other hand, of course, such failure of oven gas can be met by the use of stored tar for a time, or by fuel oil. A comparison of the reigning prices for tar and fuel oil, respectively, will be found to be very instructive in this connection, more especially in view of the probability that the former price will not rise and that the latter will not drop, at least, in the near future.

SMALL OPEN HEARTH FURNACES.*

By WALTER MACGREGOR.†

The design of a furnace to get the best efficiency from the fuel, depends entirely upon the nature of the fuel to be burned. To get the highest temperature our furnace body should be of such proportions that we could burn the necessary amount of fuel in the smallest possible space, and in order to burn a large amount of combustible in a small space a short flame is necessary. The factors governing the short flame, according to the fuel experts of the United States Navy, are: (1) a pure carbon fuel; (2) initial heating of the air which furnishes the oxygen for combustion; (3) intimate mixture of the oxygen with the fuel or diffusion; (4) large surface of the fuel presented for impact of this oxygen.

The first factor, the nature of the fuel, is settled for us, as we have decided upon fuel oil, with a probable analysis as follows: Carbon, 83.26 per cent; hydrogen, 12.41 per cent; sulphur, 0.50 per cent; oxygen, 3.83 per cent, and with a specific gravity at 60 deg. F. of 0.926. The heat value of this fuel, according to Du Long's formulæ, would be 19,481 B. T. U. per pound. From this analysis we can easily compute the quantity of air required for combustion and the products of combustion for any amount of fuel burned.

As a representative size of the small open-hearth furnace we will choose a 5-ton furnace as an example. We are to melt and reduce 5 tons of metal, and from the time of charging the heat till the time of charging the following heat we will assume 4 hours, and as the oil consumed varies so much in different furnaces, we will assume as an average fuel consumption 48 gals. of oil per ton of steel melted in this time, or 1 gal. per minute.

Considering 12 lb. of air required for burning 1 lb. of carbon, and 34.78 lb. of air required for burning 1 lb. of hydrogen, we have from the analysis of the fuel:

9.9912 lbs. of air required to burn the total carbon in fuel.
4.2161 lbs. of air required to burn the total hydrogen in fuel.
14.2073

16 correction of amount for oxygen in fuel.

14.0473 lbs. air required for complete combustion of 1 lb. of liquid fuel.

With fuel oil of 7.72 lbs. per gallon we have $14.047 \times 7.72 = 108.37$ lbs. air required per gallon of oil.

With air at 13.14 cu. ft. per lb. we have $108.37 \times 13.14 = 1424.11$ cu. ft. air required to burn one gallon of fuel oil.

Hence, to burn 1 gal. we must admit theoretically 1,424 cu. ft. of air per minute into the furnace. To

*Paper read before the American Foundrymen's Association.
†American Steel Foundries, Indiana Harbor, Ind.

this we must add the amount of air required in reducing the carbon and silicon in the metal: 10,000-lb. charge, of which $12\frac{1}{2}$ per cent is pig iron of about $2\frac{1}{2}$ per cent carbon and the rest steel scrap and billets of about 0.30 per cent carbon, all to be reduced to about 0.18 carbon at the time of tapping. From this we get the total carbon contents of the bath as 56.5 lb. to be reduced to 18 lb. of carbon, or $56.5 - 18 = 38.5$ lb. of carbon to be burned out in about 2 hr. of reducing the charge, and, as before, 157.6 cu. ft. of air are required to burn 1 lb. of carbon, we have $157.6 \times 38.5 = 6,070$ cu. ft. of air required in 2 hrs., or 50 cu. ft. per minute. All of this passes off with the products of combustion. In the same way we can determine the amount of air required in eliminating the silicon, which will run about 42 cu. ft. per minute. A certain amount of oxygen is also taken up by the manganese, but this is so small as to be neglected.

With the total theoretical amount of air required, $1,424 + 50 + 42 = 1,516$ cu. ft. per minute, we are in a position to determine the proper furnace proportions with due regard to the second circumstance in producing the short flame; "initial heating of the air." The volume of air is figured at a temperature of 72° F., which will be about the temperature of air entering our valve. The increase in volume of air at different points along its travel due to its increase in temperature must be the governing factor in designing the ports, flue openings, etc. As the volume of this air increases in a direct ratio to the absolute temperature, it follows that the volume occupied at any point may be computed when the temperature at that point is known.

In case of the air valve, due to the reversing feature of the furnace, this should be figured rather to accommodate the products of combustion than the entering air, as these are at a higher temperature, and will, therefore, require a greater area of flue. The temperature of the valve is a vital point in the problem of design, for any heat beyond this point toward the stack is lost as far as the furnace is directly concerned, and can only be used in the field of economizers. In determining the size of the valve we will first have to determine the velocity of the products of combustion through the valve due to the draft of the stack, and this in consequence gives as our starting point the design of the stack, which we would naturally consider as our finishing point.

A number of eminent authorities on chimney design have chosen 600° F. as the most economical stack temperature, and Rankine has spent considerable time in trying to prove it in his work on "steam engines." I have never seen an open-hearth stack with that low temperature, and will, therefore, base my calculation on a temperature of $1,000^\circ$ F. as being more nearly uniform with current practice. In my experience with small furnaces I find that the most satisfactory stack draft to be maintained is about 1 in. of water. This is a function of the height of the stack and the difference in temperature inside and outside the stack. With this difference in temperature and a draft of 1 in. of

water, we would get a stack 110 ft. high, and hence we will assume this as the minimum height to be desired. The velocity of gas due to the pressure head corresponding to this height of stack and temperature, allowing a 25 per cent friction factor, is a little less than 15 ft. per second, which is recommended by a number of authorities as good practice.

We have based our calculation, so far, on the theoretical amount of air required for combustion, but will design our stack and flues, as in the case of power-plant design, for an excess capacity of 100 per cent, which would be 3,000 cu. ft. of gas per minute, or 50 cu. ft. per second. This divided by the velocity of 15 ft. per second would give a sectional area of stack of $3\frac{1}{3}$ sq. ft., or a trifle over 2 ft. diameter, and we will assume 27-in. diameter of stack as best suited to this furnace, and plenty large enough to permit of any crowding of the furnace. This then will also be the size of the valve and flues leading to the valve from the checker chambers.

The second factor governing the short flames, "the initial heating of the air," spoken of before, is introduced by means of the reversing feature of the furnace through the checker chambers, and these chambers should be so designed as to slow up the travel of the products of combustion, in order that they may give up the major part of their heat to the checker brick, or that part of the heat which is not required to produce the stack draft. The cubical contents of these chambers should not be less than 75 cu. ft. per ton of steel melted per heat, and preferably in the neighborhood of 100 cu. ft. per ton. These chambers should be located behind the furnace and not immediately under the furnaces as in large ones, as they operate at a higher temperature and we should get the benefit of a good circulation of outside air under the hearth of the furnace. These chambers should be long and narrow or deep, in oil-burning furnaces, giving a very long travel to the products of combustion, before they reach the valve, as on account of the highly volatile nature of the fuel and the slowness with which many of the hydrocarbons mix with oxygen a great deal of the fuel will be out in the stack before it has undergone complete combustion.

The methods of gas analysis, as applied to steam boiler practice, will show some very interesting relations in this regard. In a 5-ton furnace which I have been operating a flue gas analysis will show the following:

	Carbon dioxide. Per cent.	Carbon monoxide. Per cent.	Oxygen. Per cent.
At the rear of the checker chambers 24 ft. back of the center line of furnace.....	6.4	3.1	0.2
In the air valve 9 ft. further back	8.8	0.3	8.0
In the stack 16 ft. further back.....	9.4	0.3	9.

With a decrease in temperature between the first and last point from $1,750^\circ$ F. in the rear of the chambers to 930° F. in the stack.

In case all the fuel were burned before it reached the stack the sum of the oxygen components of the flue gas would be 21 per cent, as there is 21 per cent by

volume of oxygen in all the air admitted to the furnace, the volume of the carbon element being so small as to be considered zero, but as a matter of fact the sum of the oxygen components at the valve is only 17.1 per cent, and even out of the stack it is only 18.7 per cent, which shows that there is some form of hydrocarbon gas occupying the other 4 per cent which is getting past the valve unburned and being wasted out in the stack. This, I think, shows very conclusively the necessity of having long chambers and flues in oil-burning furnaces to insure complete combustion of the gaseous fuel before reaching the reversing valve.

These figures are based on atomizing the fuel with compressed air instead of with steam, as with steam the hydrocarbons are slower in taking up oxygen, and a gas analysis at the valve will show a higher percentage of hydrocarbon gas unburned at the valve and a corresponding increase in stack temperature. A sample of gas at the base of the stack when steam was used for atomizing purposes showed the following analysis: CO_2 , 7.5 per cent; CO , 0.4 per cent, and oxygen, 9.5 per cent, or a total of 17.4 per cent of oxygen components out in the stack which is $5\frac{1}{2}$ per cent less than shown at the base of the same stack with air, and therefore, a less perfect combustion.

The third condition governing the short flame, "intimate mixture of oxygen with the fuel or diffusion," bears directly on the size and arrangements of the air ports and the furnace body. The size of the ports depends upon the size of the reversing valve, or vice versa, and the relation between the two is in a direct ratio to the absolute temperature at the two points, these temperatures being $1,490^\circ \text{F.}$ at the valve and $2,800^\circ \text{F.}$ at the ports, or in a ratio of 1 to 2. The ports therefore, should have an area of twice the area of the reversing valve. We will, therefore, have a total port area at one end of the furnace of about 7 ft. These ports should be carried out the full width of the furnace to prevent any short circuit of air through the furnace body, as the travel of gas through the furnace body should have the same velocity at all points to get the proper diffusion. These air ports should come well up above the hole in the monkey wall through which the oil burner enters the furnace, so that the air must come down on top of the flame rather than underneath it. This is a very important factor in designing a hot-working furnace.

The space to be allowed for hearth in small furnaces should not be under 10 sq. ft. per ton of charge and then, too, the shape should approach more nearly a square than the oblong shapes in general use, as this tends to give a better effect of the radiation of the walls and roof, and by widening the furnace we lessen the cutting action of the flame on the side walls and keep down the repair bills.

As to the length of the furnace body, this should be governed by the length of the oil flame, for the hottest part of the flame should be about the center of the furnace. It has been my experience that I have not been able to get a flame that was intense enough to melt

down a charge of metal any less than about 8 ft. from the tip of the burner to the hottest part of the flame, and as the tip of the burner should stick clear through the monkey wall, which will extend 9 in. beyond the ports of the furnace at least, we will get as a minimum furnace length twice the length of the flame as mentioned above, plus twice the width of the ports, plus twice the thickness of the end walls, plus twice the 9-in extension of the monkey wall beyond the ports, or a total of about 22 ft., as the minimum length of the outside of the furnace body.

The fourth circumstance governing the short flame, "large surface of fuel presented for impact of oxygen," as a matter of oil burners and atomizing agents, and has furnished inspiration to thousands of inventors—all to very little purpose. The matter of atomizing this fuel oil is one of overcoming the surface tension of the oil and breaking it up into very fine particles, so that it will present greater surface for contact with the oxygen, and the two methods in use, superheated steam and compressed air, give a mechanical efficiency so small that you can barely find it at all.

There is a great deal of discussion at the present time on the needless waste of using compressed air for atomizing purposes when superheated steam will answer, but in the small casting business one of the main difficulties is getting the metal hot enough to run the thin sections in the molds, and since, by its very nature compressed air, while atomizing the oil, furnishes at the same time oxygen for combustion, and that, too, very intimately mixed with the oil, it is quite evident that by using air we would get quicker combustion, a shorter flame and a somewhat hotter furnace.

In conclusion, I will say that in operating a furnace designed along these lines it will not be a difficult matter to get out six 5-ton heats in 24 hrs., and still have the metal hot enough to pour many castings weighing a fraction of a pound each. With a 5-ton heat it is not uncommon to pour as high as 175 molds, consuming about 50 min. in pouring. The metal must, therefore, be extremely hot at the time of tapping the heat.

The Manila Merchants' Association is seeking to make more extensive use of coir, or cocoanut fiber, many million pounds of which are now going to waste in the Philippines. The exports of copra from the islands indicate an annual production of 48,000,000 cocoanuts, or probably one-half of the real production. It is estimated that these 48,000,000 husks contain some 4,000 tons of yarn fiber and 1,000 tons of bristles. Notwithstanding that fact almost all the 7,000,000 lbs. of coir yarn, worth \$244,000, imported into the United States in 1910 came from British East Indies. Very little of the raw coir, or cocoanut fiber, is imported, the amount last year being 90 tons, worth \$8,581. It is desired by the Manila Merchants' Association to make use of the enormous supply of this material by spinning and weaving it into textile fabrications, etc.

THE STOCK PROCESS FOR STEEL CASTINGS.

A new type of small Bessemer converter has been used at the works of the Darlington Forge Co., Ltd., Darlington, England, with interesting results. Patents covering the construction and process have been secured by Guy J. Stock, of Darlington. The following description of the process is taken from *The Iron Age*:

The practice at the Darlington plant has been developed in the past 21 months, the converter in use there being of three tons' capacity. At seven other steel foundries in Great Britain the Stock converter is either in use or is being installed. Thos. Firth & Sons, Ltd., Sheffield, England, have ordered a second plant, which will give that firm a 3-ton and a ½-ton converter.

verter is pointed toward the air heater or economizer. The hot gases from the burning oil are drawn through the heater by means of the chimney stack, as shown in Fig. 1. Blast is supplied in the case of the 3-ton converter by a Roots pressure blower with a capacity of 3,000 cu. ft. per minute, the blowing pressure varying from ¾ lb. per sq. in. for melting to 3½ lbs. per sq. in. as the maximum for converting.

Air from the blower is delivered into the heater through the inlet pipe shown in the plan in Fig. 1, and after passing through the system of pipes goes to the converter through the pipe *a*, Fig. 2, which terminates in a two-way branch pipe *a'*, leading into the blast boxes *cc* and through the tuyeres *d* into the vessel. The oil supply pipe is shown at *b*, and *b'b'* are small oil jets extending from it into the tuyeres. In melting, the air which passes through the tuyere enters the vessel at a temperature of about 800° F., and in the burn-

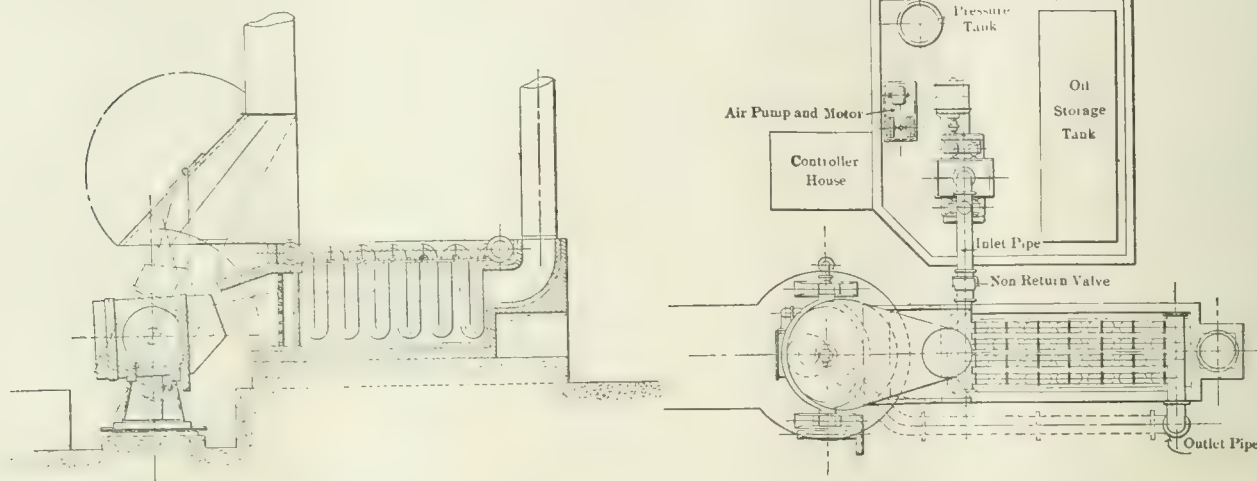


Fig. 1—Elevation and Plan of Stock Converter Installation.

One of the features of the Stock process is that the cupola is dispensed with; the melting of the metal previous to blowing is accomplished in the converter itself, crude oil being used as fuel. Another feature is that there is an "economizer," as the inventor terms it, consisting of a series of U-shaped pipes and between these the hot waste gases resulting from the melting operation are led, the blast in turn passing through these heated pipes to the converter.

In Fig. 1 is shown an elevation and plan of a Stock converter installation, and in Fig. 2 is a section showing the arrangement of the tuyeres and of the oil and blast pipes.

The vessel is made of oval section so as to expose the largest possible surface of the metal to the action of the oil burners, and the tuyere box was designed for the most effective action of the blast and of the oil jets. The converter is lined with silica brick running up to a thickness of 13 ins. around the tuyeres. For convenience of working, in addition to the mounting on trunnions working in roller bearings a turntable is provided for the revolution of the vessel. After charging, the vessel is moved through an angle of 90° into the position for melting. Here the nose of the con-

ing of the oil a very close approach to perfect combustion has been secured. With a 3-ton converter the metal is melted in about 1½ hours. The blow which follows takes from 15 to 25 minutes. Thus a blow every 2 hours can ordinarily be made. The vessel is brought into position for pouring by turning it through 180° in a horizontal plane from the position occupied during the blow.

The oil used for melting is forced by the blower or by an air compressor into a smaller tank which contains a sufficient quantity for five to six meltings. This service tank is fitted with a coil through which hot air or steam can circulate so as to decrease the viscosity of the oil. The tank is connected to a small independently driven compressor, maintaining a constant pressure of 30 to 35 lbs. per sq. in., forcing the necessary amount of oil through a flexible tube to the oil tubes in the tuyere box. These latter are steel tubes with an internal diameter of about 1-16 in. and point through the center of the tuyeres. When the melting operation is completed these oil pipes are withdrawn, the tuyere box being so arranged that this can be done in a few seconds.

In addition to dispensing with a cupola and provid-

ing the economizer for preheating the air, the advantages of the Stock converter are stated as follows: With liquid fuel for melting there is no taking up of impurities in this process. The high temperature of the melted charge permits of using pig iron low in silicon as well as high percentages of scrap. A very fluid

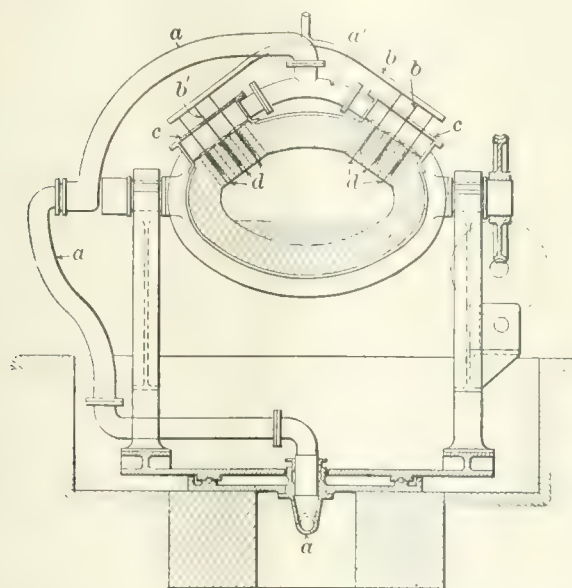


Fig. 2—Sectional View Showing Arrangement of Tuyeres.

steel is secured, making it possible to use this process for the manufacture of difficult and intricate castings. The products manufactured from Stock converter steel include wheel centers, high carbon steel wire, low carbon seamless drawn tubes, chipping and calking chisels and pneumatic tools; also high-speed tools for steel containing tungsten, chromium and vanadium.

With an operation of 52 tons a week in a 3-ton Stock converter, working single shift, the cost per ton has been worked out as shown in Table I. The cost of the charge in this table is based on a price of 60 shillings or \$14.58 per ton for Bessemer, or hematite, iron, and an ascertained loss of 13 per cent in conversion, with a further loss of $3\frac{1}{4}$ per cent in ladle spillings and floor scrap.

TABLE I.—COST OF STEEL PRODUCED IN A 3-TON STOCK CONVERTER.

Metal and mixings (23 cwt. 3 qr. 10 lb.)	\$17.02
Fuel oil (40½ gal. at 5 cents per gal.)	2.03
Electric power (42 units at 1 cent per unit)	.42
Direct wages	.73
General repairs and upkeep, including linings	1.70

Cost of converted metal	\$21.90
Add allowance for ladle linings and repairs	.50

Cost of metal in molds	\$22.40
Add allowance for depreciation and interest	\$0.53
Add allowance to cover royalty charges	.83

Total cost of steel	\$23.70
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The accompanying tables give the analyses and results of physical tests of Stock converter steel. In Table II tests are given of two pieces cut out of a 10-in. stop valve which had been subjected to a hydraulic test of 500 lbs. per sq. in. Table III gives the result of tensile and bending tests of test pieces. The

seven numbers given in the table correspond with those marked on the test pieces.

TABLE II.—TWO PIECES CUT OUT OF 10-INCH STOP VALVE.

Test No.	Description.	Orig. area. in.	T. S.		Elongation. in.	Per cent.
			1,000 lb.	Per Contrac-		
			sq. in. of orig. area.	tion. Per cent.		
1	10-in. stop valve	0.196	27.8	—	3	31.3
2	10-in. stop valve	0.250	28.1	53.6	2	37.0

Analysis: Carbon, 0.22 per cent; Manganese, 0.508 per cent; Silicon, 0.130 per cent; Sulphur, 0.014 per cent; Phosphorus, 0.041 per cent.

TABLE III.—RESULT OF SEVEN TENSILE AND BENDING TESTS OF

STOCK CONVERTER STEEL.

Test No.	Description.	Original area. In.	T.S. 1,000 lb. per sq. in. orig. area.	Elongation.		Bond.	
				Contraction Per cent.	Per cent.		
1	12 Roof bars.	0.50	57.1	51.3	3	30.0	180° N.B. 1 in.sq.
2	4 Radial wheels.	0.50	58.0	50.8	3	34.0	" " "
3	1 Ford, bearing keep	0.50	28.8	52.3	3	33.0	" " "
4	1 Ford, bearing keep	0.50	29.6	47.2	3	32.0	" " "1 in. diam.
5	1 Aft. bearing keep	0.50	30.0	52.3	3	32.0	" " "
6	4 Side frames.	0.50	29.0	50.8	3	28.0	" " "
7	10 Platform supports	0.50	29.4	51.6	3	30.6	" " "

ANALYSES.

	No. 1.	No. 2.	No. 3.	No. 4.	No. 5.	No. 6.	No. 7.
	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
Carbon	0.22	0.22	0.20	0.23	0.23	0.22	0.22
Man-ganese	0.607	0.540	0.600	0.567	0.540	0.607	0.540
Silicon	0.273	0.178	0.237	0.285	0.273	0.273	0.296
Sulphur	0.012	0.019	0.013	0.016	0.010	0.012	0.018
Phos-phorus	0.020	0.024	0.018	0.029	0.020	0.020	0.025

Charles S. Powell, 165 Broadway, New York, is the agent in the United States for the Stock process. Mr. Powell recently returned from an extended stay in England devoted to a study of the practice at the Darlington steel foundry.

Tests are to be made by the Isthmian Canal Commission to determine the value of cement mortar, applied to iron plates by the "cement gun," as a preservative of iron. Twelve plates, 63.8 by 14 ins., have been coated with a 1 to 3 mortar of cement and sand, after they were cleaned to gray metal by the sand blast process. Six of these have been covered with a $\frac{1}{2}$ -in. coating, and the remaining six with a 1-in. coat on one side and a $1\frac{1}{2}$ -in. coat on the other. Three plates of each kind have been sent to Balboa and three to Cristobal, where they will be kept immersed in salt water to test the mortar method of preventing corrosion. Two plates of each kind will be taken from the salt water bath at the end of three months, and one-half of the coating will be removed to determine the condition of the metal. The duration of the test for the balance of the plates will be determined later.

METALLURGY.

TITANIUM IN IRON AND STEEL.*

By CHARLES V. SLOCUM.

In electrometallurgy no development of late years has been of greater importance than the manufacture of ferro-titanium in the electric furnace. It was not until 1903 that Auguste J. Rossi was able to produce the remarkable material now widely known as "Titanium Alloy." Some thirty years or more of his life has been devoted to developing the methods of manufacture and the uses of this material, but it was not until he used the electric furnace in 1903 that his life-long efforts were crowned with success. Since that time the manufacture and consumption has grown with leaps and bounds, and now the plant occupies a large tract of land, and railway facilities are necessary for the prompt handling of the large output.

As the uses of this alloy are so many and so varied, it has been suggested by a well-known member of this society that the more important features be discussed in a suitable paper. Metallurgists have practically all agreed that the use of a small percentage of an alloy of titanium is of benefit in both iron and steel, but some question has arisen as to the form which this alloy should take. In a recent publication¹ the statement has been made that carbon-free metals dissolve more easily than metals containing carbon, since the latter contain carbides. The experience of the writer does not bear out this fact, he having found that a carbon-free alloy was not at all adapted to use in the iron and steel industry, and the manufacturers were obliged to make a product containing from 5 to 8 per cent carbon for such use. Our early experience with an alloy of iron and titanium free from carbon, but containing 5 to 10 per cent aluminum, resulted in almost complete failure, as the aluminum when present in any appreciable quantity made the steel brittle and its oxide showed great tendency to remain in the steel.

"Titanium Alloy," as stated above, contains 5 to 8 per cent of carbon, mostly in the form of graphite. Analyses of this material made in the laboratory of Dr. C. F. McKenna show the following proportions of the two forms of carbon:

Sample No.	Graphitic Carbon.	Combined Carbon.
141	9.661	0.117
162	9.179	0.12
201	7.012	0.13
218	6.234	0.118

From these analyses it appears that titanium acts very much like silicon in that it causes a separation of

carbon as graphite. Dr. G. B. Waterhouse recently cited an interesting experiment to bear out this fact: A ladle of iron for making malleable castings was treated with 10 per cent titanium alloy sufficient to equal 0.1 per cent metallic titanium added to the iron. The original iron gave castings showing a white fracture with a little graphite in the center. The titanium-treated metal showed a white border between $\frac{3}{8}$ and $\frac{1}{2}$ in. (1 to 1.25 cm.) deep and a gray center showing the separation of graphite. Contrary, therefore, to the viewpoint of Dr. Goldschmidt, there need be no fear of the non-solution of the electric furnace titanium alloy, as it does not contain more than the merest fraction of the unready soluble carbide.

The density of a ferro-alloy to be used as ferro-titanium is used has an important bearing on its incorporation in the molten bath. Careful determinations made on the electric furnace product containing both 10 and 15 per cent titanium show a density of 6.20 to 6.40. The metallothermically produced alloy shows a density of between 6.20 and 6.30. The difference between the densities of the two alloys is therefore hardly noticeable.

The alloy is best added to the ladle of steel after tapping from the furnace and before the slag begins to run. For open-hearth steel the supply should be placed convenient to the ladle and shoveled in precisely as so much coal. For soft steel an addition equivalent to 0.03 per cent titanium is sufficient to make the steel more ductile. Larger quantities are of increased benefit, but cannot always be added because of the carbon content and also because of the density imparted to the steel, which increases as the proportion of alloy is increased, and therefore is not always desirable. In high-carbon steels this increased density is desired and more alloy is used. The addition of the alloy permits the use of a considerably higher carbon content without increasing the brittleness, and several railroads are now using as much as an equivalent of 0.10 per cent titanium or more in securing tough, durable rails.

In the crucible practice the best results have been obtained by adding the titanium alloy to the pots before they are removed from the furnace, giving more time for washing and deoxidizing before teeming.

Leading authorities seem to agree that titanium achieves its remarkable results through its strong deoxidizing powers together with its effect of giving the slag formed sufficient fluidity to completely separate it from the metal.

"The presence of titanium oxide lowers the melting point of slags occluded in iron and steel and imparts thereto sufficient fluidity to account for their elimination."²

*Paper presented at the Twentieth General Meeting of the American Electrochemical Society, in Toronto, Canada, Sept. 21-23, 1911.

¹Goldschmidt, The Melting Point and Its Relation to Alloying Capacity. *Met. & Chem. Eng.*, 9, 348 (1911).

"Titanium . . . has a stronger affinity for oxygen than have the well-known deoxidizers; . . . it probably gives the slag such a consistency that it separates more completely from the molten iron."³

"The treatment of all steels with ferro-titanium for the purpose of purifying the metal is strongly recommended, the presence of nitrogen to the extent of .02 to .035 or .045 in certain steels being enough to cause the metal to break asunder, destroying all elongation and reduction of area."⁴

"The great affinity of oxygen and titanium is an absolutely sure means of completely deoxidizing the steel, the advantages of which need no further elaboration. It has been proved that an ingot of steel containing a very small quantity of titanium will show practically no segregation."⁵

The cost of treatment with titanium varies from a minimum of 25 cts. to a maximum of \$2.00 per ton of metal treated. It is the cheapest deoxidizer above the grade of manganese or silicon, and a far greater purifier than any other alloy. The amount to be used is very small, and should be proportioned according to the carbon content of the steel and to the amount of impurities present. To the electrometallurgist it is a triumph worthy of record that Rossi should take an element like titanium, so long considered useless, and make out of it so wonderful a servant in the iron and steel industry.

The mine output of gold, silver, copper and lead in California in 1910 had a value of \$27,020,405, according to figures compiled by Charles G. Yale and just made public by the United States Geological Survey. The production of gold was \$19,715,440; that of silver 1,840,085 fine ounces, valued at \$993,646; that of copper 48,700,756 pounds, valued at \$6,184,996; and that of lead 2,870,977 pounds, valued at \$126,323. These figures show a decrease in the output of gold, silver and copper as compared with the figures for 1909, but a large increase in lead. The 1909 figures are as follows: Gold, \$20,237,870; silver, 2,098,253 ounces, valued at \$1,091,092; copper, 57,288,281 pounds, valued at \$7,447,476; lead, 1,502,597 pounds, valued at \$64,612. There were 1,079 mines producing gold, silver, copper or lead in California in 1910, of which 564 were gold placer mines. Of the deep mines 485 were gold mines, 9 were silver mines, 10 were silver-lead mines, and 11 were copper mines. Of the placer producers, 168 were hydraulic mines, 72 were dredges operated by 41 companies, 139 were drift mines in ancient river gravels, and 185 were sluicing mines. Measured by the number of producers as well as by tonnage and metal output, deep mining decreased somewhat in California in 1910. Among the placers sluicing decreased also, but dredge and drift mining increased.

³Reedley Stoughton, U. S. Patent Office Proceedings, Ser. No. 409610.

⁴Henry M. Howe, *Ibid.*

⁵H. le Chatellier, Paper, Congress of Metallurgists, Belgium, 1907.

⁶E. von Moltitz, *Stoll und Eisen*, No. 41, 1909.

THE ANALYSIS OF MANGANESE BRONZE.*

By JAMES R. HUBER.†

In nearly all of the specifications for manganese bronze castings, it is quite noticeable that attention is called to the physical requirements and very little is said regarding the chemical analysis. This is no doubt partly due to a lack of knowledge on the part of the engineers who draw up the specifications of the exact composition of this rather complex alloy. The proper knowledge of the manufacture of this alloy is essential when one wishes to meet physical specifications, and is only obtained after long practical experience. One of the factors that go to make up this knowledge, is, undoubtedly, the proper control of the chemical composition, especially since small variations in the composition have a marked effect on the physical properties. It is with this idea in view that a paper on the Analysis of Manganese Bronze has been prepared, and it is hoped that it will help both the manufacturer and the user to check in one detail at least, this very valuable alloy.

A method for the analysis of this alloy was worked out in our laboratory last year and published in the *Metallurgical and Chemical Engineering Journal* for August, 1910. This analysis presents several difficulties, especially the determination of zinc and aluminum. By the original method the precipitate of zinc was contaminated with iron, and the precipitate of aluminum was contaminated with zinc. As we have gained knowledge by experience, we have been able to make some improvements on the original method. I will include a brief outline of the complete method for the benefit of those who may not have read the first paper.

A 5-gram sample is dissolved in 40 cc. of 50 per cent nitric acid (HNO_3). The small precipitate of metastannic acid obtained, is invariably in such a finely divided state, that much time is consumed in endeavoring to obtain a clear filtrate. A slight cloudiness in the filtrate will mean a result from .05 per cent to .15 per cent too low on the tin. We find it advisable to cool the solution after boiling off the brown fumes, stir, and pour on a double S. & S. No. 589 Blue Ribbon filter paper. Fill the funnel once or twice and then apply suction, gradually at first, until the paper is well clogged with the precipitate. If the first of the filtrate is not clear, return it to the beaker and pour it on the paper again. With these precautions it is seldom necessary to repeat this operation and a practically clear filtrate is obtained. It is better to wash the precipitate with hot 10 per cent nitric acid (HNO_3) instead of 2 per cent as the more dilute wash water has a tendency to start the precipitate running through the filter paper.

The filtrate is then evaporated to white fumes with concentrated sulphuric acid (H_2SO_4). 5 cc. is not

*Paper read before the American Brass Foundry Association, Pittsburgh meeting, May 23, 1911.

†Metallurgist, Lumen Bearing Company, Buffalo, N. Y.

enough, as the mass forms a hard cake and has a tendency to spatter when nearly dry. We now use 15 cc. to 20 cc. With this amount it will not be necessary to make a further addition when removing copper by electrolysis. The bronze will not contain any lead if made with high grade spelter, but should a trace be present, it is filtered off and weighed as lead sulphate (PbSO_4) on a platinum Gooch crucible. The filtrate is diluted to 500 cc. in a 500 cc. graduated flask.

Four 100 cc. portions are measured out with a pipette or graduated flask into tall non-lipped 200 cc. beakers. Two of them are neutralized with ammonium hydrate (NH_4OH). 7 cc. of a mixture of 2 parts 50 per cent nitric acid (HNO_3) and 1 part 50 per cent sulphuric acid are then added and the solution diluted to about 150 cc. The copper is plated out using either a rotating anode or a solenoid. With the latter a brighter and more adherent deposit is obtained in less time than with the former. The shape of the spiral anode causes the deposit to be more evenly distributed over the surface of the cathode and a much stronger current can be used. With a gauze cylinder cathode having 150 to 200 sq. cm. of surface, 10 amperes will deposit about $1\frac{3}{4}$ grams of copper in 15 minutes. Three amperes was the highest current we were able to use with the rotating anode and obtain a satisfactory deposit. We find it better to dry the cathodes in an air bath at about 100°C .

From these two portions we have duplicate determinations on the copper. In the filtrate from one the iron is precipitated with ammonium hydrate (NH_4OH), dissolved in dilute hydrochloric acid (HCl), reduced with stannous chloride (SnCl_2) and titrated with potassium permanganate (KMnO_4), according to Blair. In the other portion the manganese is determined by the well known bismuthate method. Rinse off the anode with a little sulphurous acid (H_2SO_3) to remove the slight deposit of manganese dioxide which forms upon it and add these washings to the solution in the beaker. Bromine water and ammonium hydrate (NH_4OH) are then added, the solution boiled and the precipitate of manganese dioxide (MnO_2) filtered out and re-dissolved in sulphurous acid (H_2SO_3). 10 cc. nitric acid (HNO_3) and 30 cc. water (H_2O) are added, the solution boiled a minute and $\frac{1}{4}$ gram of sodium bismuthate added. Heat till the pink color entirely disappears. Add a few drops of sulphurous acid (H_2SO_3) to clear the solution and boil again to expel all nitrous fumes. Cool the solution until quite cold and again add a slight excess of sodium bismuthate. Stir a minute or two and then add 50 cc. of 3 per cent nitric acid (HNO_3) and filter through asbestos on a platinum cone, using suction. Wash with 50 cc. more of 3 per cent nitric acid (HNO_3), reduce with standard ferrous ammonium sulphate solution and titrate the excess with potassium permanganate (KMnO_4), which has been standardized against the ferrous ammonium sulphate.

The copper is removed from the other two 100 cc. portions by electrolysis without the addition of nitric acid (HNO_3). The zinc is separated from the aluminum, manganese and iron by precipitation with hydrogen sulphide (H_2S). Instead of reducing the iron by boiling with metallic iron or aluminum, or sodium thiosulphate, as suggested in Waring's modified method, we find it saves time to reduce with sulphurous acid (H_2SO_3). The use of aluminum is prohibited as we determine aluminum in the filtrate from the zinc. Methyl orange is used as an indicator when neutralizing the solution and as it is not sensitive in a hot solution, it would be necessary to cool after boiling with any of the above reducing agents. In using sulphurous acid (H_2SO_3) it is only necessary to add about 10 cc. to the cool solution, stir and neutralize with sodium hydrate (NaOH) and sodium bicarbonate (NaHCO_3) using methyl orange as an indicator.

We obtain the zinc sulphide in a granular condition and entirely free from iron by precipitating in the presence of hydrochloric acid (HCl) and ammonium sulphocyanide, as suggested by Kemmerer in an article in the *Journal of Ind. & Eng. Chem.* for September, 1910.

After neutralizing, dilute the solution to 400 cc., add 4-5 drops of hydrochloric acid (HCl) and about two grams of ammonium sulphocyanide (NH_4SCN). Heat to boiling and pass a rapid stream of H_2S for about a half hour, keeping the beaker on a hot plate. Under these conditions the precipitate is rarely, if ever slimy, but should it form in that condition, it might as well be thrown away as it is impossible to filter it clear. After the gas has been passing about fifteen minutes, and the zinc is nearly all precipitated, add a drop of a dilute solution of sodium bicarbonate (NaHCO_3). If the black precipitate of ferrous sulphide (FeS) formed, dissolves instantly, add another drop. But do not add any more if the black precipitate dissolves slowly. This precaution is taken to insure complete precipitation of the zinc. There is a possibility that enough acid will be liberated in addition to the 4 or 5 drops added to prevent the last traces of zinc sulphide (ZnS) being precipitated. As this addition is not made until after most of the zinc sulphide (ZnS) is formed there is no danger of iron being carried down. Continue passing the gas about 15 minutes longer. Allow the zinc sulphide (ZnS) to settle, filter and wash with hot water. Dissolve in dilute hydrochloric acid (HCl) and precipitate as phosphate with sodium ammonium hydrogen phosphate ($\text{NaNH}_4\text{HPO}_4$).

The aluminum is determined in the filtrate after boiling off the hydrogen sulphide (H_2S) and concentrating to about 200 cc. Add a little nitric acid (HNO_3) to decompose the ammonium sulphocyanide (NH_4CNS). Boil off the cyanogen fumes, add five grams of ammonium chloride (NH_4Cl) and precipitate the iron and aluminum as hydroxides by making the solution exactly neutral with ammonium hydrate

(NH_4OH). Boil, filter and dissolve in dilute hydrochloric acid (HCl). Dilute to 400 cc. Add 10 cc. of a 10 per cent solution of sodium ammonium hydrogen phosphate ($\text{NaNH}_4\text{HPO}_4$) and then ammonium hydrate (NH_4OH) until a slight but permanent precipitate forms. Then add $1\frac{1}{2}$ cc. of con. hydrochloric acid and 50 cc. of a 20 per cent solution of sodium thiosulphate ($\text{Na}_2\text{S}_2\text{O}_3$). Heat to boiling and add a mixture of 15 cc. of a 20 per cent solution of ammonium acetate and 8 cc. of strong acetic acid. Boil ten minutes, allow to settle, filter and wash with hot water. The precipitate is ignited and weighed as aluminum phosphate (AlPO_4).

In conclusion I desire to express my thanks to my assistant, Mr. Woiciechowski, who has done most of the experimental work on the details of this method.

THE ELECTRICAL PRECIPITATION OF SUSPENDED PARTICLES.

By F. G. COTTRELL.

The following abstract of this paper which appeared in the *Journal of Industrial and Engineering Chemistry* for August, 1911, while of peculiar interest to all metallurgists, will also be found interesting by all chemical engineers.

Prof. Cottrell begins with a brief account of the development of the idea. In 1824 Hohlfield suggested it might be used to suppress ordinary smoke, and later Guitard repeated the suggestion. Sir Oliver Lodge rediscovered the phenomena independently, lectured on the subject, and aided in installing devices, which had been patented by A. O. Walker, at the Dee Bank Lead Works. Wilmhurst 5-ft. machines were driven by small steam engines; no further developments came at this time, the experiment probably being disappointing.

As none of the ideas or patents had come to commercial realization the work of Lodge was repeated, the industrial application of which is now greatly facilitated by the development of electrical devices.

It is found that an alternating electromotive force agglomerates particles suspended in a fairly quiet medium, this principle is used to separate water from crude oil when it is found emulsified in the California oil fields.

With large volumes of moving gases agglomeration and settling works too slowly and direct currents, which impell the particles against a flat electrode, are used. Apparatus for producing alternating currents up to 100,000 volts potential have been perfected and with the aid of the mercury rectifier promises considerable toward use for dispelling fog and smoke in the open.

For the installations which are to be mentioned the direct, pulsating current as supplied to the electrodes is made by transforming an ordinary alternating current to 20,000 or 30,000 volts and then rectifying this

to direct current with a commutator driven by a synchronous motor.

The electrode which is to receive the particles is a plane surface and easily provided. The discharge electrode presents obstacles to the both powerful and uniform distribution of the current. The accidental solution of the difficulty came by discovering that the fine points of non-conductors discharge sufficiently by virtue of dust and moisture ordinarily found upon them. A cotton^ocovered wire served admirably in air while wire coated with asbestos or mica scales serves well in hot and acid vapors.

In the spring of 1906 it was demonstrated that sulphuric acid mist generated by a small contact plant could easily be precipitated upon the inside of a bell jar with 3,700 volts discharged from a metal screen cylinder wrapped with asbestos twine. From the original 110-volt alternating line, which supplies the primary of an induction coil, is driven an induction motor to operate the synchronous motor which establishes the contact in the secondary of the coil exactly at the voltage peak, thus furnishing the interrupted high potential current. The collecting electrode being grounded left only one conductor which supplied the discharge electrode. This small apparatus precipitated the acid fume from about 20 liters volume per minute.

A larger precipitating unit was then built which would handle from 100 to 200 cu. ft. of gas per minute. This apparatus worked on some of the fume from a Mannheim contact sulphuric acid unit at the Hercules Powder Works of San Francisco Bay. The dry sulphur trioxide will not mix with water, but after water vapor has evaporated into the gas the dense white cloud of sulphuric acid would readily condense on the electrodes of the apparatus. As before, asbestos-wound screens served for discharge electrodes but the gas was made to thrice pass under the influence of the discharge. Reproductions from photographs show the complete success of the treatment.

A larger unit to handle all the gas from a Mannheim unit became a commercial success.

The next development of the process came at the plant of the Selby Smelting and Lead Company, also on San Francisco Bay. At this plant the gases from the lead furnaces were successfully treated in a bag house but the roaster gases and the vapors from the refining plant contained so much sulphuric acid that bags would be quickly destroyed if an attempt were made to use them.

The smaller volume of gas, that from the refining pots, was first experimented upon. Lead plates, each 4 ins. wide and 4 ft. long, were hung in rows in a lead-lined flue. These plates were 4 ins. apart and between each two were hung the asbestos and lead-covered iron rods of the discharge electrodes. Here the current is taken from the 460-volt system of the plant and transferred and rectified to the 17,000-volt line for the electrodes. About 2 kw. is all the power required while labor attendance and repairs have cost less than \$20 a month during three years' operation.

*Abstracted by H. B. Pulsifer.

It was thought that the fume from the roasters could be treated as to form a powdery precipitate on the electrodes, and, with this in mind quite an equipment was made with brick and iron construction. However, so much water vapor and acid was found in the gases that this construction had to be abandoned and lead substituted. The flue was finally constructed to be 6 ft. square and 32 ft. long, in which were hung 38 rows each of 16 lead plates 6 ft. long and 4 ins. wide with the discharge rods between. Between 10 and 15 kw. was all the power required, this precipitating out a grayish mud on the lead plates which could be quickly washed clean at intervals.

To protect the construction and to have all the material to be separated present as suspended particles instead of as true gases it was necessary to cool the gases below 150° C. This equipment not yet having been provided the operation was not continued.

Prof. Cottrell devotes the remainder of the paper to the case of the Balaklala Smelter at Coram, Cal.

This plant is the most recent of the Shasta County smelters, being erected after the region had been devastated by the other smelters and especially by the heap roasting practiced by the Mountain Copper Co., Ltd. Although the country is widely deforested, and the soil badly washed, and, as a whole, not susceptible to agriculture, the farmers in the Sacramento valley, 13 and more miles below, have brought court proceedings to such a point that the smelters can operate only by removing all the suspended matter from their gases, dilute the gases to not to exceed three quarters of 1 per cent by volume of sulphur dioxide, and do no damage.

The Keswick plant of the Mountain Copper Co., Ltd., was erected in 1896 and closed down in 1905 through proceedings of the U. S. Forestry Service. The company has since built a small plant on San Francisco Bay.

The Bully Hill Smelter at Delamar has also been closed by the Forestry Service.

The mammoth plant of the U. S. Smelting, Refining and Mining Co. at Kennet started in 1905. This is a large copper smelter, but the company attempted to treat their gases with a bag house, as is done at lead plants. In July of last year the company put their bag house in successful operation; as discharged, the gases are practically free from solid matter. To accomplish this it was necessary to both neutralize and cool the gases before sending them to the bag house. The neutralization is accomplished by patented process, involving the use of zinc and other metallic oxides. To cool the gases 39 4-ft. cooling pipes, each about 200 ft. long, were built. Besides this heavy expense the woolen bags, alone, in the bag house, cost about \$30,000. As a whole, this plant is a notable success.

At the Balaklala Smelter preliminary experiments were made with bags, also a centrifugal apparatus, and the electrical precipitation. The electrical process was adopted for the full-sized installation. The smelt-

er treats up to 1,000 tons of low-grade copper ore a day, most of this is handled in blast furnaces, the rest in an oil-fired reverberatory. The volume of gases to be treated amounts from 250,000 to 500,000 cu. ft. per minute; in the great flue, 18 x 20 ft., their velocity is from 10 to 20 ft. a second.

After a unit to treat 1 per cent of the gases had been operated the complete amount was put through 9 large units. Each of these flues contains 24 rows of 24 sheet iron electrodes, each 6 ins. wide and 20 ft. long. The discharge electrodes, of asbestos or mica-covered iron wire, come between the plates. In the rectifier building the 2,300-volt line for general use is transformed up to from 25,000 to 30,000 volts and rectified to the intermittent direct current. The dust and fume which falls is shaken from the electrodes and removed with a conveyor. About 120 kw. represents the power required; the cost amounted to somewhat less than \$110,000.

It was found that at this plant the large amounts of zinc oxide in the fume at times robbed it of all conductive property. This made trouble with the discharge electrodes seriously impairing their efficiency. Prof. Cottrell promises to consider this and the electrotechnical details in another paper.

The whole paper is profusely illustrated with reproductions from photographs. Some show the various stacks with and without the preprecipitating devices in operation; others show the flues with the electrodes suspended within, the stream of fume coming from the electrodes, etc. Views of the Mammoth and Balaklala smelters, plans of the flues and electrodes and a group picture of the staff add much to the article. It is pointed out how fruitful for practical metallurgy has been the expenditure for experiment and construction at these two plants. Often litigation has been expensive enough with no practical result; here, the result has been a distinct step in advance, for both methods of treatment will now find place and use in engineering.

Another field of usefulness for the process is mentioned as at the cement works where so much fine dust is given off with the kiln gases. This dust is impalpably fine and given off at a high temperature. In southern California court proceedings have already taken place between the orange growers and one plant which threatens to close the latter. To meet the new conditions precipitation experiments are under way and developing with promise.

The cleaning of iron blast furnace gas is also mentioned. The difficulties here do not appear insurmountable and the matter is being thoroughly tested out.

In the case of the ordinary smoke nuisance it has been found that electrical precipitation acts upon smoke as it does upon the dust and fume; however, it would seem more logical to avoid making the smoke at all.

A note added at the end states that farmers' affi-

days in the case of the Balaklava Smelter showed damage still being done and only 72.8 per cent of the total solids being removed by the apparatus during March and April. A few times during the month of observation the percentage of sulphur dioxide also slightly exceeded the required limits. The result is that the judgment by agreement and stipulation allows the smelter 35 days to run through the ore on hand before closing down until the exact provisions can be met.

The abstractor may add that this was carried out and that the smelter is now idle and the town deserted.

METALLURGY OF PLATINUM.*

By W. GEIBEL.

The Russian ore does not contain pure platinum, but an alloy of six metals of the platinum group. In addition, the crude ore contains some gold, that can be separated by amalgamation, then, iron, and the other heavy accompanying minerals, chromite and magnetic iron.

The separation is done chemically. In Russia two factories in St. Petersburg do this work. The quantity of ore worked upon by them is small, amounting to about 200 kilograms a year, a very small per cent of the total production. By far the largest part is sent abroad. The working up of the ore is almost exclusively in the hands of a few firms in Germany, England and France. America covers most of her large platinum demand by purchase of the pure metal from Europe although some crude American platinum is also worked up.

The separation of the platinum metals from the accompanying minerals is quite simple; on the contrary, the exact separation from one another of the individual metals (of the platinum group), which are so similar in their characteristics, forms one of the hardest tasks of chemistry. And yet it is frequently important that exact separations should be made, for even small admixtures of others may strongly affect the technically important qualities of the entirely pure platinum metal desired; thus a small content of iridium makes platinum considerably harder.

The first task is to separate the most important metal, platinum, from the ore. Then, in addition to this, is the further task of separating from each other the other five metals; for in the course of time important uses have been found for them all. As they are present in the ore only in small quantities, their value is even greater than that of platinum.

The separation can be made by the dry method by a smelting process and by the wet method by dissolving the ore. The dry method was used in the '60's by Deville and Debray, who did great service for the chemistry of platinum metals and the technics of work-

ing them up. As this method does not give a perfect separation, the wet method is now in general favor. The processes employed in this operation generally follow the methods which are used in making an analysis in the laboratory. The details, in which many variations are possible, of course, are the secret of the factories. On the whole they are as follows: By a treatment with aqua regia the largest part of the precious metals is dissolved; the residue contains, besides sand, the alloy osmium-iridium. Strictly speaking, the ore thus consists not of one alloy but of two, crude platinum and osmium-iridium. Both contain all six platinum metals, but in different quantities. The crude platinum contains, besides platinum, principally palladium, rhodium, and iridium, together with small quantities of osmium and ruthenium. The osmium-iridium contains, as the name implies, for the most part osmium and iridium, but it contains the other metals also.

It is comparatively easy to separate osmium and ruthenium. They are distinguished by the fact that they are converted by oxidation into volatile oxygen compounds, which can be removed by distillation. With ammonium chloride, platinum and iridium give compounds with good results, having been applied to most varieties difficult of solution, from which the metals are obtained by ignition.

If the solution of crude platinum in aqua regia with ammonium chloride is omitted, then the precipitate contains, besides platinum and iridium, also considerable quantities of rhodium and palladium, and complicated methods have to be used to obtain the pure metal. On the other hand, the platinum and iridium cannot be fully removed in this way, and the mother liquor of the ammonium chloride precipitation contains in consequence considerable palladium and rhodium, and also small quantities of iridium and platinum, and requires complicated working over.

Whatever progress has been made in the attainment of exact methods of separation in recent years is due to the work which Mylius and Foerster have done in the Physikalisch-technische Reichsanstalt. They found by means of the most exact analytical methods that the purest marketable platinum may contain a most 0.01 per cent of impurities.

By igniting ammonio-platinum chloride we get the platinum as a loose, gray mass, the so-called platinum sponge. From this the compact metal must be produced. The difficulty lies in the fact that platinum melts only at a very high temperature between 1700° and 1800° C. When at the beginning of the nineteenth century the value of platinum for technical uses was discovered, the means of getting such a high temperature were not available. This obstacle was first conquered by melting the platinum with arsenic into an easily fusible alloy, after which the arsenic was roasted out. Later the capacity of the metal for easy welding was utilized. By means of high pressure the sponge was compressed; the cake formed was then ignited and

*From The Pacific Miner; abstract of an article from the Australian Mining Standard for the Transactions of the American Metallurgical Society.

hammered, and in this way the individual particles were welded together.

The temperature necessary for melting platinum is obtained with an oxy-hydrogen blow pipe, or in an electric oven. With the apparatus available to-day we can melt not only platinum, but also the other platinum metals, which, with the exception of palladium, have still higher melting points.

ASSAY METHOD FOR TUNGSTEN.*

By B. M. DIVANI.

When tungsten is in solution in the condition of tungstate, it is possible to precipitate and estimate the tungsten in the form of the trioxide WO_3 . To effect this, all that is needed is to render the solution acidic by means of hydrochloric or nitric or even sulphuric acid.

Tungstic acid is soluble to some slight extent in mineral acids. On that account it is advised in the usual works on analytic chemistry to render the tungstic acid insoluble by repeated evaporation on a hot-water bath the solution to which an excess of acid has been added, after which the dry residue is warmed for some length of time at a temperature of $120^\circ C$.

To avoid these operations, which are rather tedious and time-consuming, the writer recommends for study a method of determining tungsten based on the precipitation of tungstic acid by an excess of a solution of freshly prepared stannous chloride. The stannous chloride precipitates the tungsten from solution in the form of the tungsten oxide W_2O_5 . It is well known that this reaction is quite sensitive.

In order to assure himself to what extent precipitation is complete, and also that the oxide be not reduced during washing, the following experiments were tried by the writer:

Two grams of absolutely pure tungstic acid was dissolved in just enough concentrated ammonia to dissolve that amount of acid, and the volume of this solution was brought up to one liter. Caustic soda or potash might have been chosen just as well, the choice of alkali having no effect on the precision of the operations.

For each test determination, 50 ccms. of this solution were taken, containing therefor 0.1 gm. of tungstic acid. To precipitate the tungsten oxide from this solution, the author added 20 ccms. of a solution of stannous chloride. The latter was so concentrated as to contain 50 gms. of crystal salts per 200 ccms. of concentrated hydrochloric acid. After boiling the mixture for one or two minutes, the precipitate was permitted to settle.

The precipitate was then washed in warm water.

Since the oxide of tungsten is deposited very readily, it is possible to wash the precipitate several times within a few minutes without having the water cool sufficiently to float the precipitate. Under these conditions, the precipitate remains flocculent, and no cloudiness is produced in the water bath. The oxide thus precipitated was calcined and weighed as the trioxide. The results indicated that this method was susceptible of an unusually high degree of accuracy.

In order to be sure that in the calcined residues measured in the above tests no tin was carried along, the following analyses were made: Several of the residues were collected and heated in a current of hydrogen. The production was subjected to hydrochloric acid and filtered. A current of sulphuretted-hydrogen gas was passed through the filtrate. This reaction gave no precipitate. Hence no tin was carried along in the precipitation of tungsten trioxide.

The author also tried some experiments, the aim of which was to ascertain whether the presence of iron affects the accuracy of the method. With this end in view, there were added to a couple of tested samples 10 to 20 ccms. of a solution of ferric chloride, half normal. The results showed that iron produces no effect on the accuracy of the method.

QUICKSILVER PRODUCTION IN THE UNITED STATES IN 1910.

The production of quicksilver in the United States in 1910, as obtained from confidential returns to the U. S. Geological Survey by every producer in the country, was 20,601 flasks of 75 lbs. each. At the average domestic price at San Francisco, \$46.61 a flask, the value was \$958,153. As compared with the production of 1909, which was 21,075 flasks, valued at \$957,859, this shows a decrease in quantity of 474 flasks but an increase in value of \$294. Although the production of California increased in 1910 the output from Oregon decreased to nothing, as that of Arizona did in 1909, the small Nevada production fell off considerably and the output from Texas decreased. In no state, except possibly in Nevada, can an increased output be expected for 1911, the present outlook being for a total production for the United States not exceeding 20,000 flasks. A good domestic demand for quicksilver was noted throughout 1910. The principal uses are for gold mining and placer mining, for the manufacture of vermilion, fulminates, physical instruments and drugs, and for lighting. The use of quicksilver in making the fulminate of percussion caps for igniting powder is increasing in importance probably more than any other use. The consumption for all purposes in 1910 was somewhat greater than the domestic production. Of the 19 mines producing quicksilver in the United States in 1910, 15 are located in California, two in Nevada, and two in Texas.

*From The Pacific Miner; Abstract of an article from the Bulletin Société Chimique de Belgique, for the Transactions of the American Metallurgical Society.



TWO METHODS OF TESTING ASPHALT.

By PROFESSOR S. F. PECKHAM.

At the last meeting of the Institute of Chemical Engineers, at the Hotel Astor in New York, last December, John Puroy Mitchell, who welcomed the members of the Institute to the city, announced, according to the *Times*, that the city had decided to establish and maintain a fully equipped chemical laboratory for the examination of all chemicals purchased by the city. He said that this laboratory would be governed by a Board of Directors, who would be men of high standing in their profession. Manufacturing chemists have been urging the city for some time to establish such a laboratory. They say the present laboratories of the city are not equipped to do the work satisfactorily, and that as a result the city often pays dearly for poor material. With such a laboratory as has been suggested, manufacturers and chemical engineers say that such as a thing as graft or favoritism in the granting of contracts will be impossible.

Of course, I need not say to the members of this Institute that this wished for result will depend entirely upon the Board of Directors and the men selected by them to conduct said laboratory. As an illustration in point, I will relate the following facts:

A short time prior to the meeting of the Institute in December, one of the engineers attached to the Department of Finance of the city brought to me four samples of asphaltic material, two of which he said he wished to compare with the other two. Accompanying these samples was the manuscript of an unsigned report, which had been made upon them, which is as follows:

Analysis of the sample of rock asphalt mined at Asphalt, Edmonson County, Ky., shows that it is asphaltic sand stone, and has the following composition:

Laboratory Test No. K-6687

	Pct.
Bitumen	8.2
Passing 200 mesh sieve.....	5.5
Passing 100 mesh sieve.....	2.3
Passing 80 mesh sieve.....	1.0
Passing 50 mesh sieve.....	24.0
Sand, Passing 40 mesh sieve.....	22.0
Passing 30 mesh sieve.....	29.5
Passing 20 mesh sieve.....	5.5
Passing 10 mesh sieve.....	2.0

Analysis of the sample of Asphalt Mastic manufactured from the above, and known as "Standard Kentucky," resulted as follows:

Laboratory Test No. K-6688

	Pct.
Bitumen	12.3
Passing 200 mesh sieve.....	28.9
Passing 100 mesh sieve.....	8.2
Passing 80 mesh sieve.....	2.1

Mineral, Passing 50 mesh sieve.....	22.5
Aggregate, Passing 40 mesh sieve.....	7.5
Passing 30 mesh sieve.....	9.8
Passing 20 mesh sieve.....	3.1
Passing 10 mesh sieve.....	3.0
Retained on 10 mesh.....	2.6

A 10-lb. weight resting on an area of four square inches of this mastic at a temperature of 90° F., for one hour, caused no appreciable indentation.

Analysis of the sample of Ragusa Rock Asphalt showed that it is an asphaltic limestone of the following composition:

Laboratory Test No. K-6689.

	Pct.
Bitumen	8.2
Limestone	90.6
Sand	1.2

Analysis of the sample of Asphalt Mastic manufactured from the above rock asphalt has the following composition:

Laboratory Test No. K-6690.

	Pct.
Bitumen	16.8
Passing 200 mesh sieve.....	30.2
Passing 100 mesh sieve.....	12.0
Passing 80 mesh sieve.....	2.6
Mineral, Passing 50 mesh sieve.....	12.0
Aggregate, Passing 40 mesh sieve.....	2.5
Passing 30 mesh sieve.....	2.5
Passing 20 mesh sieve.....	4.0
Passing 10 mesh sieve.....	7.4

A ten-lb. weight resting on an area of four square inches of this mastic at a temperature of 90° F. showed no appreciable indentation.

Respectfully,

— — —
CHEMIST.

The engineers, as usual, were in a great hurry, apparently thinking that such examinations could be made in a few hours.

The four samples were labeled as follows:

713A was a sample of Kentucky Rock which I have had in my possession for several years, and was one of a number of samples of such rock that I have examined at intervals.

713B was a sample of Mastic reputed to have been made from Kentucky Rock.

714A was a sample of Ragusa Sicilian Rock Asphalt.

714B was a sample of Mastic reputed to have been made from the Sicilian Rock of which 714A was a sample.

I have examined at different times a number of samples of Kentucky rock reputed to have been taken from this locality, in the area, that is, in which such rock is found. These analyses all showed that the Kentucky rock consists of a very sharp-grained hard sandstone, that was a solid rock before it was saturated with bituminous material. The percentage of bitumen in the different samples varied, as might be expected to result from the varying degree of porosity of the rock. The sandstone, in all cases, however, was a very compact rock after the bitumen was removed. The problem submitted was to determine the identity

or resemblance between 713B and 714B. It was not expected that, inasmuch as the natural Kentucky rock is a sharp-gritted sandstone and the Ragusa rock is a soft chalk, consisting almost entirely of carbonate of lime and magnesia, that any identity could be established between the original Kentucky rock and Ragusa rock from which the mastics were made.

The two mastics, however, were, to be according to the specification, identical or equal to each other. The four samples were subjected to chemical experiments and a physical test.

CHEMICAL EXPERIMENTS.

Portions of the four samples were broken into coarse fragments, care being taken not to pulverize the samples, and weighed in duplicate portions of 5 grams each. These portions did not contain pieces larger than a pea. They were placed in stoppered funnels with balanced filters. The samples were made as nearly alike as possible under the circumstances and they were all treated with petroleum ether out of the same bottle of a specific gravity of 90° Baumé, until the ether passed colorless.

The solutions in petroleum ether were variously colored. 713A, by transmitted light, was *dark red*; by reflected light, *dark brown*.

713B was, by transmitted light, *light red*; by reflected light, a strong *green fluorescence*.

714A was, by reflected light, *brown*; by transmitted light, *red*. 714B was the same.

These different colors indicate that the three numbers, 713A, 714A and 714B, did contain a different bitumen from 713B. 713B must have consisted in large proportions of some sort of petroleum residuum which may or may not have been mixed with other natural bitumens, the proportion of petroleum residuum being sufficient to disguise the other ingredients.

I have examined hundreds of natural bitumens and have never found a specimen of asphaltum or bituminous rock from any part of the world to yield a solution in petroleum ether having a green fluorescence. Solid asphaltum and the bitumen of asphaltic rocks invariably yield a brown solution in petroleum ether.

The exhausted filters were dried and weighed against the balanced filter and then returned to the funnels, where they were exhausted with boiling spirits of turpentine. The residues from the boiling turpentine were treated with 95 per cent alcohol and after being thoroughly exhausted of the turpentine were dried and weighed. The dried filters were then returned to the funnels and exhausted with chloroform.

These different solutions and residues yielded the following percentages:

	713A	713B	714A	714B
Petroleum ether	5.603	8.261	7.189	10.622
Boiling turpentine	1.765	4.389	1.860	6.101
Chloroform	.387	1.272	Zero	Zero

From an inspection of this table it will be observed that the two samples 714A and 714B were exhausted of bitumen by treatment with petroleum ether and spirits of turpentine.

These results indicate two very different varieties of bitumen. The bitumen of the limestone asphalts of Europe is wholly soluble in spirits of turpentine and in this respect is quite different from almost any other variety of bitumen. There was no indication of a mixture of any petroleum residuum in either 714A or 714B.

The following table gives the percentage of the total bitumen found in the four different specimens which yield to a solution of the three solvents:

	713A	713B	714A	714B
Petroleum ether	72.25	59.20	78.75	63.5
Boiling turpentine	22.75	31.65	21.25	36.5
Chloroform	5.00	9.15	Zero	Zero

The total bitumen found in the four samples is as follows:

	713A	713B	714A	714B
	7.755	13.865	9.049	16.723

The total composition of the four specimens is:

713A—Bitumen, 7.755, sand 92.245.

713B—Bitumen 13.865, sand 45.675, carbonate of lime, 40.460

714A—Bitumen, 9.049, carbonate of lime, 90.951

714B—Bitumen 16.723, carbonate of lime and sand, 83.277

This specimen of 713A consisted, like all other samples of Kentucky rock, of bitumen saturating a sandstone hard and sharp-gritted. 713B consisted of bitumen 13.865 per cent, sand 45.675 per cent, pulverized limestone 40.460. These ingredients indicate a material that is practically a street mixture prepared from sand, powdered limestone and bitumen. The bitumen consisting largely or wholly of petroleum residuum.

714A consisted of bitumen 9.049, carbonate of lime 90.951. The carbonate of lime contained, as is usual with chalk formations, a small percentage of grit, which in the unbroken rocks are usually found to be silicious shells. 714B consists of bitumen 16.723, the remainder consisting practically of carbonate lime, but containing a larger proportion of silica than the limestone residuum from 714A.

The excess of silica may have been added, or, as it is more likely, it may have been a constituent of the natural rock from which the mastic was prepared.

These results show, as far as chemical results can show, that 713A was natural Kentucky rock; 713B was a mixture identical with or resembling a street mixture, consisting, as before stated, of sand, pulverized limestone and bitumen that was wholly or largely petroleum residuum. 714A is natural Ragusa bituminous limestone. 714B was a mastic prepared from such limestone with an addition of bitumen that was probably extracted from poor rock out of the same quarry.

PHYSICAL TESTS.

Portions of 713B and 714B were pulverized in the first instance completely and in the second instance only partially, as it was very difficult to reduce the Ragusa mastic into small fragments. These crushed portions were compacted into a mold such as is used in making cement briquettes. One was prepared at laboratory temperature, a second was prepared by warming the mass to a temperature of about 180°. They were allowed to cool and set over night and were

broken the next morning in the cement testing machine, with the following results:

713B cold, broke at 170 pounds.
713B hot, broke at 210 pounds.
714B cold, broke at 176 pounds.
714B hot, broke at 189 pounds.

Both briquettes 713B broke short, while both 714B stretched before breaking in such a manner as to render the exact determination of the point at which they broke difficult. The briquettes made from Ragusa rock were very much more elastic than those made from so-called Kentucky rock. The halves of the broken briquettes were very nearly of the same size. They were exactly alike in shape and were an inch thick, with a superficial area approximating two square inches.

These broken briquettes were crushed in an Olsen machine and were found to sustain the following crushing pressures:

713B cold sustained a crushing pressure of 3,600 pounds.
713B hot sustained a crushing pressure of 3,000 pounds.
714B cold sustained a crushing pressure of 4,200 pounds.
714B hot sustained a crushing pressure of 5,200 pounds.

All of these varied experiments, both chemical and physical, established the fact beyond any question that the mastics were neither identical nor equivalent and that the mastic prepared from Ragusa rock is very greatly superior to the so-called Kentucky rock mastic.

The inadequacy of any proof of identity or equality between these specimens exhibited by the unsigned report needs no farther demonstration than a perusal of the reports of these two materials. The mastic was to be used in one of the city parks for walks, over which, at certain seasons of the year, a constant stream of pedestrians was moving. The fact that a ten-pound weight on a surface of 4 sq. ins. made no impression is ridiculously inadequate. The mastic walks were to be continually used at a temperature often reaching 120° F. in the sun, and subject to the impressions of boot heels, supporting a weight of 10, 15 or 20 times 10 lbs. If the conditions under which this report was made were not as serious as they really are, the report might be denominated a complete farce.

The value of the mine production of gold, silver, copper, and lead in Texas in 1910, according to C. W. Henderson, of the United States Geological Survey, was \$209,061 against \$199,446 in 1909. The quantity of ore treated was 21,917 tons against 21,762 tons in 1909. The production of gold was \$423, against \$314 in 1909; that of silver was 380,322 fine ounces, valued at \$205,374, against 374,444 ounces, valued at \$194,711; that of copper was 3,157 lbs., valued at \$401, against 3,862 lbs., valued at \$502; and that of lead was 65,068 lbs., valued at \$2,863, against 91,140 lbs., valued at \$3,919. The metal production of Texas is therefore mainly in silver and lead, and much the greater part of this is from the well-known Presidio silver-lead mine, in the Shafter district of Presidio County. The output of gold, silver, copper, and lead in Texas is all from trans-Pecos or far western Texas.

SPECIFICATIONS FOR THE PURCHASE OF FUEL OIL.

The U. S. Bureau of Mines, Washington, D. C., has recently issued Technical Paper No. 3, on "Specifications for the Purchase of Fuel Oil for the Government," with directions for sampling oil and natural gas.

General specifications for the purchase of fuel oil are given in the paper as follows:

In determining the award of a contract, consideration will be given to the quality of the fuel offered by the bidders, as well as the price, and should it appear to be to the best interest of the Government to award a contract at a higher price than that named in the lowest bid or bids received, the contract will be so awarded.

Fuel oil should be either a natural homogeneous oil or a homogeneous residue from a natural oil; if the latter, all constituents having a low flash point should have been removed by distillation; it should not be composed of a light oil and a heavy residue mixed in such proportions as to give the density desired.

It should not have been distilled at a temperature high enough to burn it, nor at a temperature so high that flecks of carbonaceous matter began to separate.

It should not flash below 60° C. (140° F.) in a closed Abel-Pensky or Pensky-Martens tester.

Its specific gravity should range from 0.85 to 0.96 at 15° C. (59° F.); the oil should be rejected if its specific gravity is above 0.97 at that temperature.

It should be mobile, free from solid or semi-solid bodies, and should flow readily at ordinary atmospheric temperatures and under a head of 1 ft. of oil, through a 4-in. pipe 10 ft. in length.

It should not congeal nor become too sluggish to flow at 0° C. (32° F.).

It should have a calorific value of not less than 10,000 calories per gram (18,000 British thermal units per pound); 10,250 calories to be the standard. A bonus is to be paid or a penalty deducted, according to the method stated under section 21, as the fuel oil delivered is above or below this standard.

It should be rejected if it contains more than 2 per cent water.

It should be rejected if it contains more than 1 per cent sulphur.

It should not contain more than a trace of sand, clay, or dirt.

Each bidder must submit an accurate statement regarding the fuel oil he proposes to furnish. This statement should show: The commercial name of the oil; the name or designation of the field from which the oil is obtained; whether the oil is a crude oil, a refinery residue, or a distillate; the name and location of the refinery, if the oil has been refined at all.

The fuel oil is to be delivered f. o. b. cars or vessel, according to the manner of shipment, at such places, at such times, and in such quantities as may be required during the fiscal year.

A CONTINUOUS LIQUID EXTRACTOR, AND AN APPARATUS FOR THE MEASUREMENT OF AN EVOLVED GAS.

In a recently issued circular of the United States Bureau of Chemistry Messrs. R. F. Bacon and P. B. Dunbar, Assistant Chemists, Division of Foods, describe two interesting pieces of chemical apparatus. The first of these is an apparatus for the continuous extraction of liquids with immiscible solvents lighter than water, and the second is an apparatus for quantitative reactions which depend on the measurement of an evolved gas.

CONTINUOUS LIQUID EXTRACTION.

This apparatus was designed originally for use in the extraction of lactic acid from ketchup and other fruit products. Its principal advantages are compact-

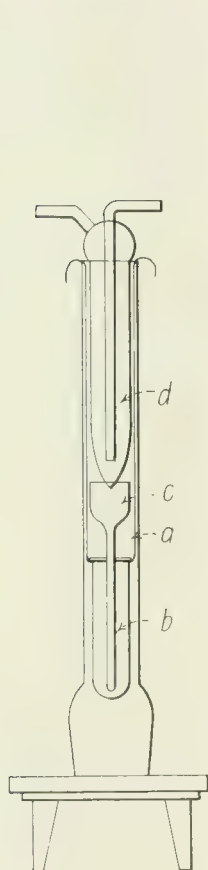


Fig. 1.

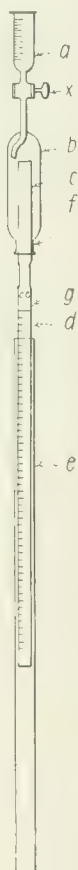


Fig. 2.

ness, the elimination of ground joints and stoppers, and practically complete condensation. Seven of the extractors may easily be placed side by side on a 24-in. hot plate. They have been made at a cost of \$1.50 each, but may be constructed in the laboratory by anyone having ordinary skill in glass blowing.

The apparatus consists of four parts: (1) A jacket flask; (2) an extractor thimble; (3) an ordinary Gooch funnel; (4) a condenser.

(1) The jacket flask (Fig. 1, A) is made of glass tubing 2 ins. in diameter and approximately 1-16 in. thick; it is 20.5 ins. long and is enlarged to a diameter of about 3 inches at its lower or sealed end, as

shown in the illustration. (2) The extraction thimble (Fig. 1, B) is an ordinary test tube having a diameter of 1.5 ins. and a capacity of 100 cc. when filled to within 1.5 ins. of the top. One-fourth of an inch from its top and on opposite sides of the tube are placed two holes about $\frac{1}{4}$ in. in diameter. (3) The Gooch crucible funnel is Fig. 1, C; those used in this laboratory are 8.5 ins. long. When dense liquids are to be extracted it is sometimes necessary to increase the length of the funnel. The lower end of the stem is ground at an angle of 45 degrees. (4) The condenser (Fig. 1, D) is designed to hang loosely in the jacket flask. Its details are evident from the drawing. A simpler form of condenser, which is just as efficient, may be made by sealing one end of a $1\frac{3}{4}$ -in. tube and drawing it to a point. The open end is flared somewhat to permit the tube to hang in the jacket. The condenser tube is closed by a two-holed rubber stopper, through which pass the inflow and outflow tubes.

Operation.—Place from 100 to 150 cc. of ether in the jacket flask (A); put the liquid to be extracted (100 cc.) in the test tube (B), insert the funnel in the same tube and suspend it in the jacket flask, about 3 ins. above the bottom, with a copper wire which passes through the holes in the test tube and is hooked over the rim of the jacket flask. Insert the condenser in the top of the jacket flask. The condensation is usually so perfect that no ether vapors escape into the room. The condensed ether drops from the point of the condenser into the funnel and is carried to the bottom of the test tube whence it flows up through the liquid and overflows at the top.

The efficiency of the extractor may be increased by the use of a glass spiral attached to the stem of the Gooch funnel, as described by Kampf.¹

APPARATUS FOR THE MEASUREMENT OF AN EVOLVED GAS.

The apparatus represented in Fig. 2 was devised to give in a compact and easily manipulated form an apparatus which will allow of the measurement of an evolved gas without first sweeping out all air or other indifferent gases. It consists of a graduated funnel tube (A), a reaction chamber (B), an absorption tube (C), filled with glass beads, a eudiometer (D), and a leveling tube (E). A heating coil may be wrapped around the reaction chamber (B) when desired. The absorption tube (C), which is sealed onto the eudiometer, fits into the reaction chamber (B) by means of a ground joint at (F).

The manner of using the apparatus may best be illustrated by one of the reactions which can be advantageously carried out in it. Spica² estimates citric acid from the carbon monoxid evolved by decomposing this acid with strong sulphuric acid at 100° C. He runs air-free carbon dioxid through a flask containing the citric acid until all air is displaced from the

¹Chem. Ztg., 1910, 34; 1365; Chem. Abst., 1911, 5; 1350.

²Chem. Ztg., 1910, 34; 1141.

apparatus. He then adds concentrated sulphuric acid, heats to 100°C ., drives over the evolved carbon monoxid with a stream of air-free carbon dioxid, and collects the gas in a eudiometer over a sodium hydrate solution. The method as carried out by him is accurate, but requires considerable care and attention to remove the air from the apparatus completely, to insure that all evolved carbon monoxid is swept over into the eudiometer, and to prevent the strong sodium hydrate solution from sucking back into the reaction flask containing carbon dioxid and sulphuric acid. In the apparatus proposed no attempt is made to remove the air before reaction, the volume of evolved gas being simply measured by the increase in volume of the total gases after reaction. Thus the Spica method as carried out in this apparatus is as follows:

Place about 0.2 gram of citric acid in the reaction chamber (B), open the stopcock (X) and bring the liquid (in this case a strong sodium hydrate solution) to the zero mark (G) in the eudiometer tube by raising or lowering the leveling tube (E). The air in the apparatus is then at atmospheric pressure. Close (X). To (A) add about 15 cc. of concentrated sulphuric acid. Lower the leveling tube so that the air in the apparatus is under reduced pressure. By carefully opening the stopcock (X) run in exactly 10 cc. of the sulphuric acid. Close (X) and heat to about 100°C . by means of the heating coil until the reaction is complete. Let the apparatus stand until the absorption of other gases (sulphur dioxid, etc.) is complete and it has reached room temperature. Bring the gas in the apparatus to atmospheric pressure by means of the leveling tube (E). The gas reading in the eudiometer minus 10 cc. (for the added sulphuric acid) equals the volume of the evolved gas at the existing temperature and pressure. The apparatus has been used in this laboratory with accurate results to estimate citric acid by the Spica method and for the estimation of amino acids by the Van Slyke method,³ which depends on the evolution of nitrogen by the action of nitrous acid on the amino acids. The apparatus is readily cleaned by taking it apart at the ground joint (F).

Of Montana's zinc output in 1910, which was 31,638,184 lbs. (figured as spelter), against 9,359,724 lbs. in 1909, the Summit Valley district, in Silver Bow County, produced all but 18,000 lbs. Nearly all of the zinc ore was sent to concentrating plants, and the concentrates produced contained 31,334,204 lbs. of zinc; the crude ore sent to smelters yielded only 303,980 lbs.

The total quantity of ore treated or sold in Idaho in 1910 was 1,786,174 short tons having an average value per ton of \$8.99, an increase of 13,791 tons and of \$0.31 per ton over 1909.

THE ESTIMATION OF ESSENTIAL OILS.*

By CHARLES D. HOWARD.†

During 1908 the writer published a method¹ for the determination of essential oils in extracts and pharmaceutical preparations, involving precipitation and extraction in a Babcock milk bottle, the small quantities of chloroform and ether used being volatilized by rapid evaporation in a water bath. This method was applied to a variety of extracts, including, incidentally, benzaldehyde, although there was no intent to claim that it afforded results of any value in the latter instance.² The method was subsequently criticized by Hortvet and West,³ the authors claiming that the procedure did not serve to eliminate all of the chloroform and that extraction was not complete.

As a result of further experience, it is admitted that this method sometimes affords erratic results in the case of certain oils and that the average worker is liable to encounter some difficulty in securing concordant and accurate figures. It has proved of value, however, in the case of such oils as lemon, orange, wintergreen and peppermint. A method of this character has the advantages of being simple and quickly carried out, requires no large quantity of solvent, and avoids the difficulties in connection with drying and weighing the oil—the latter, in our experience, proving not inconsiderable.

Based upon the conviction of the writer that a much smaller quantity of solvent than that prescribed by Hortvet and West can be made to serve for complete extraction, the following modification was devised and has been used in this laboratory during the past year ties of essential oil preparations. The procedure involves application of the principle, referred to by the writer in his original paper and since confirmed by Hortvet and West,³ that when an ethereal solution of an essential oil is *rapidly* evaporated, no appreciable loss of oil occurs.

Procedure.—Transfer 20 cc. of the extract to a four-ounce separatory funnel; in the case of preparations containing more than five per cent of oil, take but 10 cc. Add 50 cc. of water and (except in the case of oils of the type of cinnamon and clove two drops of strong hydrochloric acid. Shake out with three portions of ether, using 15 cc., 10 cc. and 5 cc. After each extraction except the last, the ether solution may be run out into a small flask, which is kept stoppered if a series of determinations is being run simultaneously. The combined ether extracts are washed once with 10 cc. of ether-saturated water for removal of the bulk of the alcohol, then cautiously transferred to a 10 per cent milk bottle, rinsing the flask and tip of the funnel with an additional 2 or 3 cc. of ether. Attach a bulb tube to the stem of the bottle and connect with a

*From The Journal of Industrial and Engineering Chemistry.

†State Laboratory of Hygiene, Concord, N. H.

¹Jour. Am. Chem. Soc., 30, 608 (1908).

²The Journal of Industrial and Engineering Chemistry, 1, 84 (1909).

³Loc. cit., p. 88.

filter-pump, immerse the bottle in nearly boiling water, start the pump and shake with a gentle rotary motion at first. When all danger of spirting has passed, shake *violently* and toward the last immerse in boiling water for a few seconds, or until the application of a match flame demonstrates the complete elimination of the ether. The removal of most of the latter should require not more than two or three minutes. Finally add cold water and centrifuge. In the case of oils heavier than water, salt solution must be used as the floating agent, except with wintergreen, for which cold sulphuric acid (1:2) may be safely and most conveniently used.

The Northern Steam Boiler Association of Sweden reports trials of peat powder as fuel under steam boilers in the month of February last. For blowing the powder into the combustion chamber under a boiler an electric fan was used for which 1 h. p. was required. The air used was heated in the smoke flues of the furnace to about 200° F. previous to blowing the powder into the furnace, and it was possible to regulate the amount of air and powder with great precision. It was found that 1 ton of coal was equivalent in "heat effect" to 1.397 tons of peat powder, the latter being of poor quality, and to 1.2 tons if better quality had been used. The "heat effect" of the boiler is stated to have amounted to an average of 75.31 per cent when heated by peat powder and 64.17 per cent when heated by coal. This is said to be partly because of the more intimate mixture of the heated air and coal possible when peat powder is employed and the more complete control of the proportions. The cost results found in this experiment in raising the same amount of steam with peat powder and with coal are stated to be as follows: The ton of coal required to raise the required amount for the same amount of steam cost \$3.95, and the 1.214 tons of peat powder required for the same amount of steam cost \$2.76. In using the powder, however, some coal is required for the start, the additional cost being 50 cents, bringing the total cost for the powder to \$3.26. Thus the saving effected in using the powder was 69 cents, or 17 per cent. The experiment was made at a point near the peat bog and the manufactory of the powder and not at a port where coal was cheapest, but the proportions of value were not affected by that.

The value of the mine output of silver, copper, lead and zinc in the Central States for the calendar year 1910, according to B. S. Butler and J. P. Dunlop, of the United States Geological Survey, was \$62,866,732, compared with \$63,997,999 in 1909. The decrease in total value was occasioned by the smaller production of copper in Michigan, which yields practically the entire output of that metal in these states. The production of silver, lead, and zinc increased both in quantity and in value.

A CONVENIENT METHOD OF CHECKING RESULTS.

By R. E. BRADLEY.*

Suppose we have obtained two results, A and B, of which A is the greater, and suppose we do not wish to have them differ by more than 0.2 per cent.

Given $A > B$

$$\begin{aligned} &< \\ A - B &= .002 A \\ A - B &< \\ \hline &= .002 \\ A \\ 1 - B &< \\ \hline &= .002 \\ A \\ B &< \\ \hline &= .002 - 1 \\ A \\ B &> \\ \hline &= 1 - .002 = 0.998 \\ A \end{aligned}$$

$$\begin{aligned} &> \quad - \\ \text{Log. } B - \text{log. } A &= \text{log. } .998 = 1 + .9991 = 0.0009. \end{aligned}$$

$$\begin{aligned} &< \\ \text{Log. } A - \text{log. } B &= .0009. \end{aligned}$$

This, interpreted, means that if an accuracy of 0.2 per cent is desired, the logarithms of the two results cannot vary more than 0.0009, and further, that this will be the case no matter how large A and B may be. This is often convenient in noticing the divergence of results while still in the logarithmic state.

A little consideration will also show that for a divergence of 0.1 per cent, the allowable logarithmic divergence will be 0.0004, 0.3 per cent = 0.0013, 0.4 per cent = 0.0017, 0.5 per cent = .0022, etc.

The production of silver in Montana in 1910 was 12,162,857 fine ounces, against 12,378,714 ounces in 1909. Nevada now takes the leading place in silver output in the United States, which Montana held in the years 1909 and 1908. Of Montana's 1910 output of silver, 9,153,590 ounces, or 75 per cent, came from copper ores, and 2,139,465 ounces, or 17.6 per cent, from silicious ores. Bullion at gold and silver mills produced only 130,900 ounces, while concentrates produced 8,208,644 ounces and crude ore shipped to smelters contained 3,766,375 ounces. Silver Bow County produced 10,400,840 ounces, or 85.5 per cent of the state's production, compared with 10,609,328 ounces in 1909.

The production of lead in Montana increased from 2,983,430 lbs. in 1909 to 4,106,292 lbs. in 1910. Nearly all the metal-mining counties in Montana made a small yield of lead, but Jefferson County, with a production of 1,975,492 lbs. in 1910, against 1,568,696 lbs. in 1909, produced nearly half of the state output in both years.

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NOTES AND COMMENTS.

The Ethics of the Practice of Chemistry.

EDITOR CHEMICAL ENGINEER:

Sir: Your recent editorial on this subject leads me to bring up a point in connection with it which heretofore has been ignored or overlooked by those favoring state regulation of the same sort as is now in force with some other professions, notably law and medicine. That point is, that if inadequately prepared analysts are legally debarred from practice it will completely shut us off from our source of supply. For the young college graduate in chemistry cannot by any stretch of the imagination be considered adequately prepared in analytical chemistry. As a general scientist full of the promise of future great things in some particular line of engineering effort he is entitled to be considered with the most profound respect. But as an analytical chemist he is at graduation obviously nothing more nor less than a somewhat mitigated humbug. Imagine the same system of preparation and performance in vogue in other professions! Suppose a doctor whose college training has been mostly

engineering whose after study has been almost entirely engineering and whose highest ambition is to get from the practice of medicine to that of engineering at the very first opportunity!

Your correspondent's four years' college course was almost exclusively chemical, at a greater money cost and at the sacrifice of a degree, no degrees being given for special self-selected courses. At the end of his junior year he refused a position as furnace analyst. At the beginning of his senior year declined another as works analyst (positions were plentiful in those good old days). Towards the close of his first term Senior did accept one, but resolutely stood his employers off for six months until he had finished his self-appointed analytical course. But in all this he wishes to confess he was impelled by nothing heroic. On the contrary, his guiding motive was pure cowardice—he feared to face his future employer poorly prepared (but had no fear on the other hand of missing his degree in the regular course, his grades ranging 95 to 97, and no conditions against him when he changed to special in his sophomore year), and the young chemical engineer taking up with easy assurance his more or less strange and unfamiliar analytical duties at the works has always been to him an awe-compelling spectacle.

Now he has the effrontery to believe that for an analyst with no ulterior engineering aims, the training he mapped out for himself was far better than that mapped out for him by his college, and he has always hoped that the four year's exclusively chemical course that he was compelled to sacrifice a degree for would eventually be supplied by our colleges without such a sacrifice. But although over a quarter of a century's fruitless waiting has dimmed this hope, never very bright, yet he must still insist as earnestly as ever that no analyst should attempt practice without a preliminary four years' college course exclusively chemical.

Now, then, believing this, should he agitate for the legal requirement of such a course? Heaven forbid! The employer has a right to get his analytical work done by anyone—a coal heaver if he wishes to do so, and the principal relief that your correspondent experiences in writing this letter is the relief it affords him to protest against the silly and vicious spirit of paternalism everywhere rampant in this land of the free. The state has the right (it is its business, in fact) to brand the incompetent analyst as such, but has no right to say that the individual may not nevertheless, employ him if he wishes to do so. This unceasing warfare against individualism everywhere, and at all times in evidence, is excusable in the ignorant and unthinking, and we must expect it, but we find the educated even more inclined than the ignorant to fall in with the miserable instinct, always manifest in a majority to decide for the individual what the individual should be permitted to decide for himself, and the petty disposition to deny to other the cakes and ale it does not care for itself. In the great (small in a sense) local option wave for instance, what chemist has not manfully voted his conviction that the indi-

vidual cannot be trusted to decide for himself whether or not a glass of beer is good for him (although if the state interfered with his two cups of black coffee for breakfast every morning he would consider it a singularly impertinent and outrageous performance), or in the great pure food crusade, what chemist is satisfied with Dr. Wiley's magnificent and legitimate accomplishments as they stand, and would not vote, if he had the chance, to go further and prohibit the cheap and unscrupulous dealer instead of merely showing him up? Ignoring the sacred right of the individual under the constitution to make an ass of himself in any way that he sees fit (provided he does not hurt others) and eat benzoate of soda when, where and how he likes. Overlooking also the obvious fact that if a man cannot afford the pure article (as may well happen these days of heavy costs and light pay), it is for him and not for the state to decide whether or not he shall take his chances with the benzoated one. Undoubtedly a market entirely bare of "bum" chemists, benzoated food products, and the like would be a grand good thing, but freedom is a still better thing—not the freedom to deceive, rob and cheat, and herein, and herein only, lies the state's opportunity and its duty, to safeguard, it is true, not only the individual, but his rights as well, and the prohibitionists and paternalists of all shades and classes are seeking to get the government to do to a certain extent the very thing it was instituted to prevent.

In this matter of the ethics of the practice of chemistry, let us not jump into a narrow-minded and petty paternalism because everybody else is doing it.

GEORGE AUCHY.

At the recent general meeting of the United Malaysian Rubber Co. in London a resolution was passed to increase the capitalization from \$10,000,000 to \$11,000,000. The company has constructed two large factories, one at Karimon and one at Goebilt, on the former of which \$600,000 has already been spent. It is estimated that about \$350,000 will be needed for additional works on the Goebilt factory. When these two factories are completed and in working order, the former will have a capacity of 30,000 lbs. of rubber per day, and the latter a capacity of 10,000 lbs. per day.

A large glass factory recently built in Damascus to utilize the sand in that locality has not proved a success, owing to the impurity of the native sand. The factory will therefore have to depend for its raw material on sand imported from Germany. Coal is very expensive in Damascus, as most of it has to be imported.

The zinc production of Idaho was 4,252,104 lbs. greater in 1910 than in 1909. With the exception of a small quantity of zinc produced in Blaine County the output was derived from concentrates shipped from the Success mine, in Shoshone County.

STANDARDIZATION OF TOOL STEEL.

In a paper on the standardization of tool steel read at the last annual convention of the Railway Tool Foremen's Association Mr. Henry Otto of the Atchison, Topeka & Santa Fe Ry. brought out the following points:

"I wish to call your attention to the necessity of standardizing carbon and high-speed steel for tool work. We all know that there is a great liability of the purchasing agents being deceived in purchasing steel for tool work by the low prices quoted by manufacturers, more attention being paid to the purchase price, rather than to the quality of the material. Steel can be purchased at a very low cost which would not be fit for the work for which it was intended, in which event, the breakage of tools would be so great that the users would have been 50 per cent ahead if they had paid twice as much for a better steel. On the Santa Fe Railroad, we manufacture all the tools used on the whole system at the Topeka shops, thereby keeping all tools standard. If we learn of a better tool on the market, we make a special effort to get it, test it, and if found to be superior to the ones which are in use, we adopt them at once.

"We all know that we cannot use the same kind of carbon or high-speed steel for the same class of work, and we also know that it is impossible to use steel containing the same percentage of carbon for all kinds of tools. We cannot make a shear blade or a frame reamer from the same grade of steel, and if we were able to adopt a standard of this class of work, we would be able to make a very large saving during the course of a year. If each one of us were to make a study of the subject and arrange to discuss it at our next meeting, we would be able to adopt standards of high-speed and carbon steel for each particular tool used in railroad shops."

In the discussion that followed the reading of the paper, Mr. A. M. Roberts, of the Bessemer & Lake Erie Railroad, Greenville, Pa., stated:

"I have not found any tools that I felt afraid of making of high-speed steel. We do not make a ball reamer or a radial reamer of any kind except of high-speed steel. We use high-speed steel for most of the taper reamers and taps. We use milling cutters of high-speed steel, and we have milled a crucible stay-bolt tap about 30 inches long in 15 minutes, provided nothing went wrong. It almost looks impossible, but it is a fact, and if we were in a hurry or holding up a job we could do a little better. The solid tap is almost a thing of the past with us. We cannot afford to put in time making solid taps. They crook and it costs money to straighten them. There are some things in coke and coal that are detrimental when used in connection with the tempering of steel. The electric furnace is ideal for this purpose and gives an accurate heat. The trouble has been that the men handling the steel have not studied it sufficiently to keep up with the progress of its development. Get your heats

right and use a proper grade of steel for the work, giving the right temper and you will not have trouble. The class of men who use reamers are not mechanics, therefore there will be some breakage, but if we get the full efficiency of the steel, hardened in the proper manner, we will have less breakage than if we guessed at it and hardened haphazardly."

A NEW WATER-SOFTENING PROCESS.

The following description of a new water-softening process being introduced in England is taken from *Engineering* of London:

It is probable that few people, if asked to describe the nature of "zeolites" could do so; indeed, we doubt very much whether many persons outside the ranks of geologists have ever heard of them. Yet they are distributed pretty freely among the older rocks of the earth's surface, though not entering into the constitution of the rocks themselves. They are found in a crystalline form in amygdaloidal fissures or cavities of trap or plutonic rocks, where they have apparently been deposited from water which has percolated into the cavities, thus probably being products of decomposing nepheline, or felspar, of hydrated felspar themselves. They present a great variety of development, and are mostly white and glassy in appearance. Briefly, they are hydrated silicates of alkalies, generally with silicates of aluminum, and seldom containing magnesium. The water is, we believe, generally regarded as basic, in union with silica.

Owing to the great number of forms that constitute this group of minerals, and their varying chemical constitution, it is quite out of the question to discuss them fully here, nor is it necessary; all we need say is that they are composed generally of varying quantities of silica, alumina, lime, soda, potash and water, the silica always largely predominating in all forms, though the other constituents we have named are not necessarily found in all "zeolites." As an example of one form we may take analcime, which is composed of 54.5 per cent of silica, 23.3 per cent of alumina, 14.1 per cent of soda, and 8.2 per cent of water. These zeolites have, in contradistinction to other silicates occurring in nature, the property of being soluble, and they also decompose in dilute acids. They have also the very important property of being able to exchange their bases for others.

Having said this much with regard to the zeolites found in nature, we come to the connection which the property they possess of exchanging their bases for others has with the subject of our article. One would not, on first thought, see much connection between zeolites and the softening of water, but it has been found that when hard water is allowed to filter slowly through layers of these hydrated silicates of alkalies the lime in the water changes place with the soda in the filtering medium, and the water passes out softened; and this fact has been the means in Germany of

instigating a series of experiments during the last two years, which have proved of great value and which have shown that these zeolites can be produced artificially, with the result that the substance is now made much more regular in its composition and freer from impurities than that found in the state of nature. To these artificial zeolites has been given the name of "permutit," which in a moist condition is of a granular flaky form, and has a luster like that of mother-of-pearl. It has a high porosity, and in the dry state readily absorbs about 50 per cent of water. The details of its manufacture cannot be gone into here, but we may say that it is obtained by fusing together felspar, kaolin, clay and soda in definite proportions, the resultant material being lixivated with hot water, when permutit is left as a residue. The granular material is freed as much as possible from the final alkaline lye by washing and centrifugal action. The empirical formula of an artificial zeolite is approximately $2 \text{SiO}_2, \text{Al}_2\text{O}_3, \text{Na}_2\text{O}, 6 \text{H}_2\text{O}$.

Now, the use of this material for softening water may be briefly as follows: As is well known, the total hardness of water may consist of temporary hardness or of permanent hardness, or of the two combined, the former being caused by calcium and magnesium carbonates, and the latter by other salts of lime and magnesia. Boiling at atmospheric pressure precipitates the carbonates and the magnesia, but not the salts forming permanent hardness. Now, as is well known, in the commercial processes at the present time sodium carbonate is added to water as a means of precipitating the hardening constituents of the water, there being an exchange of bases between the lime and manganese and the sodium carbonate. In like manner, if hard water be allowed to filter slowly (the slower the better) through layers of permutit, there is likewise an exchange of bases, the lime in the water taking the place of the soda in the permutit, one molecule of calcium bicarbonate being converted (in the case of temporary hardness) into two molecules of sodium bicarbonate, which latter remains in the water, being very soluble.

The question that will, of course, at once arise in the mind is, how long will the permutit in the filter retain this power? The answer is that it will only retain it so long as any soda remains to exchange with the lime in the water, and here really comes the most interesting part of the process. Permutit suffers practically no loss during use, but when a certain amount of water has been passed through it, its softening powers disappear, but they can easily be restored, and the material can be used over and over again practically indefinitely.

The power of regeneration appears to be the chief novelty of the process; but before it is carried out a few essential points must be attended to. In the first place, the filter must be cleaned. Experience shows that filtration is most effective from the top to the bottom, and that, therefore, cleaning should take place

in the reverse direction—namely, from the bottom to the top—so as to loosen the mass and remove any air that has collected in the material, soft water, if possible, being used for the purpose. After washing, the permutit is regenerated by a solution of common salt, the solution generally being of 10 per cent strength. Previous to regeneration all water is removed from the filter down to the layer of permutit, after which the salt solution is introduced and allowed to flow slowly through the filter from four to five hours. In addition to this, the brine is allowed to stand for a further four or five hours, just covering the layer of permutit, after which the filter is filled with water from the top, and an outlet cock at the bottom is opened for 20 or 30 minutes, or until the water no longer shows any hardness with ammonium oxalate or with soap solution.

The chemical reaction that takes place during regeneration consists in an interchange between the soda in the sodium chloride and the lime in the permutit (derived from the water which it has softened), calcium chloride remaining in solution in the regeneration water. It may be stated that in practice the most suitable rate for the water that requires softening to pass through the layers of permutit has been found to be from 13 ft. to 17 ft. per hour. The permutit is not only active at the surface, but also in the interior, in consequence of its porosity.

From information supplied from Germany it appears that filters using these artificial zeolites have been in practical use during the last two years, treating water for a variety of purposes, including use in boilers and for washing fine textile goods, and a plant has recently been installed in this country for the latter purpose. The results, so far as we are able to gather, have been entirely satisfactory. The treated water is, we understand, particularly well adapted for textile industries, as it can be reduced in hardness practically to zero. In one case where a permutit filter was supplied to a steam laundry in Berlin more than two years ago, it is stated that in about nine months 1,174,800 gallons of water passed through the filter, which volume was completely softened without any apparent loss of permutit. The charge of permutit was about half a ton, and about 1,050 gallons of water passed over each pound of the material. In this case about 44,000 gallons of water per week were used, and regeneration was carried out twice per week, 88 pounds of salt being used per regeneration, although it is stated that it need not have been done quite so often. The cost of regeneration naturally depends on the price of salt, and on the degrees of hardness in the water. It is stated that water having 53° of hardness has been reduced to 3.7° by the process.

All reports of this new method of softening water speak very favorably of it, but the future only can show whether it can stand the continued test of practical application in a variety of ways. It is, we understand, being now introduced into England by

Water-Softeners, Limited, 20 Copthall avenue, Throgmorton street, E. C., and its course will no doubt be followed with much interest. We have not ourselves seen it at work on a large scale, but have had the opportunity of inspecting what might be called a small domestic installation, which was treating water of about 20° total hardness. We tested the water after treatment, and found it to be about 3° of hardness; but we understood at the time that the filter required regeneration, and that usually the hardness after treatment was almost zero. Specially prepared permutit can also be used for removing iron and manganese from water, but this is a matter which we need not go into here.

RULES FOR THE CONTRIBUTION OF PAPERS FOR THE 8TH INTERNATIONAL CONGRESS OF APPLIED CHEMISTRY.

The officers in charge of the 8th International Congress of Applied Chemistry, to be held in September, 1912, have issued the following bulletin for the guidance of prospective contributors of papers and the like to the Congress.

The following summary of suggested requirements and mutual obligations between the Congress and prospective authors of contributions thereto is based upon the Constitution and By-Laws and the Tentative Rules on Papers and Publications and on Sectional Procedure, all of which were first published in the Preliminary Announcement dated March 6, 1911. This announcement, translated into four different languages, has not only been widely distributed throughout the entire world by the Congress itself at considerable expense, but has also been very widely noticed in the scientific, technical and trade press of the world. In distributing this publication of the Constitution and Tentative Rules, a request was always made that criticisms of the latter be sent to the secretary of this Congress. In the more than five months that have since elapsed, very few such criticisms have been received, and this is regarded as tantamount to an acquiescence in those Tentative Rules on the part of all who may be contemplating making contributions in the shape of papers and the like.

This summary is compiled by the Congress, in response to numerous requests for such information, in a concise and compact form. Criticisms thereof, when accompanied by suggested remedies, will be gladly received by the secretary of the Congress at 25 Broad St., New York City.

It is contemplated to announce the final and definite rules governing the subject matter of this summary during December, 1911; criticisms received before December 1, 1911, will be duly considered; criticisms received after December 1, 1911, run the risk of not being considered at all.

The final rules so adopted will be published in the four official languages of the Congress, and will then

be binding on all contributors to the Congress, and all papers and the like offered to the Congress must be offered as subject to those rules; any offer of a paper is a tacit agreement to abide by those rules.

In order that there may be no misunderstandings or disappointments, all prospective authors and all other persons interested in this subject should carefully read the rules that follow, and submit in writing any criticisms they may have to offer, together with suggested remedies for such criticisms.

1. All papers must be in duplicate and legibly written, preferably typewritten.

2. Each sheet must be written on one side only and *not* on both sides.

3. Each paper must be accompanied by an abstract thereof, also legibly written, preferably typewritten, and which must be in duplicate.

4. Papers and their abstracts, both in duplicate, must be in the hands of the American Committee not later than June 30, 1912. All papers received prior to that time and accepted will be printed in their respective Sectional Volumes and distributed to such of the attending members of the Congress as may desire them at or before the opening of the Congress. Papers received after that time, if accepted, will be printed, but may appear in an appendix which may or may not be ready by the opening of the Congress; the Congress cannot then undertake to print them along with the papers of those sections to which they may be assigned.

5. All papers or like contributions must be as concise as possible and must contain the full name and postoffice address of their respective authors; further, what number, if any, of reprints of the paper or like contribution is desired.

6. Papers or other like contributions must be original and not elsewhere read or published, nor contributed or offered to any other society, association or publication for presentation or publication. The offer of any paper to the Congress is a tacit and understood agreement to the foregoing requirement. No paper should deal with historical matter any more than is needful for a proper understanding of the new subject matter presented, which subject matter, as far as practicable, should be of a date subsequent to June 1909, the date of the Seventh International Congress of Applied Chemistry, except by special request. Non-conformity to this requirement may be a reason for rejection; a remedy herefor is offered in paragraph 18.

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and proceedings be not ready for distribution by that date, authors of all papers received and accepted after June 30, 1912, may then publish in any journal or publication that they may elect. This refers only to the report and proceedings bound in paper; members desiring cloth-bound sets can obtain them at an advanced charge over the \$5.00 membership fee; such advanced charge will be announced later, but will probably be \$2.50; delivery of these cloth-bound sets will be about 90 days later than of the paper-bound sets. Authors of papers received before the close of June 30, 1912, may publish those papers in any publication they may elect after the paper is read or after the Congress has adjourned.

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A pamphlet containing a list of topics of each section as made up June 12, 1911, has been printed and widely circulated; copies thereof, as well as additional copies of this circular, can be obtained upon request from Bernhard C. Hesse, Secretary of the Congress, at 25 Broad St., New York City.

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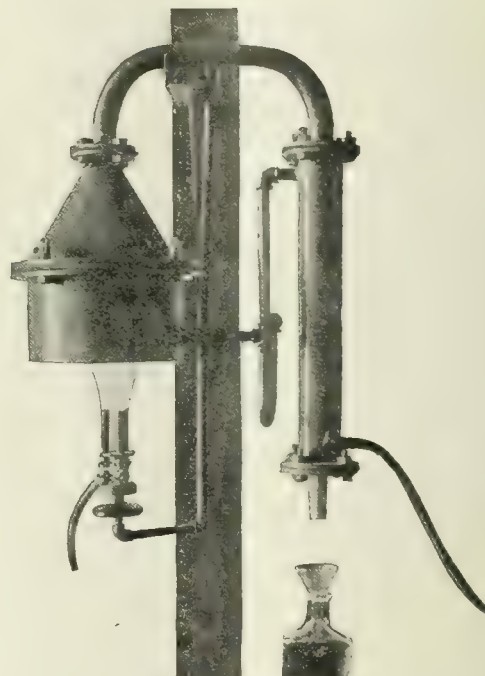
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No. 5.

BALANCE OF COURSES IN CHEMICAL ENGINEERING.*

By CHARLES H. BENJAMIN.†

INTRODUCTION.

Chemical engineering as a distinct profession is growing and its development, although steady, is slow. The pressure for the introduction of separate courses in this branch of engineering has come from without rather than from within. Professors, whether of chemistry or of engineering, have been inclined to look askance on this newcomer as the offspring of an unholy alliance, but the growing demand for men who possess a working knowledge of both professions is gradually reconciling them to the new arrangement.

In studying the development of chemical engineering, we have but to hark back twenty years and read the description in college catalogs of the courses in electrical engineering as they then existed.

The increasing use of electricity in the production of light, heat and power and in telephone service brought with it a demand for men who combined engineering training with a knowledge of the new agent. This demand received at first a half-hearted consideration from the faculties of our engineering schools. Perhaps the professor of physics took charge of the new department, bought a dynamo or two and drafted a little engineering on the science course. More frequently the professor of mechanical engineering assumed a double title and brushed up his knowledge of volts and amperes.

Gradually the warring elements were brought into harmonious relations, the useful was retained, the useless dropped, and under the administration of an electrical engineer, the course attained the place and the dignity which it deserved.

In a somewhat similar way, new enterprises and new combinations are seeking for men who have both chemistry and engineering, and these men will be called in the future "chemical engineers."

The professor of chemistry and the professor of mechanical engineering are just now a trifle at sea as to the parentage of this last addition to the college fam-

ily, but we have a right to infer from past history that their differences will be adjusted, that the right balance will be established and that under the leadership of a chemical engineer, the new course will take its rightful place among its brethren.

DEFINITION.

The chemical engineer may perhaps be described as a man who is competent to so superintend a manufacturing plant as to insure both the chemical purity of the product and economy in the mechanical operation of the factory. As a natural result of conservation and concentration, establishments are now built and equipped for the manufacture on a large scale of various products used in the arts, which were formerly made in smaller quantities at a much greater unit cost. Paints and oils, varnishes, acids, petroleum products, soap, sugar, starch, glass, pottery, explosives, paper, cement, bleaching powders, nitrates—are examples of products requiring for their economical manufacture, knowledge of both mechanical and chemical processes.

Some companies have attempted to solve the problem by employing two men, an engineer and a chemist, and trusting to their joint efforts. As its best, dual control is unsatisfactory and inefficient, leading to divergence and misunderstanding. The one man should rule and he should be capable of meeting all problems that arise, whether chemical or mechanical.

HOW TAUGHT.

There have been and still are marked differences of opinion among educators as to the methods of teaching chemical engineers and as to the relative amounts of engineering and chemistry to be put in the curriculum. Naturally, when the department has been an outgrowth or development of the course in chemistry, that science has received the more attention. In other colleges, the addition of more chemistry to the course in mechanical engineering has left the greater weight on the engineering side.

The accompanying tables and diagrams show the distribution of work by hours in the chemical engineering courses of eight representative institutions. These figures were originally compiled from catalogues and in all cases but one were referred to the professors in the various colleges for correction. They probably represent closely enough for our purpose the relative times assigned to the different branches.

*Paper presented at the 19th Annual Meeting of the Society for the Promotion of Engineering Education at Pittsburgh, Pa., June 27, 28 and 29, 1911.

†Dean of the Schools of Engineering, Purdue University.

For convenience of comparison, the subjects have been arranged in four groups: (1) Letters, comprising all language work, economics and history; (2) science, embracing mathematics, physics and geology; (3) engineering, covering all technical subjects not

do the totals differ considerably but the percentages of subjects vary widely.

Examining the tables and diagrams somewhat in detail, we find that Michigan differs from the other three in Table III by having less engineering, and reference to Table I shows this deficiency to be due to a relatively light class room schedule in mechanics and mechanical engineering. Wisconsin is noticeable for the small amount of time in letters. A few hours taken from engineering and added to letters would restore the balance.

Illinois has relatively less engineering and more chemistry than the other institutions, while the total number of hours is less than the average. Reference to Table II shows this to be due to the almost entire absence of mechanical and electrical engineering.

The course at Pennsylvania State College is marked by a relatively large amount of letters and a small amount of chemistry, in other words, the course is broader and less technical than the others.

The two last institutions on the list, Pennsylvania and Columbia, have apparently much heavier schedules than is usual, averaging thirty-three per cent greater than the 145 hours which represents the mean weight of the courses in Table I.

Comparisons of total weight are, however, somewhat uncertain, owing to the different ratios used in determining credit hours at the various institutions.

A peculiarity of the course at Columbia is the omis-

TABLE I.—CHEMICAL ENGINEERING. ACTUAL HOURS PER WEEK.

	Mich.		Mass.		Purdue.		Wis.	
	Rec.	Lab.	Rec.	Lab.	Rec.	Lab.	Rec.	Lab.
English	4	..	8	..	9	..	6	..
Language	16	..	8	..	16	..	8	..
Economics	..	..	8	..	6	..	..	..
Letters	20	..	24	..	31	..	14	..
Mathematics	18	..	14	..	22	..	19	..
Physics	8	6	11	6	9	4	8	8
General science.....	..	..	..	..	..	4	..	..
Science	26	6	25	6	31	8	27	8
Drawing	12	..	32	..	24	..	18	..
Shop	12	..	2	..	12	..	10	..
Mechanics	4	..	8	..	2	13	..	..
Mechanical Engineer- ing	4	6	9	2	8	3	8	19
Electrical Engineer- ing	2	6	2	3	6	6	4	8
Civil engineering.....	2	6	..	..	..	..	..	..
Engineering	12	42	19	39	22	47	25	25
General chemistry....	4	..	7	8	4	6	4	8
Qualitative analysis...	4	12	..	8	4	12	2	8
Quantitative analysis...	4	18	..	21	4	12	2	8
Organic chemistry....	..	..	7	11	4	12	4	12
Industrial chemistry...	12	21	6	12	8	24	11	18
Inorganic chemistry...	..	..	2	..	..	..	..	..
Theoretical chemistry...	..	..	6	6	..	..	..	..
Chemistry	29	51	28	66	24	66	23	54
Electives theses.....	5	12	..	..	..	..	4	24
Totals	92	111	96	111	108	121	93	141

chemical in character; (4) chemistry, embracing all chemical subjects whether theoretical or technical.

The exact arrangement and definition of subjects in the chemical group is probably not always correct but as the comparison is to be made by groups, this is not important.

The tables show the actual or clock-hours per week for the entire course added together.

Diagram 1 represents the times of the various groups graphically, the upper line in each case being class room work and the lower line laboratory, shop or drafting room time.

In Diagram 2, the two sets of hours have been added by reducing to credit hours. Each class room hour is rated as one credit hour and two and one-half laboratory hours are allowed the same weight. This is the ratio obtaining at Purdue University and is about an average of those in common use.

Whenever the hours of chemistry are divided in two parts, the right hand portion represents elective work and is not necessarily chemistry.

In Diagram 2, the first four lines represent the courses at present in the Massachusetts Institute of Technology and in the universities of Michigan, Wisconsin and Purdue. These are somewhat similar in the balance of subjects and perhaps represent a happy mean. The percentages of total time are shown in Table IV.

The courses represented by the last four lines in Diagram 2 can not be so easily arranged. Not only

TABLE II.—CHEMICAL ENGINEERING. ACTUAL HOURS PER WEEK.

	University			
	Illinois.		Pa. State.	
	Rec.	Lab.	Rec.	Lab.
English	6	..	13	..
Language	8	..	15	..
Economics	6	..	14	..
Letters	20	..	42	..
Mathematics	15	..	20	..
Physics	4	10	7	4
General science....	4	6	..	2
Science	23	16	27	4
Drawing	..	..	2	17
Shop	..	..	7	12
Mechanics	8	3	7	17
Mechanical engi- neering	3	6	12	19
Electrical engineer- ing	..	..	6	15
Civil engineering...	..	..	..	9
Engineering	11	9	27	43
General chemistry...	5	8	6.5	5
Qualitative analysis	1	7	..	7.5
Quantitative analy- sis	4	20	2	16.5
Organic chemistry...	6	12	2	8
Indus. chemistry...	11	30	11	36
Inorganic chemistry...	..	..	..	7
Theoret. chemistry...	..	..	..	2
Chemistry	30	77	21.5	65
Elective theses...	2	4	..	..
Totals	86	106	117.5	112

sion of all work in language or economics, the schedule being entirely of a technical character. The wisdom of this is doubtful. It is but fair to say that a collegiate preparation is recommended and that the regu-

lar requirements for admission to the school include considerable language and history.

In institutions for the Middle West, it is necessary for a well-balanced education that literary training should form a considerable part of the technical courses.

Cornell, Worcester and Minnesota have not been included in the list. At Cornell there is no course in chemical engineering as such. A special course in chemistry allows the student to take a small amount

or college of engineering. On the other hand, it is usually administered by a chemist. At Michigan, Wisconsin and Columbia he is to all intents and purposes a chemical engineer, if we may judge by his title.

CONCLUSIONS.

The chemical engineer occupies a different field from that of the analytical chemist and requires a different training. The analytical chemist, as employed for instance, in the manufacture of iron and steel, is usually a subordinate; he examines the product and determines its good or bad characteristics but has little or nothing to do with the management of the business or the superintendence of the processes of manufacture.

On the other hand, the chemical engineer in a cement or an acid factory may well be the superintend-

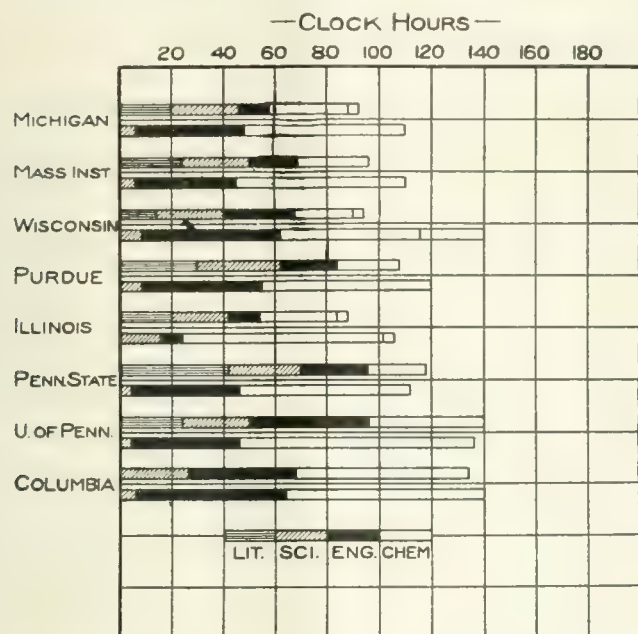


Diagram 1.

of engineering work, amounting to about 20 credit hours for the entire course. Similarly, students in mechanical engineering may take certain electives in chemistry covering 10 or 12 credit hours.

At Worcester, a fifth year, leading to the degree of chemical engineer is added to the regular course in chemistry and is made up almost entirely of mechanical and electrical engineering subjects.

At Minnesota there is a five-year course in applied chemistry.

TABLE III.

	Letters.	Science.	Eng'g.	Chem.
Michigan	11.7	20.6	21.3	43.1
Massachusetts Institute.....	17.1	19.3	25.0	38.6
Wisconsin	9.4	20.0	31.3	39.3
Purdue	19.9	21.8	26.2	32.1
Average	15.3	20.4	25.9	38.4
Or, in round numbers.....	15	20	25	40

The work at the three institutions last named can not consistently be compared with the four-year courses shown in the tables.

ADMINISTRATION.

The location and administration of courses in chemical engineering vary so widely that no general statement can be formulated as may be seen by reference to Table V.

The tendency seems to be to regard this course as belonging with the engineering courses in the school

TABLE IV.

	Letters.	Science.	Eng'g.	Chem.
Illinois	15.5	22.5	11.6	50.4
Pennsylvania State.....	25.8	17.8	27.0	29.4
University of Pennsylvania....	12.3	14.8	31.8	41.0
Columbia	15.3	33.6	51.1	

ent or works manager and must be competent to direct the mechanical as well as the chemical processes of

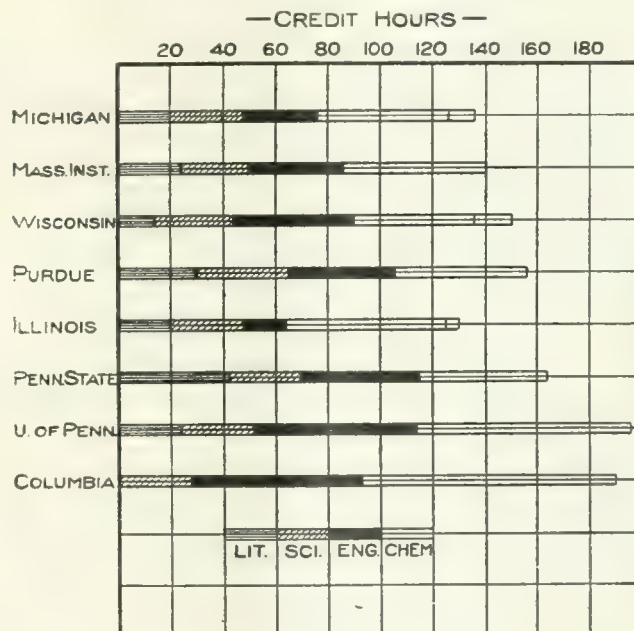


Diagram 2.

manufacture, to buy and maintain machinery and to hire and discharge men. He must then have a fair

TABLE V.

Institution.	School or College	Course in Chemical Engineering.	Administrator.
Michigan.....	Engineering.		Professor of chemical engineering.
Mass. Institute....	Chemistry.		Professor of chemistry.
Wisconsin.....	Engineering.		Professor of applied electro-chemistry.
Purdue.....	Engineering.		Professor of chemistry.
Illinois.....	Science.		Professor of chemistry.
Pa. State.....	Natural Science.		Professor of chemistry.
University of Pa..	Engineering.		Professor of chemistry.
Columbia.....	Mines, engineering chemistry.		Professor of industrial chemistry.

engineering training in addition to a working knowledge of technical or industrial chemistry. His course in college must not be narrow but broad and general

and it is better that the course be administered by a chemical engineer, one who is entirely competent himself to fill the positions for which he is trying to fit the student.

No attempt has been made in this paper to discuss the technical details of the work either in chemistry or engineering. Perhaps some difference of opinion may exist as to the proper place of subjects such as electro-chemistry, but in this analysis everything has been credited to chemistry which involves chemical knowledge and manipulation.

As most of the material herein used has been taken from current catalogs, some mistakes may have occurred, but it is believed that the relative proportions have been preserved. It is hoped that attention may be called to any mis-statements or inaccuracies as the sole purpose of the paper is to bring out facts as a basis for discussion.

ON THE MEASUREMENT OF VERY SMALL GAS PRESSURES.*

By C. F. HALE.†

The extension of studies of the phenomena in rarefied gases to the region of very low pressures has made increasingly important the development of manometers whose indications of pressure are not dependent upon the use of mercury. Thus, with the modern rotary pump it is a relatively simple matter to evacuate glass vessels to a pressure of 0.01 mm. mercury, but if a McLeod manometer is employed to measure the residual pressure, the mercury in it contributes about 10 per cent to the total pressure, and, furthermore, the manometer does not indicate the pressure of the vapor which it contributes; its reading is the partial pressure due to other gases, and not the total pressure. In some experimental work, too, it is imperative that mercury vapor be excluded from the apparatus.

During the last few years a number of non-mercury manometers have been described in the literature. Ladenburg and Lehmann¹ have applied the Bourdon Spiral to the measurement of small changes in pressure, and later it was employed by Johnson² in the investigation of dissociation phenomena with ammonium chloride. This manometer consists in a flat tapered glass tube, bent into the form of a spiral. The wider end of the tube is directly connected to the system, the pressure of which it is desired to measure, while the smaller end of the tube is drawn down to capillary dimensions and sealed. To protect the spiral and to promote the accuracy of the instrument it may be enclosed in a vessel which can be more or less exhausted and maintained at constant pressure. Changes of pressure within the spiral cause a movement of the capillary end of the spiral, and this displacement may be magnified by a stylus or a mirror. The sensitiveness

of the instrument may be considerably increased by using glass tubing with thinner wall, by increasing the number of turns in the spiral and by increasing the diameter of the turns.

Another manometer, which likewise depends upon the mechanical effect produced by changes in pressure, has been devised by Scheel and Heuse.³ This instrument consists of two chambers, separated by a copper membrane. One of the chambers is maintained at constant pressure, while the other is connected with the system which is under observation. The deformation of the membrane under the pressure difference is measured by Fizeau interference bands.

Hogg⁴ has shown that a vibrating disc serves admirably as a manometer for the measurement of low pressures, and that the decrement is connected with the pressure by a simple equation, for pressures below 0.1 mm. down to 0.000016 mm. when the gas in the apparatus is hydrogen.

Dewar⁵ has employed Crookes radiometers in the study of very low pressures, and more recently Knudsen⁶ has made use of radiometric forces in developing an absolute manometer which has been shown to be very reliable at pressures of the order of 0.0001 mm.

Quite different in principle is the manometer suggested by Pirani.⁷ This instrument depends for its ability to measure pressures upon the fact that at low pressures the heat conductivity of gases is a function of the pressure. It is only necessary, therefore, to observe the effects of varying heat dissipation, with pressure changes, upon some source of heat within the vessel undergoing evacuation. Pirani has accomplished this by connecting to the system an ordinary tantalum lamp. The wire in this lamp was heated by a constant-voltage current. As the pressure in the system was decreased more and more, the heat lost by conduction through the gas was continually decreased; consequently the temperature of the wire mounted steadily, and, with the temperature, the resistance of the wire increased, thereby cutting down the current. One had then only to calibrate ammeter readings against the indications of some form of manometer. A better instrument was obtained when the tantalum wire was replaced by platinum, and the wire was clamped tightly to the anchor wires in order to keep constant the heat loss through the supports. With the improved instrument two other methods were used in following the changes in energy loss by conduction through the gas. In order to control the resistance, and consequently the temperature, of the wire, the platinum manometer-lamp was connected into a Wheatstone bridge. One could then follow changes in energy loss by either of two methods.

(a) The wire could be maintained at a constant temperature, and the energy required at different pressures could be recorded.

*Paper prepared for the Twentieth General Meeting of the American Electrochemical Society, in Toronto, Canada, Sept. 21-23, 1911.

†Research Laboratory, General Electric Co., Schenectady, N. Y.

¹Verh. d. d. phys. Ges., 8, 20 (1906).

²Z. phys. Chem., 61, 457 (1908).

³Ber. d. d. phys. Ges., 1909, 1-5.

⁴Phil. Mag., 19, 376-390 (1910).

⁵Proc. Roy. Soc. A 81, 280-286 (1908).

⁶Ann. d. Phys. IV, 32, 809 (1910).

⁷Ber. d. d. phys. Ges., 1906, 686.

(b) The current in the wire could be maintained constant, and record could be made of its resistance at different gas pressures.

In order to compensate for the effect of changes in room temperature upon the resistance of the manometer, Pirani suggested the use of a duplicate of the instrument which could be exhausted and sealed at a pressure in the region where most accurate measurements were required. This duplicate was then substituted for the resistance coil in the arm of the bridge in series with the manometer.

In simplicity both of construction and of operation, involving only current and resistance measurements within the range of instruments readily accessible, the Pirani manometer recommends itself at once. However, in point of sensitiveness to pressure changes it is somewhat disappointing. With the most sensitive arrangement described in Pirani's paper, it would scarcely be possible to follow, with accuracy, pressures lower than 0.0002 mm. or possibly 0.0001 mm., since the change in pressure from 0.0014 mm. to 0.00007 mm., as registered by a McLeod manometer, gave a change of only six divisions upon the chosen scale.

However, certain theoretical considerations led the writer of this paper to believe that the sensitiveness of the resistance manometer could be considerably increased. It is obvious that the apparent sensitiveness, i. e., the change of resistance for a given change in pressure, could be increased simply by increasing the total resistance of the manometer, without in any way altering the real sensitiveness. A little reflection will make it clear that, in comparing manometers as to sensitiveness, one must take account not only of the number of units change in resistance for each unit change of pressure, but also the relation that this change bears to the total resistance of the manometer. For instance, one of the manometers constructed in the course of experiments in this laboratory was found to have a total resistance of 264 ohms. A change of pressure amounting to 0.001 mm. of mercury caused a resistance change of 2.73 ohms in this manometer, while in another manometer, whose resistance was 129 ohms, the change was 1.92 ohms for the same change of pressure. Expressed in percentage of the total resistance, these changes amount to 0.78 per cent and 1.13 per cent per 0.001 mm. change in pressure, and the latter manometer would be considered the more sensitive in the meaning of the foregoing definition.

It will be clear that the sensitiveness of the manometer, since its indications of pressure variation are dependent upon the changes in dissipation of heat by conduction through the gas, will be affected by conditions that will tend to make this dissipation of heat larger in comparison with the heat loss by direct radiation and by conduction along the supports and anchors. Thus the relation of the gas conduction loss to the loss of heat by direct radiation will be different, according to the temperature reached by the wire of the manometer. Also the relative amounts of heat

lost by conduction through the gas and by conduction through the anchors employed to hold the wire in place will be subject to variation with different sizes of anchors and anchors of different materials, as well as their number. The heat loss by conduction through the gas may of course be increased by conditions which would give a steeper temperature gradient between the wire of the manometer and the enclosing bulb. This steepening of the temperature gradient may be brought about by (A) increasing the difference in temperature between the wire and the bulb; (B) decreasing the distance from the wire to the bulb. Again, one would use an enclosing bulb having thin rather than thick walls, thus facilitating the dissipation of heat through the gas. As was to have been expected, the temperature coefficient of resistance of the wire

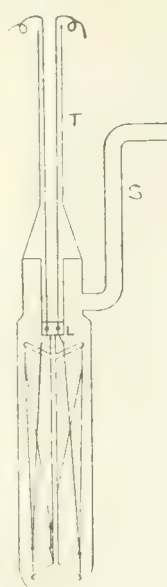


Fig. 1.

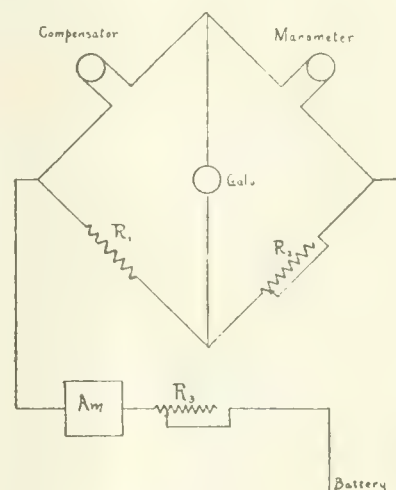


Fig. 2.

used has an important effect upon the sensitiveness of the manometer.

These factors, together with questions regarding the form of the filament, whether the wire should be straight, crimped, or wound into spirals between anchors, and the most favorable size of wire, were made the subjects of careful experiments. The manometer described in the following paragraphs embodies the results of these experiments. The structure of the manometer is shown diagrammatically in Fig. 1. A piece of pure platinum wire, 0.028 mm. in diameter and 450 mm. long, is mounted upon a glass stem carrying two radial glass supports near the top and three at the bottom. The wire is anchored to these radial supports by means of short pieces of platinum wire 0.052 mm. in diameter. The anchor is fused into the radial supports at one end, and the other end is made fast to the manometer wire either by an arc weld or by a tiny glass bead. The leading-in wires at L, to which the ends of the manometer wire are welded, are of platinum, 0.31 mm. in diameter. All of the

heated oven and baked at 360° C. for at least eight hours, the pumps being in continuous operation. Sufficient current was sent through the manometers to raise the temperature of the wire somewhat above that of the oven, although its temperature was always below red heat. Toward the end of the exhaustion, liquid air baths were adjusted to the traps (A, A' and A'', Fig. 3), and the exhaustion continued to 0.00002 mm. by means of the mercury pump alone. The ovens were then removed and the pressure kept at that point until the compensators were cool enough to be sealed off from the apparatus.

Another manometer was now attached as shown at R (Fig. 3), and baked and exhausted for at least an hour without liquid air baths; then the oven was removed and the manometer allowed to cool. Liquid air baths were readjusted, in the order A'', A', A, and then the ice bath was put into place. This order was adopted after we had experienced some trouble with the condensation of traces of mercury in the resistance manometer, and it was found to obviate the difficulty entirely. Probably the trap at A is not necessary; no trace of mercury appears in it except when the liquid air baths have been in place 8 or 10 hours. The compensators were now put into the bath and the electrical connections were made in accordance with the method already described. By operating the mercury pumps or by admitting air, purified by bubbling through strong caustic potash solution, over solid caustic potash, calcium chloride and phosphorus pentoxide, the pressure in the apparatus was brought to the point at which the calibration was to begin, usually between 0.005 and 0.010 mm. As soon as the system had come to equilibrium, as indicated by cessation of resistance changes in the manometer, the resistance was recorded and the pressure read on the McLeod manometer. A few strokes of the mercury pump served to give a new resistance reading and a new pressure, the process being continued until a pressure between 0.00003 and 0.00001 mm. was reached. The resistance readings obtained are, of course, not the actual resistances of the manometer, for the reason that the ratio arms of the bridge are not exactly 100:1,000, but about 90 to 925, and the ratio varies somewhat, of course, with different compensators. However, it is not of importance to obtain actual resistances, since the indications of the manometer were always observed with the same electrical system.

A few sample calibrations chosen at random from a large number of manometers are given above in Table I and are plotted in Figs. 4, 5, 6. The curves for the lower pressures, from 0.0001 mm. down, are plotted on the upper half of the chart, the scale being ten times that of the lower half. The slope of the curve shows that for each change of pressure amounting to 0.00001 mm. the resistance changes more than 0.02 ohm, and as resistances can be measured to one part in 10,000 with a well-made Wheatstone bridge, it seems not unwarrantable to conclude that, with the

apparatus described, we should be able to detect and measure changes of pressure of the order 0.00001 mm. and to follow exhaustion down to a pressure of the order 0.00001 mm. The absolute values of the pressure are not obtained directly from the calibration curves, owing to the fact that the resistance manometer is at 0° C. and the McLeod manometer is at room temperature. The values may, however, be calculated in accordance with the formula derived by Knudsen,*

$$\frac{p_1}{p_2} = \sqrt{\frac{T_1}{T_2}}$$

Where p_1 and p_2 are the pressures, at equilibrium, in two vessels connected by a tube whose diameter is vanishingly small as compared to the mean free path of the molecules of the gas contained in the system

TABLE I

Manometer No.	Date.	Pressure Mm.	Resistance Ohms.
20	July 25, 1910	0.0069	82.68
		0.0047	85.73
		0.0032	88.37
		0.0019	91.37
		0.00095	93.75
		0.00044	95.28
		0.00014	96.23
		0.00008	96.44
	July 26, 1910	0.0049	85.66
		0.0033	88.33
		0.0018	91.48
		0.00080	94.17
		0.00047	95.18
		0.00020	96.02
		0.00011	96.33
		0.00006	96.54
34	April 10, 1911	0.00004	96.63
		0.00384	78.57
		0.00159	81.46
		0.00073	82.90
		0.00024	83.77
		0.000023	84.23
		0.000003	84.24
		0.00778	74.56
	April 11, 1911	0.00331	79.08
		0.00145	81.77
		0.00068	83.01
		0.00015	84.03
		0.00005	84.26
		0.000003	84.28
		0.00291	88.00
		0.00197	89.60
42	July 29, 1911	0.00078	91.49
		0.00032	92.47
		0.00010	92.93
		0.000015	93.12
		0.00159	85.66
		0.00331	87.27
		0.00200	89.52
		0.00124	90.60
	July 31, 1911	0.00056	91.97
		0.00022	92.68
		0.00007	92.99

and T_1 and T_2 are the absolute temperatures of the two vessels. While the average temperature of the gas in the resistance manometer is probably somewhat higher than 0° C., the temperature of the bath, the extreme condition would be represented by basing a correction on 273° as the value of T_1 . T_2 will of course be variable with room temperature changes, but 293 may be taken as an average value. On substituting these values, p (the actual pressure in the manometer) is found to be 0.964 of the pressure recorded by the McLeod manometer. At pressures above 0.0001

mm. this factor is of some importance, but it may be neglected at pressures lower than this.

Some observations have also been made to determine within what degree of accuracy the readings of the manometer from day to day may be relied upon. As a means of checking the system as a whole we have found the use of two compensators a good working method. The two compensators are immersed in the

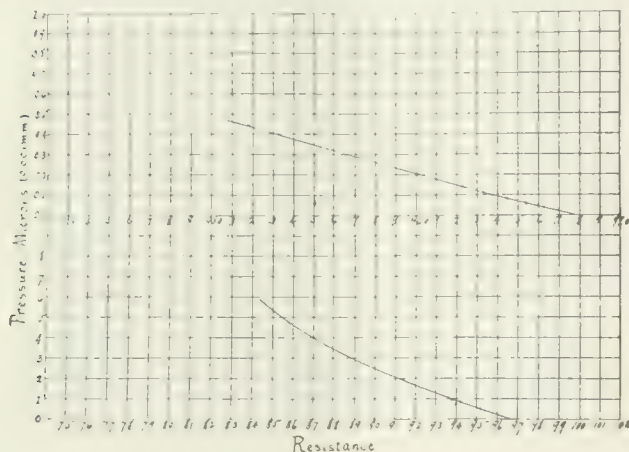


Fig. 4.

bath, one being connected into the bridge in the place usually occupied by a manometer, the usual current is sent through the apparatus, and when equilibrium is established, the resistance is recorded. Then the places of the two compensators are interchanged and the equilibrium resistance obtained as before. These two observations would enable us to detect any trouble in the bridge instruments and in the compensators. We have obtained readings with two compensators over a period of several weeks in which the maximum variation in resistance was 0.01 ohm.

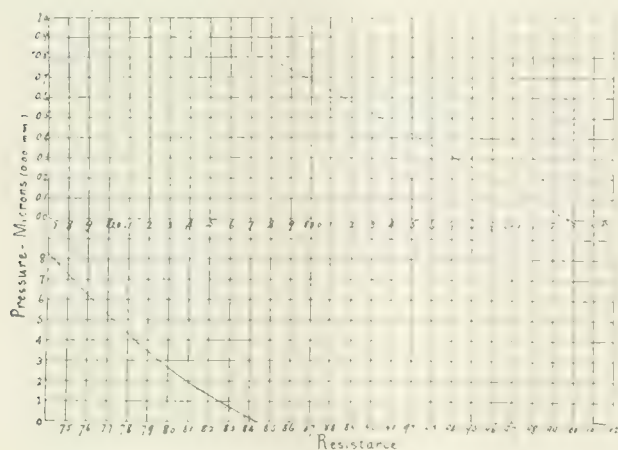


Fig. 5.

Another point which has been subjected to experimental inquiry is the effect of changes in the temperature of the bath upon reading of the manometer. It is clear that when the pressure within the manometer is identical with that in the sealed compensator, variations in external temperature would have a negligible effect, but at pressures above and below that in the compensator, the compensation would be less perfect.

The results of an experiment in which the bath temperature was varied are given in Table II and are plotted on Fig. 7. The manometer used in this experiment was No. 20, whose calibration figures, under normal conditions, are given in Table I.

Pressure Mm.	Bath Temperature 0°	Bath Temperature 17.5°
	Resistance	Resistance
0.0055	84.59	86.98
0.0036	87.70	89.58
0.0020	91.01	92.33
0.00071	94.49	94.90
0.00027	95.80	95.91
0.00007	96.49	96.37
0.00002	96.70	96.48

It will be seen that for this particular system the compensation is perfect at a pressure of 0.00015 mm. At a pressure of 0.005 mm. the change in temperature of the bath from 0° to 17.5° increases the resistance of the manometer by 2.39 ohms, or 0.136 ohms per degree. This change of resistance would involve an error of 0.00087 mm. per degree in the reading of the

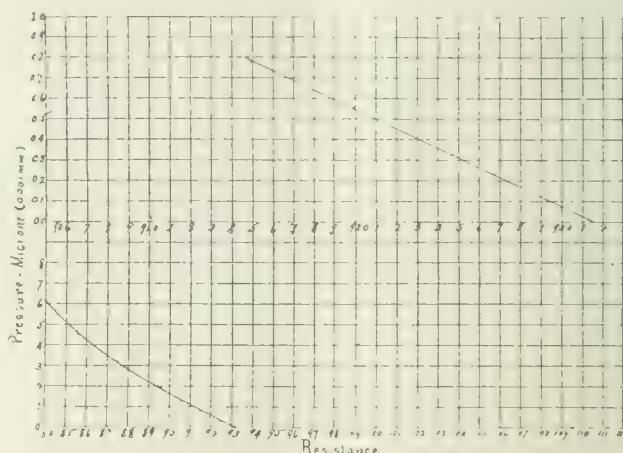


Fig. 6.

manometer. At a pressure of the order 0.00001 mm. this error would be 0.00001 mm. per degree of change in bath temperatures. It has been shown that the bath may be kept constant within 0.1° C., so that the error involved in our measurements, on account of fluctuations in bath temperature, are negligible.

One objection which may very legitimately be raised against this form of manometer arises from the fact that the heat conductivity varies with the nature of the gas so that the manometer must be calibrated with the same gas with which it is to be employed subsequently. At present we have calibrations only in pure dry air and hydrogen. In these experiments the compensator was not filled with hydrogen, so that the compensation for changes in bath temperature would be less perfect than otherwise, but this was considered a small effect as compared with the effect of the change of gas in the system. The calibration resistances under normal conditions and after the apparatus had been carefully filled with hydrogen are given in Table III.

These values are plotted in Fig. 8. Owing to the better conductivity of the hydrogen, the resistance

readings of a manometer, which has been calibrated in air, would, with pure hydrogen, lead to an error of from 20 to 25 per cent of the real pressure. Experiments with other gases have not yet been carried out, but one may estimate whether the effect would be

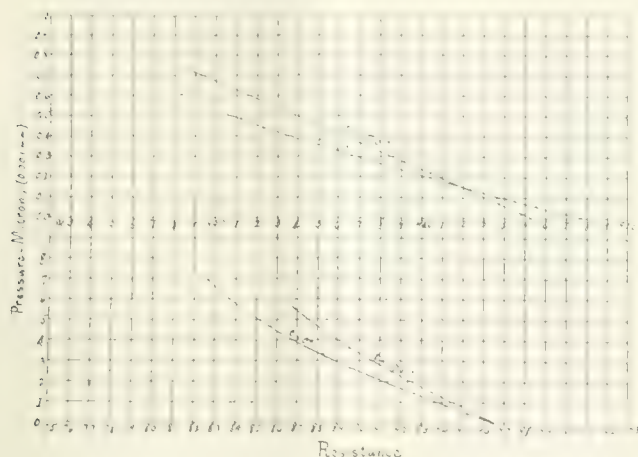


Fig. 7.

greater or less than that of hydrogen, by reference to the recent work of Soddy and Berry,⁹ upon the conductivity of rarefied gases.

It is hoped that further data concerning a number

Pressure	Resistance with Hydrogen	Resistance with air
0.0060	82.80	84.75
0.0040	86.41	87.95
0.0026	89.43	90.95
0.0018	91.66	92.75
0.00053	95.30	95.65
0.00040	95.60	95.64
0.00010	96.56	96.68
0.00008	96.63	96.74

of phenomena in gases at very low pressures may be available for publication in the near future.

The author desires to thank Mr. A. E. Freeman, of this laboratory, for much assistance in carrying out the details of the experiments.

SUMMARY.

The chief results of this paper may be summarized as follows:

1. The resistance manometer suggested by Pirani has been found to be susceptible of considerable improvement in regard to its sensitiveness.
2. A method of calibration of the resistance manometer has been described.
3. The manometer has been found to give reliable measurements of pressure down to 0.00001 mm. of mercury.
4. The manometer must be calibrated in the same gas in which it is to be used to indicate pressures.

The basket-willow industry has not made the progress in the United States which it has in Europe, where every grade of basket is made from the willow.

CENTRIFUGAL PROCESS OF ZINC-COATING.

A patent has been taken out recently in Germany for a process for coating articles with zinc by a heating method. In a recent issue of *Metal Industry*, Mr. Robert Grimshaw briefly describes this process as follows:

The articles to be zinc-coated are first pickled to remove scales, and after being put in the bath of melted zinc are rotated about a vertical axis. The use of "centrifugal" (which might much better be called "tangential") force for freeing zinc-coated articles from superfluous zinc while the latter is still fluid, has been already known and applied; but as far as I know only to wire and the like, which, after being dipped in bundles or coils in the bath are then whirled in a separate containing vessel. Where, however, there is a great number of small articles which cannot be handled in bundles, as wire cans, much time is lost in the transfer; and there takes place a cooling of the metal deposit, sufficient to prevent thorough removal of superfluous zinc.

The application of a heating device to the centrifugal machine, or to the vessel thereto attached, would increase the complication of the process and the accessibility of the apparatus.

A more simple method is to place the articles to be

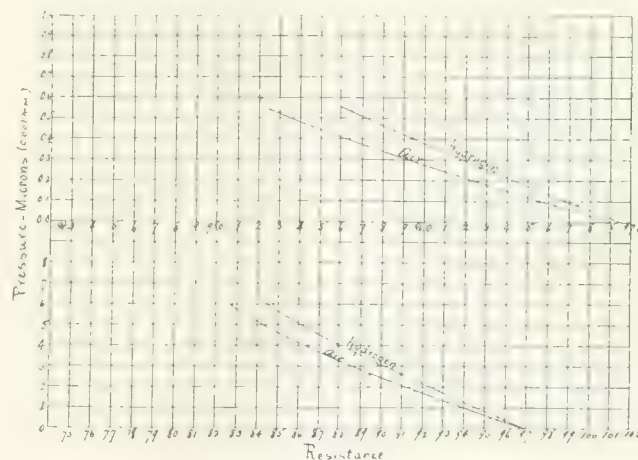
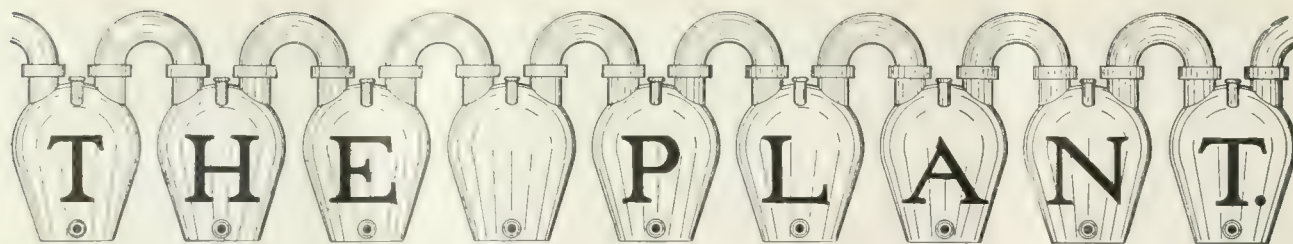


Fig. 8.

coated in a vessel with perforated lateral walls, and to dip this, together with its contents, in the zinc bath. This "basket" and its newly-coated contents are then rotated at the desired speed to drive off the superfluous coating before it has had time to cool. The advantage claimed for this procedure is not only that time is saved, and complication avoided, but that the containing vessel acquires the same temperature as the bath and the articles to be coated.

A complete plant for drying and grinding felspar has been designed by the Ruggles-Coles Engineering Co. for the Bedford Felspar Co., Bedford, N. Y. The capacity of this plant will be 50 tons, 180 mesh felspar, per day.

⁹Proc. Roy. Soc., A 83, 254-264 (1910), and A 84, 576-585 (1911).



THE RECOVERY OF BENZOL FROM COKE-OVEN GASES.

It has been known for many years that ordinary coal gas contains proportionately large percentages of benzol. The rapidly increasing demand for this product by the aniline trades has caused many inventors to attempt to discover means for the extraction of the benzol contained in the gas evolved in the production of furnace coke in by-product ovens. In a recent issue of the *Iron and Coal Trades Review* of London, England, Mr. D. Bagley of Bagley, Mills & Co., London, describes some of the methods that have been employed. The following matter has been taken from Mr. Bagley's article:

The discovery of benzol in coal tar is attributed to A. W. Hoffman, about the year 1845, while its separation was entrusted to Hoffman's pupil, Charles Mansfield, who eventually carried out the process on a large scale, the principle of the separation of the various liquid hydrocarbons being the usual system of dephlegmation, previously used for the rectification of spirits of wine. The huge aniline dye industry owes its inception to the successful production of this product.

The rapidly advancing demand for benzol for the aniline trades stimulated the attempts of inventors to discover means for the extraction of the benzol contained in the gases evolved in the production of furnace coke in by-product ovens. Seeing that in some cases the yield per ton of dry coal is over $2\frac{1}{2}$ gals.—exceeding that contained in the tars derived from the like quantity of coal, generally not exceeding 5 per cent of the volume contained in the coal gases from the unit referred to—and that the removal of this benzol may be effected without unduly affecting the heat volume of the gases used for heating the ovens, it was natural that attempts should be made to recover this product. Various methods were suggested and tried. By cooling the coal gases to $-70^{\circ}\text{C}.$, benzol will separate in the liquid form, but the obvious difficulties of employing this method on a practical scale and the high cost associated therewith preclude its adoption. Another system, that of attaining the separation by means of compression of the gases, was found to be equally impossible.

As it was well known that the heavier oils obtained from the distillation of coal tar possessed the property of absorbing benzol from coal gases, and that by this medium a simple and practical means of recovering the benzol from coke-oven gases was available, the idea was made use of and formed the subject of a

patent in England in 1884 by Carves, who successfully applied the process to his own ovens. Others, notably Brunck and Huessener, worked upon the same principles, the latter using a lighter oil as his absorbent, containing no naphthalene when cooled to $0^{\circ}\text{C}.$, and distilling between $160^{\circ}\text{C}.$ and $210^{\circ}\text{C}.$ His process involved the use of cooling plant for cooling the absorbing oil down to $0^{\circ}\text{C}.$, and in practice a yield equal to 93 per cent of the benzol contained in the gases was attained.

Today the recovery of benzol from coke-oven gases is generally, perhaps exclusively, effected by means of liquid absorbents. The degree to which the cooling of these absorbents shall be carried after separation of the benzol in the separating column remains a matter of argument. If the cooling is to be carried below atmospheric temperature, then special refrigerating appliances must be employed, necessitating an increased capital expenditure, and obviously a rather more complicated system of operation than if the oil be cooled down merely to near atmospheric temperature. As it is possible to extract the like quantity of hydrocarbons from the coal gases by means of absorbent oils not subjected to special refrigeration, this system would appear to possess advantages over any system which requires refrigerating apparatus for cooling the oils prior to their use as absorbents in the gas scrubbers or other vessels.

The two most important parts of the complete apparatus for the recovery of benzol are the gas scrubbers and the separating columns for separating the benzol from the absorbent. Obviously the former must be so proportioned and arranged as to provide the maximum of contact between absorbing oil and gas to ensure the best results, as it must be remembered that the affinity of the oil for the benzol is considerably less than of water for ammonia. As concerns the separating column, this must be capable of continuously separating the benzol and absorbent, and the separation must be complete if losses are to be avoided. The great difficulties attending the construction of this part of the plant to ensure these peremptory conditions will be better appreciated by the distiller.

Until very recently British practice—and then only on a relatively small scale—was almost entirely confined to the production of crude benzol, the buyers usually specifying that 65 per cent should distill over at $120^{\circ}\text{C}.$ A little consideration will show that the market for this product is an extremely narrow one, and that the producer is more or less tied to pur-

chasers possessing the apparatus necessary for its rectification and the production of standard products having a much wider sphere of usefulness.

Further consideration discloses the fact that the producer has facilities generally very much more favorable for the production of finished products at his own works, quite apart from the above. As by this means his net results are considerably enhanced, it appears curious that so little should have been attempted in Great Britain, notwithstanding the twenty-five or more years' lead given by Germany. The answer will probably be found in the desire of most by-product coke oven owners to limit their chemical ac-

tinghausen, Germany, the principal of that firm having devoted many years to the improvement and simplification of benzol recovery and rectification plant. Probably over 90 per cent of the benzol plants in connection with coke ovens in Germany have been erected by this firm.

The ammonia-freed and cooled coke-oven gases from the pressure main A, Fig. 1, ascend two of the oil scrubbers B in series. These scrubbers are provided internally with specially shaped and arranged wooden hurdles to insure the uniform distribution of the oil, fed from a number of sealed sprays situated in the top of the shell. The most intimate contact be-

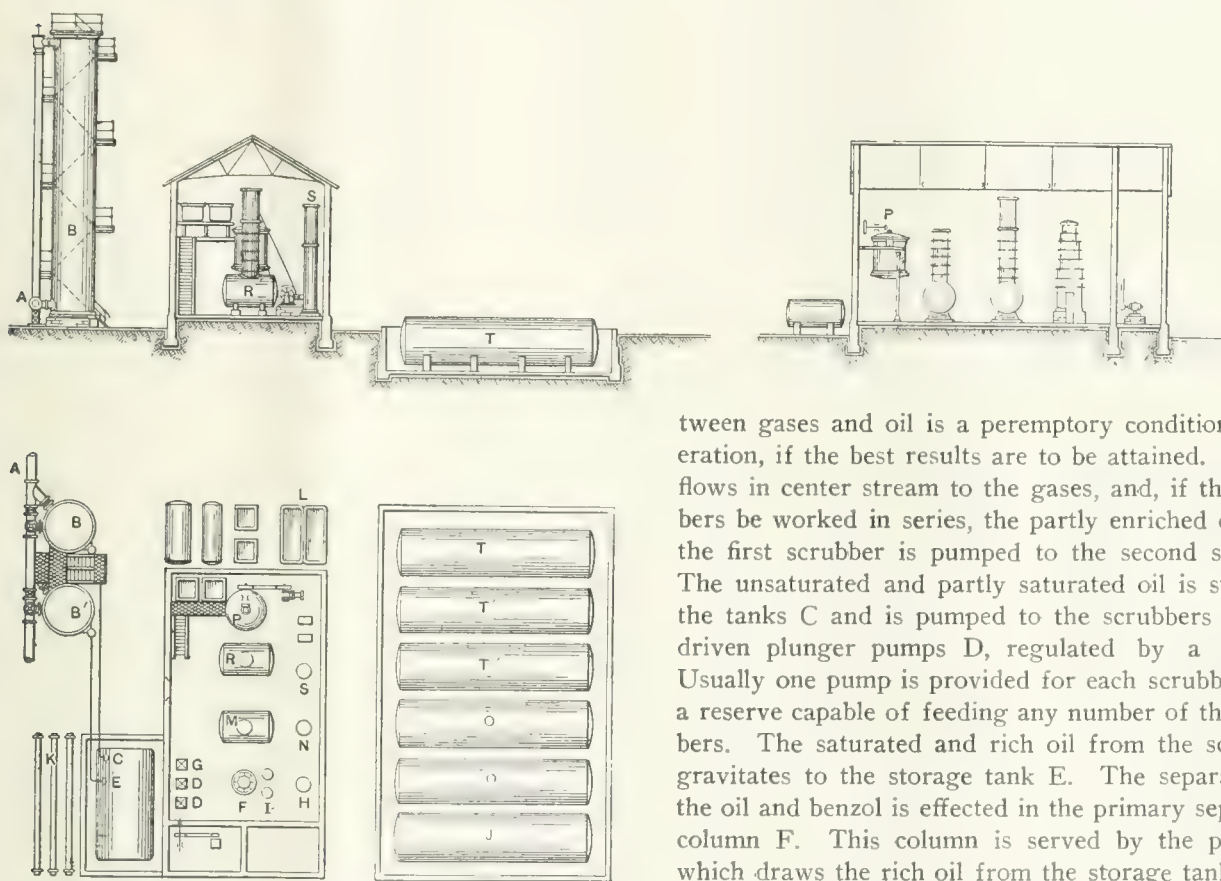


Fig. 1.—Plan and Elevation Showing General Arrangement of Carl Still Plant.

tivities. A trade specification for 90 per cent, of pure, benzol, or toluol, or even solvent naphtha, is not calculated to encourage the cautious colliery agent to extend his operations into unknown fields. But, with modern plant, the advancing of crude benzol to these products and the fulfilment of the tightest specification extant occasion no special skill, the operations being attended with less difficulty than the production of sulphate of ammonia. No one nowadays would think of disposing of crude ammoniacal liquor. The production of sulphate is a simple matter, the markets are world-wide, and the profits from the manufacturer are a prime factor in the cost of furnace coke.

A description is given below of a complete plant according to the system of Messrs. Carl Still of Reck-

tween gases and oil is a peremptory condition of operation, if the best results are to be attained. The oil flows in center stream to the gases, and, if the scrubbers be worked in series, the partly enriched oil from the first scrubber is pumped to the second scrubber. The unsaturated and partly saturated oil is stored in the tanks C and is pumped to the scrubbers by belt-driven plunger pumps D, regulated by a by-pass. Usually one pump is provided for each scrubber, with a reserve capable of feeding any number of the scrubbers. The saturated and rich oil from the scrubbers gravitates to the storage tank E. The separation of the oil and benzol is effected in the primary separating column F. This column is served by the pump G, which draws the rich oil from the storage tank E and forces it through the hot-oil coolers K, and a worm situated in the condenser H, the object being the abstraction of the useful heat from the hot oil flowing from the separating column, and the distillates. The rich oil is then forced through the pre-heaters I, provided with steam-heated worms, which bring it to the proper temperature for separation prior to entering the separating column F. Special arrangements are provided for the prevention of the frothing over of the heated rich oil in the heaters, which forms a distinctive feature of the apparatus, permitting continuous operation of the separating column. The heated rich oil flows in a downward direction through the separating column, where it is subjected to the action of live steam, finally effecting the separation of benzol and oil. The benzol vapors and steam issue from the top of the column, and are condensed in the condenser H, the lower portion of which is arranged as a sep-

arator for water and benzol. The crude benzol flows to the storage tank J, and the condensed water is returned—if needed—to the water storage tank. The now unsaturated oil passes from the separating column to the oil coolers K, a part of its useful heat being employed for raising the temperature of the rich oil, and eventually returns to the storage tank C. It will be noted that the oil is in practically continuous circulation through scrubbers, separating column and coolers.

The product at this stage consists of crude benzol, 55 per cent to 60 per cent distilling over at 120° C., according to the hydrocarbons of the benzene series contained in the oven gases. From the storage tank J, the crude benzol is pumped to the primary still M. This still is fitted with a special column and dephlegmator, and is heated by direct and indirect steam controlled by valves capable of extremely fine regulation. The controlling instruments of this still are so arranged that an ordinary workman can regulate the working, which divides the crude product into the desired number of fractions for producing pure 90 per cent toluol, xylol, and solvent and heavy naphtha. Any heavier products, such as naphthalene and creosote oil, gravitate from the still on the conclusion of the distillation to separating pans L, and are there recovered in marketable form. The fractions are condensed in the condenser N, fitted with water separators, each fraction flowing to separate storage tanks for unwashed products, O. The unwashed products are pumped separately to the washer P. This washer consists of a cylindrical vessel constructed of acid-proof material, fitted with a central agitator. The washing process has for its object the removal of impurities, the nature of which it is not proposed to describe here. First, the benzol product is treated with strong sulphuric acid, admitted to the washer from a special measuring tank fed from the acid storage tank. The acid is removed, and the product then treated with water, and afterward with a solution of caustic soda, to remove all traces of acid. For the removal of acid, water, or caustic solution, a separator is fitted underneath the washer, in order that any slight quantities of benzol or benzol product escaping with these purifying agents may be recovered and returned. A large storage tank is provided for the washed products, so that a sufficient quantity may be stored to fully charge the still, which may be working while the washing is proceeding.

From this storage tank, the final still R is charged with the product to be rectified. This still is similar to the primary still M, but the dephlegmator is of a special description for producing extremely fine fractions to insure compliance with the most exacting requirements of consumers. The heating arrangements are also similar to the primary still, the same fine steam regulation being a feature for efficiently controlling the distillation. Here again, no special skill is necessary, owing to the simplicity of control. The

condensates are condensed in the condenser S. At this stage a tank, divided into a number of compartments, is provided for the preliminary reception of the finished products. This provides a means of effecting any corrections prior to passing the rectified and finished products to their respective storage tanks.

The washing acid is not wasted in the Still system. After leaving the washer, the tarry acid flows to acid-proof vessels, where it is regenerated and any solid products removed. It is then quite fit for use in the ammonia house, and has no deleterious action upon the salt; indeed, its addition to the acid in the saturator is attended with beneficial results. Thus a nuisance is avoided and a source of profit created.

The consumption of steam in the entire plant is extremely low, owing to the inclusion of every available means for utilizing the waste heat in the distillates and hot oils. The power required to drive the belt-driven pumps is approximately 15 hp. for a plant having a capacity equal to the gases from 150 ovens. It is usual to ship the bulk of the finished products in galvanized drums, each holding about 100 gal.

There are many patented and entirely novel features in the installation, to which prominence may not be given in this description. But the combined effect is to enable the whole to be controlled without trained chemists, with the minimum of labor, steam and power, and yet to produce products complying with the more arbitrary trade requirements, thus insuring a ready market.

Naturally, the market outlook for the various products is a matter for consideration. Fortunately an assured future for the finished products may be predicted. But as regards "crude" the maker possesses hardly one tithe of the channels for disposing of his product compared with the manufacturer of pure, washed naphthas, toluol, xylol, solvent, and heavy naphtha. Some time ago attempts were made to introduce "crude" as a motor spirit. It can hardly be said to have proved a success. Containing many impurities chemically inert, its vaporization at ordinary temperatures was attended with difficulties, the deposition of carbonaceous residues on the working parts of the motor was a constant trouble, and the higher efficiency expected by increased thermal value per volume with petrol was hardly ever attained in practice. Now, with a properly constructed and arranged rectifying plant, the production of a motor spirit suitable for any temperature is possible, the constituents of which may be changed at will for abnormal circumstances. To enumerate the various pure commercial products obtained from crude benzol:

Pure Benzol.—This product is largely used in the manufacture of aniline dyes, and is the base of a numerous array of such products. It is also used as a solvent and in the production of very fine varnishes.

Ninety per cent Benzol.—The prefix "90 per cent" is used to denote the fact that 90 per cent will distill over at a temperature of 100° C. This product, like-

wise, is largely consumed by dye manufacturers, but its principal use would not appear to be as a motor spirit, for use in internal combustion engines. As the principal component of crude benzol, the increasing demand in this direction is one of the most favorable and promising features. Used almost exclusively in Germany, and as the preponderating constituent of motor spirits in France, and other countries, it has found favor in Great Britain, and is likely to be in still further demand. Tests made by the writer with a specially prepared quality of benzol fully proved its superior efficiency as compared with petrol. These tests were carried out on a 40-b.h.p. four-cylinder water-cooled motor, fitted with standard apparatus for determining the power developed. The conclusion was, that by adopting proper means for carbureting the air, quite 20 per cent less benzol was used than 0.740 petrol for a given output. These tests were further proved by practical road tests. Other uses are as a solvent, for gas enrichment, dry cleaning, etc.

Toluol.—This product, again, enters largely into the manufacture of aniline dyes, and is used in the preparation of explosives. The percentage in crude benzol varies, but the demand easily absorbs the quantities produced.

Xylol, Solvent and Heavy Naphtha.—These products may be considered together. They form a relatively small quota of the products from crude benzol, nevertheless the demand for these articles is easily sufficient for the make. Xylol and solvent naphtha are extensively used as solvents in the rubber industry, while among the uses for heavy naphtha is that of an illuminant, and for enriching certain gases.

The above brief descriptions do not pretend to denote all the uses to which the refined products from crude benzol may be put. There are a variety of other uses, and these are being periodically augmented. It suffices to show that the ability to dispose of the finished articles at remunerative prices is a real factor in the situation.

It may be argued that increased production abroad will curtail the Continental demand. This is hardly a market view. During the last nine months, with an increasing output, prices have consistently hardened, excluding solvent naphtha, which already commands a fair valuation.

As an economic factor in the production of furnace coke, the recovery and rectification of benzol plays an important part. Probably this phase of the situation appeals to the owners of coke ovens. Although no hard and fast net result per ton of coke made can be stated, owing to the differing content in the gases from various coals, from 25 cents to 35 cents per ton of coke made can be achieved, after providing for all outgoings of the usual character, so the subject from the financial side is worthy of attention. But to insure these results it may be reiterated that the plant of whatever system must be of a proven and practical description, capable of easy control with the min-

imum of labor. These are, and have been, the essential requirements for plants in connection with coke ovens. They will remain in force.

The following results represent a number of tests at the New Barncepeth Collieries, which were recently carried out with the Still benzol plant in order to prove the guarantees given by the contractor. The results attained far exceeded the guarantees given:

- (1) Pure Benzol Distillation.
Charge of the still: 10,535 kg. crude benzol.
Result: 6,130 kg. pure benzol—58.1 per cent.
Test of quality: Within 0.50 deg. C., 90 per cent distilled over. Within 0.70 deg. C., 95 per cent distilled over.
- (2) Pure Benzol Distillation.
Charge of the still: 10,377 kg. crude benzol.
Result: 7,823 kg. pure benzol—75.3 per cent.
Test of quality: Within 0.30 deg. C., 90 per cent distilled over. Within 0.60 deg. C., 95 per cent distilled over.
- (3) Pure Benzol Distillation.
Charge of the still: 10,046 kg. crude benzol.
Result: 6,781 kg. pure benzol—67.5 per cent.
Test of quality: Within 0.40 deg. C., 90 per cent distilled over. Within 0.55 deg. C., 95 per cent distilled over.
- (4) Pure Benzol Distillation.
Charge of the still: 10,414 kg. crude benzol.
Result: 7,653 kg. pure benzol—73.4 per cent.
Test of quality: Within 0.45 deg. C., 90 per cent distilled over. Within 0.60 deg. C., 95 per cent distilled over.
- (5) Pure Benzol Distillation.
Charge of the still: 10,297 kg. crude benzol.
Result: 8,122 kg. pure benzol—78.8 per cent.
Test of quality: Within 0.50 deg. C., 90 per cent distilled over. Within 0.70 deg. C., 95 per cent distilled over.
- (1) Pure Toluol Distillation.
Charge of the still: 22,571 kg.
Result: 10,208 kg. pure toluol—45.2 per cent.
Test of quality: Within 0.35 deg. C., 90 per cent distilled over. Within 0.50 deg. C., 95 per cent distilled over.
- (2) Pure Toluol Distillation.
Charge of the still: 10,911 kg.
Result: 8,184 kg. pure toluol—75 per cent.
Test of quality: Within 0.65 deg. C., 90 per cent distilled over. Within 0.70 deg. C., 95 per cent distilled over.

According to figures of the United States Geological Survey, in 1910 the sales of mineral water in the United States amounted to \$6,357,590, the product being 62,030,125 gals. Minnesota was the greatest producer, with 9,962,370 gals derived from 19 springs. New York was a close second, selling 8,780,903 gals. from 46 springs. Wisconsin, however, obtained the greatest income from her mineral waters, her sales amounting to \$974,366; New York was second, with \$858,635, and Indiana third, with \$514,958. Minnesota's sales amounted to \$281,000. Louisiana has only 4 commercial springs; they produced 2,313,000 gals. Maine's output of mineral waters, from 29 springs, decreased 277,370 gals., but on account of high prices increased in value over 1909, the figures for 1910 being 1,238,171 gals. and \$404,539. Of Wisconsin's mineral waters, 2,151,782 gals. were used in the manufacture of "soft drinks." Pennsylvania has 44 springs and produced 2,536,337 gals., valued at \$221,685. The mineral water trade continues to be prosperous, although there was a decrease in output of about 4 per cent as compared with 1909. The future outlook is good. The importation of mineral waters in 1910 was 3,306,303 gals., valued at \$983,136.

PRODUCER GAS FROM CALIFORNIA CRUDE OILS.

In a topical discussion on fuel oil at the meeting of the American Society of Mechanical Engineers at San Francisco, last March, Mr. E. C. Jones contributed a paper containing some interesting data on the making of producer gas from the California crude oils. This paper was as follows:

Crude oil in California has entirely superseded all other raw materials in the illuminating gas industry. Some progress has been made in the manufacture of oil gas suitable for use in gas-engine cylinders, but practice has not yet demonstrated the economy of the fuel. Its exact status is briefly defined in a paper by E. C. Jones, contributed to the topical discussion on fuel oil at the San Francisco meeting of the American Society of Mechanical Engineers in March last, which we reproduce in full.

"California, with its immense deposits of petroleum, is the natural and logical field for the exploitation and industrial use of oil producer gas. It is to be deplored that such an important subject was first considered by men not conversant with the manufacture of oil gas, and in casting about for apparatus to make producer gas from oil they naturally gravitated to the old familiar methods of retorting the oil, and any improvements that grew out of these methods seemed to have retained the objectionable features of the retort system. Briefly stated, these objections consist of shutdowns for the purpose of removing coke and frequently for burning out accumulated soot and lamp black. A typical analysis of gas made in this way is: CO₂, 4.5; CO, 7.4; O₂, 0.4; CH₄, 12.0; H₂, 3.1; N₂, 71.9; B.t.u. per cubic foot, 172; claimed thermal efficiency, 39 to 62 per cent; actual, 55 per cent.

This gas has been applied to the operation of small gas-engine units up to and including 100 h. p. Owing to the abundance of petroleum in California, it has superseded all crude material in the manufacture of illuminating gas. To attain its present degree of perfection, elaborate experiments were performed for the purpose of changing the chemical and physical condition of the gas to best adapt it to modern domestic and industrial gas appliances.

The first oil gas manufactured on a large scale in California had the following analysis:

	Per Cent.
Heavy hydrocarbons	6.2
Marsh gas	25.6
Hydrogen	62.4
Carbonic oxide	3.0
Carbonic acid gas	0.2
Oxygen	0.4
Residual nitrogen	2.2
	100.00
Specific gravity	0.303

By improvements in apparatus and refinement of operation the hydrogen content of the gas has been reduced to less than 40 per cent; the marsh gas has been increased to 34 per cent; the carbonic oxide has been increased to 9 per cent; the specific gravity to

0.485, and the B.t.u. from 624 to 680 cu. ft. The oil-gas generators used at present for manufacturing illuminating gas are so elastic in their operation that any of them can be immediately adapted to the manufacture of producer gas from oil. The chemical composition of the gas and its heating value depend only on the manipulation of the generator.

The writer has carried on a series of experiments with oil producer gas, using a large generating unit, and the only change in equipment was the use of compressed air at from 35 lbs. to 40 lbs. pressure for the injection of oil and to assist in the partial combustion of the oil. During these experiments producer gas was made having a thermal value as low as 103 B.t.u. per cubic foot, and as high as 482 B.t.u. per cubic foot. Unfortunately no ready means was at hand for measuring the quantity of gas and the amount of oil used to produce 1,000 cu. ft. A typical analysis of this gas is as follows:

	Per Cent.
Carbonic acid	4
Heavy hydrocarbon	2 or less
Oxygen	1
Carbonic oxide	10
Hydrogen	5
Marsh gas	8
Nitrogen	70
	100

This gas has a calorific value of 160 B.t.u. per cubic foot. To use this gas successfully in gas engines it is necessary that it shall be thoroughly cleansed by efficient scrubbing, and that it shall be uniform in calorific value and chemical constituents. This last can be readily accomplished in the operation of oil-gas generators, used as producers, by careful measurement of the oil used, and of the air supplied for its partial combustion, and by the maintenance of a fairly constant temperature in the generator. This can be done, as in oil-gas manufacture, by a determination of the necessary temperature by the observation of color in all checker brick and the maintenance of this temperature by frequent observations through sight cocks.

Making producer gas from oil in the ordinary oil-gas generator has many advantages over any special process. The gas can be made in very large quantities and the amount made can be easily regulated to the needs. The operation is continuous and without interruptions for cleaning. This is essential in the manufacture of producer gas for power purposes. In any other known process the interruptions are not at stated intervals, but occur when the producer refuses to work, owing to the clogging of its parts by coke or lamp black. Oil-gas generators have no easily destructible parts, as the lining and checker brick are constructed with a view to resisting high temperatures, and although a much higher temperature is desirable in making producer gas than that employed in making oil gas, the checker brick is not seriously affected by the high temperature. In the decomposition of oil, in the presence of air, there is a complete disposition of all the gas-making constituents of the oil, so that pro-

ducer gas can be made without a by-product of any kind. Any accumulation of carbon in the generator may be removed by adjusting the temperature and quantity of air supplied. This method of making gas requires a small gasholder for momentary storage, and the process as at present understood could not be used as a suction gas producer. Producer gas made from oil and containing a small percentage of hydrogen possesses advantages over illuminating oil gas, inasmuch as it can be subjected to higher compression in gas-engine cylinders, and with a gas of uniform analysis uniform piston speed can be obtained.

THE FIXATION OF ATMOSPHERIC NITROGEN.*

By DR. MILTON W. FRANKLIN.

One of the most pressing of modern problems is the supply of combined nitrogen for agricultural purposes. There are three substances, essential to the life of plants, which are being constantly extracted from the soil by the crops growing thereon. These are nitrogen, phosphorus and potassium.

The continual abstraction of these substances from the soil without any corresponding replacement renders essential their periodic application if the soil is not to be unduly impoverished. In early agricultural operations, animal fertilizer sufficed for the needs of the time; but during recent years the supply has become greatly inadequate, and it becomes essential that some form of artificial fertilizer be employed. Nitrate of soda and sulphate of ammonia have been extensively used for this purpose and have answered perfectly well. The supplies, however, are extremely limited and the consumption is increasing at an enormous rate. It is estimated that the consumption of Chili saltpetre increased from 250,000 tons in 1850 to 1,540,000 tons in 1903, and at the present time the rate of consumption is of the order of 2,000,000 tons per year. It is estimated that the supply cannot last more than about 20 years. Within the last year or two, the price in Germany has increased about 40 per cent. For many years Peruvian guano has been used, but the supply has become exhausted. The sewerage of cities, if utilized in an efficient manner, would prove at best but an insignificant item in the total world's demand. It has been estimated that in England alone there is wasted through her sewers \$80,000,000 per year; and to date, no efficient method has been proposed for the satisfactory utilization of city sewerage.

Sulphate of ammonia is now manufactured in large quantities as a by-product in gas works, and also in the Mond Power Gas generating stations. It is, however, improbable that this class of combined nitrogen can successfully replace Chili saltpetre when the latter shall have become exhausted. The ratio in which the two substances are used at the present time is 2:5, the consumption of ammonia sulphate in Germany during

1904 being estimated at about 202,000 tons and that of Chili saltpetre about 500,000 tons. The above ratio applies to Germany, the largest user of artificial fertilizer, but the ratio for the whole world is placed at 1:4. At the same time it must be considered that the world's supply of coal is extremely limited, and that therefore even should the production of ammonia and sulphate by the Mond process be enormously increased, the limit is nevertheless defined.

The atmosphere surrounding the earth contains a supply of nitrogen equal to about $4.041 \times (10)^{18}$ tons. This is estimated on the basis of 31,000 tons of nitrogen per acre of surface of the earth, at which rate the air over every nine acres contains about 280,000 tons, equivalent to the amount of Chili salt petre used in the year of 1907. Roughly speaking, four-fifths of the air which surrounds the earth is nitrogen.

Nitric acid is an acid compound of hydrogen, nitrogen and oxygen. The formula is HNO_3 . It is interesting to know that in 1669 Mayo wrote of nitric acid as containing two components; one from the air and the other from the earth. In 1776 Lavoisier demonstrated its oxygen content, and Cavendish demonstrated its complete composition by preparing it synthetically from oxygen and nitrogen in the presence of water. Nitric acid does not occur in a free state in nature; but after thunderstorms, traces of it may be found in the air and in rain water, and according to one authority, amounts up to 0.66 mg. per liter are found in rain falling on the Alps. It occurs largely combined in the form of alkalin nitrates in Chili and elsewhere, the formation of the nitrates being supposed to have originated in the putrefaction of nitrogenous organic matters. The latter are assumed to be converted into ammonia, and this to be oxidized in the presence of hydroxides of sodium, potassium and calcium. Chemically, it is an exceedingly inert element, and its compounds until recently have been produced artificially only from the decomposition of more complex organic compounds, as in the manufacture of coal gas.

Nitrogen compounds are essentially unstable chemically, and it is this instability which has rendered the fixation of nitrogen difficult of accomplishment by artificial means, the critical temperature of dissociation existing so near the temperature of combination that disintegration is prone to occur immediately the combination has been brought about. This in fact is but a phase of the instability of nitrogenous compounds.

There are two proven methods for fixation; viz., the Birkeland-Eyde process, and the Cyanamide process of Caro and Frank. A modification of the Birkeland-Eyde process, due to Dr. Schönherr and operated by the *Badische Aniline und Soda Fabrik*, is also worthy of description.

In the Birkeland-Eyde process, a high-tension alternating current flame is blown into a disk transversely to a direct current magnetic field. The gases mixed with steam are passed directly over lime. This process

*From the General Electric Review.

is modified in the *Badische Aniline und Soda Fabrik* in whose process long threadlike arcs are employed. The air is blown tangentially so as to circulate spirally around the arc.

The difficulties which have beset the development of these processes in the past have been the high cost of electric power, and the fact that the process of nitrogen fixation in the flame is a reversible one. At certain critical points the combined nitrogen is again decomposed; and it is with the perfection of this detail that the whole development has been concerned, cheap water power having been available in many places for some time.

In the processes based upon this principle, the object is to produce NO_2 and NO which combined with water give a mixture of nitric and nitrous acids. From this is produced calcium nitrate, in which form the fertilizer is sold.

In the cyanamide process, of Caro and Frank, the calcium carbide is first produced in an electric oven, and, while red hot, is treated with nitrogen, the result being a calcium-nitrogen compound.

The first attempt on anything like a commercial scale at the fixation of atmospheric nitrogen was made by Bradley and Lovejoy at Niagara Falls, but although the greatest credit is due to both of these experimenters the results were unsatisfactory technically. The Birkeland-Eyde process followed, and was really the first to produce satisfactory results. This was, however, owing in a large measure to the fact that cheap water power was available in abundant quantity. The Birkeland-Eyde process may be described briefly as follows:

The apparatus consists of a pair of water-cooled copper tubes 15 mm. diameter employed as electrodes. These electrodes are suitable for flames up to 750 kw. at 3,500 volts with a gap of 1 cm. The electrodes are placed in a magnetic field of about 4,000 to 5,000 lines of force per square centimeter in the center. This field blows the arcs into the form of a large fan, the maximum diameter of flame being about 140 cm. The object of the magnetic blow and arrangement is to increase the rate of cutting of the flame and the air, which is also assisted by the air blast supplied. The furnace in which the discharge takes place is lined with fire-brick and is said to last about six months. The inside temperature of the lining does not rise above 700°C . during normal working, owing to the cooling effect of the blast of air supplied, although the temperature of the disk of flame is very considerably higher. The results are extremely good as will be seen by reference to Table I. This is partly because the power employed is large, and partly because the rate of cutting of the air with the flames is high, both being conducive to efficient working.

Of the foregoing description, Prof. S. P. Thompson made the following criticism: The authors had hardly appreciated the importance of the spontaneous further oxidation of the gases formed in the passage of the air through the flaming arcs. The gases, after they

left the furnace were not treated with water or lime until they had been allowed to remain (and cool) a considerable time in large oxidation chambers. If that were not done a very large proportion of liquid formed in the water tanks would be nitrous, instead of nitric acid. It seemed to him that 900 kg. of nitric acid per kw.-year was too high for the average commercial yield of that process. If he remembered rightly, this was an exceptional, not an average figure; the average figure being nearer 600 kilos. It was well that there should be no exaggeration, and he happened to know that in Norway the experts expressly took a lower figure, so as to have a margin of safety in their calculations. The large scale working of the new factory at Notodden had certainly shown a higher economical yield than that assumed in the estimates.

That the Birkeland-Eyde process might not be successful in this country is suggested by Dr. Frank as follows: The production of calcium nitrate $(\text{Ca N O}_3)_2 \cdot 4\text{H}_2\text{O}$, prepared by dissolving CaO or Ca CO_3 in (HNO_3) Ag. by the Birkeland-Eyde process, seems to be developing exceedingly well in Norway. It should, however, be noticed, that this industry has not made much progress in any other land save Norway; but it should not be forgotten that it is only in this northern clime that electrical energy can be cheaply obtained on account of its unrivaled resources of water power.

The *Badische Aniline und Soda Fabrik* process is one of the newer ones, which promises to give results superior to those obtained by the Birkeland-Eyde process. The air is blown tangentially so as to circulate spirally around the quietly-burning arc. In some examples of this apparatus recently exhibited the tubes are of glass, coated on the inner periphery by a wire spiral, which is in metallic contact with a knob at the top of the tube. At the bottom of the tube, the wire spiral is separated only by a short distance from the other electrode, which is placed in the axis of the tube. When a 3,000 volt supply is applied between the two electrodes a spark jumps across and the arc travels rapidly up the tube. If air is forced through the bottom, in a number of symmetrically placed tangential nozzles, the arc burns quietly in the axis of the tube, without touching the sides, and absorbs about four amperes at 3,000 volts. The air charged with nitrous gases which leaves the top of the tube gives up some of its heat in raising the temperature of the fresh air, about to enter the tube, to some 500°C . and the rest is employed in steam generation. The cooled gas is then mixed with water to form nitric acid, which, after treatment with lime, results in the required manuring material. An experimental installation on these lines was opened in Christiansand (South Norway) in the spring of 1907, with a total power of 2,000 h. p. Three ovens are at work, each with a length of flame of five meters and absorbing 600 h. p., alternating current of fifty cycles is employed, and the arc burns quietly and reliably.

The third and perhaps the most important process at the present time is the cyanamide process of Caro and Frank, which has been briefly referred to above. The process appears to yield the best results per horse power expended and is described by Dr. Frank as follows:

The carbide coming from the electric furnaces is ground-charged into retorts made of fireproof material which are mounted in a furnace similar to gas-house furnaces. The nitrogen is then passed over the carbide at a temperature of from 800° to 1,000° C. The carbide used is of the same quality and percentage as that employed for lighting purposes, and the nitrogen is obtained by fractional distillation of liquid air by the Linde system, or the so-called "copper" process in which air is passed through heated copper particles. The copper takes up the oxygen and the free nitrogen passes to the furnaces. The resulting copper oxide is reduced in the same apparatus by treatment with reducing gases or vapors, and the copper which is removed is then ready for a new cycle. In the Linde process the oxygen remaining after separation of the nitrogen may be utilized for any purposes. As soon as the carbide in the retorts is saturated with nitrogen, a fact which will be made apparent by the controlling gas meter coming to a standstill, the calcium cyanamide is extracted from the retort in the form of a hard cake and cooled while the air is excluded. It is then ground to a fine powder and is ready for use. Recently a new electric furnace has been developed for treating carbide with nitrogen and is being universally adopted by all the new cyanamide factories.

The calcium cyanamide obtained is only about 57 to 63 per cent pure, and is known as lime nitrogen or nitrolim. It contains about 20 to 22 per cent nitrogen, about the same as sulphate of ammonia. Most carbide works yield about two tons of carbide per kw.-year, and two tons of carbide will combine with practically 500 kg. of nitrogen in the form of nitrolim; a power of 2 kw. is required per year for fixing one ton of nitrogen by the Caro and Frank process. In addition thereto about one-third of one horse power is required for grinding and all other separations.

At \$20 a kw.-year then the cost of power to produce one ton of nitrogen would be \$45, and for one ton of nitrolim \$9. Comparing the Birkeland-Eyde process with the Caro and Frank cyanamide process, Dr. Frank pointed out that the works at Odden which produce 2,500 tons of nitrogen only employ 5,000 kw. to 6,000 kw.; whereas from the statement of Mr. Eyde it appears that in the works at Notodden in Norway, for the preparation of an equivalent amount of nitrogen in the form of nitrate of calcium, 25,000 kw. are required. At \$20 a kw.-year this would mean \$200 for electric power per ton of nitrogen or \$25 per ton of calcium nitrate ($\text{Ca}[\text{NO}_3]_2 \cdot 4\text{H}_2\text{O}$).

The figures given in the accompanying tables are derived from data given in papers by various authors and may not be accurate.

In regard to the relative merits of cyanamide and nitrate of lime as a commercial product, Sylvanus P. Thompson expressed himself as follows: There was no doubt that the two successful processes today were the nitrate of lime process and the cyanamide process, but though both of them took nitrogen from the air, the products were of different character. Nitrate of lime has proved to be just as satisfactory for agricultural purposes as nitrate of soda, and, in the case of a heavy clay soil, the lime in it gave it a preference. Cyanamide, on the other hand, was akin to sulphate of ammonia rather than to nitrate of soda, and competed with the former rather than with the latter. Moreover, nitrate of lime had certain important applications in the coal tar color industry for which the cyanamide would not serve.

While the above figures would indicate that the cyanamide process is the more economical, the results

TABLE I.

	Formula.	Cost per Ton.	Per Cent N.	Cost per Ton N.
Sodium nitrate	NaNO_3	\$40.00	19	\$210
Ammonia sulphate ...	$(\text{NH}_4)_2\text{S}_2\text{O}_8$	57.80	22	260

TABLE II.

	Formula.	Cost of production per ton at \$20 per Kw.-year.	Per cent N.	Cost of production per ton of N at \$20 per Kw.-year.
Calcium cyanamide.	CaCN_2 (Crude)	\$9	21	\$45
Calcium nitrate.....	$\text{Ca}(\text{NO}_3)_2 \cdot 4\text{H}_2\text{O}$	25*	12	200*

*Birkeland-Eyde process.

of experiment do not seem to demonstrate that the product of this process is as generally satisfactory as that of the direct air process.

In practice it has often been found that some uncombined calcium carbide has remained in the nitrolim, and has proven offensive in generating acetylene upon becoming wet. Cyanamide manufacturers claim that this objection has been satisfactorily remedied and that the process is cheaper; but the fact remains that a good deal of capital has been, and constantly is being, invested in processes which effect a direct combination of oxygen and nitrogen from the air.

In *Zeit. f. Elektrochemie*, Jan. 1, 1911, a review is given by F. Haber and A. Koenig of the present status of fixation of atmospheric nitrogen. The following table sums up the results obtained with the three successful processes in commercial operation: the process of the *Badische Company* (Dr. Schönherr) the process of Birkeland and Eyde, and the process of *Salpetersaure-Industrie Company* (Pauling):

	Grams HNO_3 per kw.-hour.	Concentration in per cents NO .
Badische Company	75	2.5
Birkeland and Eyde.....	70	2
Pauling	60	1 to 1.5

The table gives both the yield in grams HNO_3 per kilowatt-hour and the concentration in per cent of

NO, since both items are of fundamental commercial importance.

The comparison with theoretical figures is difficult. The commercial yield of the process of the *Badische* Company is about 63 per cent of the theoretical value obtainable (without regeneration of heat), on the hypothesis that the oxidation of the nitrogen is a purely thermal process and that the temperature of the arc is $3,800^{\circ}\text{C}$. The authors give a very useful review of the various scientific investigations made in recent years on the mechanism of the oxidation of atmospheric nitrogen by electric discharges through the air.

It may be concluded that the fixation of atmospheric nitrogen opens a vast field for manufacturers of electrical apparatus. The total kilowatt capacity of electrical machinery already installed is very large, and with the refinement of existing processes and the development of new methods, it may be expected that there will be created an increasing demand for specialized electrical appliances. The subject of frequency and its influence on the process has not been closely examined as yet, and much research will have been prosecuted before a knowledge has been obtained of just what potentials fulfill the most nearly ideal requirements.

DIRECTIONS FOR THE RECOVERY, CONCENTRATION AND TESTING OF AMMONIA.*

The following set of instructions for the recovery, concentration and testing of ammonia were prepared in the Detroit office of Hodenpyl, Hardy & Co., for the use of the superintendents and foremen in their gas plants. This paper is not intended to be a general discussion of the chemical principles involved in the different methods of recovering and concentrating ammonia, but merely a set of instructions covering the method in ordinary use in small plants. Thinking possibly these instructions might be of some use to those in charge of other small plants throughout the state, they are herewith contributed.

A few of the things that make the recovery of ammonia necessary are as follows:

1. Waste gas liquor cannot be run into the sewer, because the odor would be obnoxious and cause trouble.
2. The ammonia must be removed from the gas, because if allowed to go forward it would damage the works' apparatus and consumers' meters and fixtures.
3. A net revenue amounting to 30 to 40 cts. per ton of coal carbonized, or 3 cts. to 4 cts. per 1,000 cu. ft. of gas, is derived from the ammonia recovered when a proper yield is obtained.

The retort conditions most favorable for the production of ammonia are medium heats and large charges. The yield of ammonia increases with the yield of gas per pound of coal, up to 5.1 lbs., at which

point it begins to decrease as the yield of gas further increases. The reason for this is that the NH_3 is broken up by exposure to a very high retort temperature.

The crude ammoniacal liquor from different coals differs greatly in composition. The liquor from what is ordinarily known as "Island Creek" coal, will contain about 25 per cent of fixed ammonia, while the liquor from what is ordinarily known as "Youghiogheny" coal will contain about 10 per cent of fixed ammonia.

Only 10 to 15 per cent of the total nitrogen in the coal is recovered as NH_3 .

The ammonia is recovered from the gas in various places in the plant, in the form of condensation and by absorption in water. The following table gives approximately the amounts recovered at the different points:

	Per cent Recovered.	Per cent Strength.	Per cent fixed. ¹ NH_3 .
Hydraulic and foul mains.....	5	0.6	60
Condensers	35	2.0	6
Prim. tower scrubber.....	45	1.0	..
Fresh water tower scrubber.....	15	0.3	..

The point of importance in the condensing system is the maintenance of the proper temperatures throughout. The solubility of ammonia is much greater at low than at high temperatures, and as the temperature throughout the scrubbing system is controlled by the condensers it is necessary to bring the temperature at the outlet of the condensers to the proper point, in order to recover the ammonia in the scrubbers. No cooling of the gas is done in the scrubbers except that from radiation and the addition of the fresh water. These temperatures must be maintained at the proper point for other reasons than the recovery of ammonia.

TABLE OF TEMPERATURES.

Position.	Degrees F.
Inlet of P. & A.	100-120
Inlet secondary condenser.....	95-110
Inlet primary scrubber.....	70-80
Outlet of fresh water scrubber.....	60-70

It will be seen from the above table, under "Point of Recovery," that about 60 per cent of the ammonia remains in the gas at the outlet of the condenser, to be absorbed by the water in the washers and scrubbers. To accomplish this absorption, intimate contact must be obtained between the gas and the water for sufficient time. In the ordinary type of tower scrubber this contact is obtained by wetting the surface of the scrubber, filling and allowing the gas to travel along this surface at a slow rate.

The scrubber filling which gives the greatest amount of wetted surface, and allows* the gas to pass with the least amount of back pressure, has been found in wooden trays, made up of boards on edge, with dividing strips. Another scrubber, which has been found very efficient in getting this intimate contact, is of the rotary type, but it is not ordinarily used in small works.

Ammonia being very soluble in water, and at the ordinary temperatures at which scrubbers should be

*Paper prepared for the 20th Meeting, Michigan Gas Association, by James A. Brown.

maintained, 1 volume of water will absorb 600 volumes of NH_3 ; 1 gal. of fresh water will absorb $3\frac{1}{2}$ lbs. NH_3 at these temperatures, and as there is only about .3 lb. NH_3 in 1,000 cu. ft. of gas at the scrubbers, it is apparent there is always sufficient water added if proper time contact with the gas is obtained. To insure this intimate contact, the following points must be watched:

It is very necessary to have the liquor with which the gas is to be scrubbed evenly distributed over the surface of the scrubber filling. This is accomplished by various methods, among the best of which is:

1. To have the liquor at the top of the scrubber impinge on a properly shaped block, which will scatter the liquor evenly over the surface of the scrubber.

2. To admit the liquor at *intervals* with a flush tank in such large quantities that there will be no question of its being distributed over the entire surface of the scrubber filling. This is necessary where it is difficult to properly distribute a small quantity of liquor.

3. To use a revolving distributor, made up of a pipe extending radially from the center, and attached in such a manner that it may revolve around same. This pipe has two or three openings in the side, and these openings being made in such a way that they distribute the liquor over a certain part of radius of the scrubber, the resistance against the gas of the liquor emerging from these openings will cause this pipe to revolve around the center of the scrubber.

To insure the absorption of the greatest amount of NH_3 gas, the liquor is sometimes recirculated over and over in the scrubbers by means of a pump; 50 gals. per 1,000 cu. ft. of gas should be a sufficient rate to pump liquor through the scrubber. As this recirculated liquor is always saturated, no appreciable loss of illuminating value will result from re-circulation. It is necessary, however, to separate the liquor from any tar that might accumulate and re-circulate only the liquor, for the reason that tar at the temperatures maintained in the scrubbers will rapidly absorb the illuminants from the gas. Re-circulating a large quantity of crude liquor in the primary scrubber takes out considerable H_2S , thus lessening the work on the oxide purifiers.

Sufficient fresh water is admitted to the final scrubber to absorb the last traces of ammonia, and the liquor from this scrubber is also sometimes re-circulated. With the ordinary type of tower scrubber, the amount of fresh water necessary to completely absorb the ammonia from the gas is found to be in the neighborhood of 4 gals. per 1,000, and this is a safe quantity to use at the proper temperature. When using 4 gals. of fresh water per 1,000 cu. ft. of gas, the strength of the crude liquor in the well should be from .9 per cent to 1 per cent NH_3 . If, with the proper temperature, 4 gals. of water per 1,000 cu. ft. of gas are found insufficient, and there is an even distribution of liquor in the top of the scrubber, it

would indicate that the scrubber trays need cleaning or replacing.

Daily tests should be taken at the outlet of the fresh water scrubber, to insure that no ammonia is being carried forward in the gas. These tests should be made with sensitive red-litmus paper, and should it be found at any time that the litmus will discolor to any appreciable degree within 30 seconds, too much ammonia is going forward. This would indicate that insufficient fresh water was being admitted to the fresh water scrubber, that the distribution throughout the scrubber was poor, that the temperatures carried throughout the condensing and scrubbing system were too high, or that the scrubber trays were dirty or clogged.

If scrubbers are maintained at the proper temperature and liquor is evenly distributed over the filling and 4 gals. of water will not completely absorb the NH_3 , it is evident that the scrubber filling should be removed and cleaned. This filling will possibly, after some time, become clogged with naphthalene or tar.

The crude liquor should be stored in a covered well or tank. If open to a circulation of air, the amount of fixed ammonia in the liquor will increase, also some ammonia will be volatilized and lost. Any leakage from these wells will represent considerable waste of ammonia and should be constantly guarded against.

In the recovery of NH_3 in the washers and scrubbers, a double purpose is accomplished. If sufficient time contact is allowed, the NH_3 will combine with H_2S and CO_2 , and the scrubbing water will also absorb the H_2S and CO_2 , thus relieving the purifiers of considerable work.

To accomplish any appreciable absorption of H_2S , the gas must be thoroughly scrubbed with liquor containing uncombined NH_3 . Under the best conditions with tower scrubbers, about 40 per cent of the H_2S in the gas can be removed in the crude ammoniacal liquor. In order to accomplish a proper absorption of H_2S , it is usually necessary to re-circulate in the scrubbers the liquor from the hydraulic main and condensers, as that liquor contains uncombined ammonia.

It should be unnecessary to emphasize the importance of guarding against any leakage in the pipe lines and through the walls of tanks and wells, for any leakage, however small, will have a serious effect on yield of ammonia obtained.

In order to recover the ammonia from the gas, the following operating conditions must be maintained throughout the condensing and scrubbing system:

1. Maintain the proper temperatures in the condensers and scrubbers.

2. Provide for even distribution in the top of the scrubbers.

3. If a re-circulation system is used, see that the circulation is continuous and sufficient.

4. Use about 4 gallons of fresh water per 1,000 in final scrubber.

¹This amount will vary greatly, according to kind of coal used.

5. Make daily litmus tests of the gas at the outlet of final scrubber.
6. Keep trays and scrubber filling clean.
7. Keep all wells and tanks covered and protected from air circulation and the sun's rays.
8. Insure against all leaks in lines and wells.

In this article it is assumed that the ammonia still consists of the usual series of chambers known as the volatile still, the large mixing chamber into which the lime is injected known as the lime leg, and the series of chambers known as the fixed still in the bottom of which all the steam for heating the still is ordinarily admitted.

Besides the still proper, there is usually an overhead tank for the crude liquor and another for the lime water. There is also a condenser with leading coil, absorber, and second absorber with sometimes other accessories to facilitate the proper working of the still.

Ammonia stills of the following sizes should be sufficient to easily take care of the liquor from the daily carbonization of the following amounts of coal:

Still, inches.	Tons, daily.
48.....	150
36.....	100
24.....	50

Crude liquor to be 1 per cent.

Keep flowing at a constant rate:

Still inches.	Rate per hour, gallons.	Rate per minute, gallons.
48.....	450	7.5
36.....	300	5.0
24.....	150	2.5

Stills of the above sizes will easily take care of the above quantities of crude liquor per hour under ordinary conditions. To determine the proper rate of feed the following is the procedure:

With a certain strength of crude liquor, the necessary temperature to get a certain strength concentrated liquor is determined. With sufficient lime of constant good quality being fed to the still, the feed of crude liquor can be set to the greatest rate that will allow the waste to be free from NH_3 . A graduated quadrant should be placed on the regulating cock of the crude liquor feed line and the proper point determined. The still should always run with this cock set in the proper position.

For a certain strength of concentrated liquor, a certain temperature at the outlet of the still must be maintained. This temperature is determined by the desired strength of the concentrated liquor. For 16 per cent liquor it is usually found to be in the neighborhood of 190° . The temperature to be carried at this point is also dependent upon the strength of the crude liquor. If the crude liquor is very weak this temperature will have to be carried lower in order to obtain a concentrated liquor of good strength, and if carried too low there will be a loss of ammonia in the waste. If the strength of the crude liquor is high, this temperature must be carried higher in order that the waste be kept free from ammonia. With a high strength of crude liquor, a higher temperature can be carried and a stronger concentrated liquor obtained than when the

still is worked with low strength, crude liquor and low temperature.

In order to keep this temperature constant, a reducing pressure valve is always used on the steam inlet to the still. This reducing pressure valve must be in good working condition in order to have any success in keeping the still at a constant temperature. A steam gauge should be installed at the outlet of this reducing pressure valve, in order to observe and regulate the pressure on the still. Automatic temperature controls, that regulate the amount of steam entering the still according to the temperature carried at the outlet of the still, are sometimes installed.

This is regulated by adjusting the amount of water admitted to the condenser for cooling purposes, and is usually carried at about 100° at this point, if there is a cooling coil in the absorber; if not, about 70° . The temperature that can be carried at this point depends upon the quality of the crude liquor. Some liquors that carry a high content of CO_2 and H_2S salt in the condenser very easily. Other liquors, of a low content of CO_2 and H_2S , allow a very low temperature to be carried at this point. The lowest temperature possible without salting the lead coil in the condenser, should be the one carried. Carrying a low temperature here keeps the uncondensed ammonia gas from getting over into the second absorber and prevents any possibility of waste at this point.

If the temperature of the absorber is carried very low, the concentrated liquor being very strong, the absorber will become more or less coated with salt and its efficiency thereby cut down. If the temperature of the absorber is carried too high the uncondensed ammonia gases from the condenser will pass through this absorber and go over into the second absorber, resulting very likely in a loss.

The function of the second absorber is to catch and absorb any vapors which may be unabsorbed in the first. The liquor in this second absorber should be drained off daily and replaced with fresh water. The pipe leading from the first absorber to the second is very apt to stop up with salt and should be examined daily. The liquor drained off from the second absorber should be put back into the crude liquor well.

If the temperature at the outlet of the still is allowed to vary to any great degree, strong liquor will be made at such time as the temperature is low. If the temperature is too low, and too strong liquor is made, it will salt in the condenser coils and there will be an accompanying loss through the waste. Under some conditions it is possible to make a much stronger liquor than under others. The liquor from some coals can be concentrated to as high as 18 per cent in the ordinary still. The liquor from other coals cannot be concentrated higher than 14 per cent without salting the still. Under some conditions of carbonization and subsequent condensing and scrubbing, the crude liquor will contain considerable CO_2 and H_2S , which makes it impossible to obtain a concentrated liquor of over 14 per cent.

It has been determined by experiment that $2\frac{1}{2}$ lbs. of lime are necessary to free the fixed ammonia in the liquor from one ton of Island Creek coal. One pound of lime is required to free the fixed ammonia in the liquor from one ton of Youghiogheny coal.

In ordinary practice a good rule to follow for the quantity of lime to be used is: 1 lb. of lime for every 10 per cent of fixed ammonia in the crude liquor from one ton of coal. The lime should be used in the greatest concentration possible, as any excess water increases the work on the fixed still, making it more difficult to remove the last trace of ammonia. To have the lime water of constant good quality, flowing into the machine, it is quite necessary to have some means of agitating it in the overhead lime tank. This can be done with compressed air, if available, if not, with exhaust steam from one of the pumps. The use of exhaust steam is recommended, as it preheats the milk of lime before entering the still. There is considerable settlement in the lime leg of the ordinary ammonia still, consequently this should be blown off often enough to keep all passages clear. This should not be necessary more than once in 24 hours. The plates should be removed from the lime leg and cleaned every time the machine is shut down, or once in 10 days. It is sometimes found necessary to allow a small amount of steam to enter the lime leg, near the bottom of the pipe through which the ammonia liquor comes down from the volatile still, in order to keep the waste free from ammonia. Just enough steam should be admitted to make up for the radiation of heat from the lime leg of the still. A $\frac{1}{4}$ -in. valve, $\frac{1}{4}$ turn open, should be sufficient on most stills.

By reference to the table of specific gravities of lime water, the amount of lime being admitted to the still can be ascertained. The rate of flow of the lime water in gallons per minute is measured and the specific gravity taken with a Twaddell hydrometer. By reference to the table the pounds of lime being admitted per minute can be found.

No matter how small, any leaks on the still, especially on the lead coil at the outlet of the still, should be stopped at once. A small leak of ammonia gas will seriously cut down the efficiency of a still. In fact, the importance of keeping the still free from these leaks cannot be overestimated. Any small leaks, for instance, in the stuffing box of concentrated liquor pump, or through a cock on concentrated liquor line or tank, runs 24 hours a day, and the loss of ammonia is considerable. Strict attention should be paid to keep these leaks eliminated. If this is not done it will have a very serious effect on the ammonia yield.

Daily tests should be made of the waste liquor from the still to check up the operating conditions. Any amount of fixed ammonia found in the waste indicates either too small an amount of lime being used, too low a temperature carried at the outlet of the still, or too much water being used with the lime.

If too low a temperature is being carried at the outlet of the still, insufficient steam is being admitted

to drive off the ammonia. Too much water with the lime increases the quantity of liquor passing through the fixed still, consequently the liquor is subjected to a shorter boiling period. Weak crude liquor also increases the work upon the still. Periodical tests should be made on the still to determine its efficiency. When the still is running under proper conditions 97 per cent of the ammonia in the liquor should be accounted for in the concentrated liquor.

In estimating the loss of ammonia, in pounds per ton of coal, after taking a test of the waste, it must be remembered that the waste liquor per ton of coal is made up of the crude liquor, the lime water and the condensed steam, amounting with 1 per cent crude liquor to nearly 100 gallons per ton of coal, or 825 to 850 lbs. of waste liquor. Example: Waste testing .01 per cent is equivalent to .085 lbs NH_3 per ton of coal.

The steam necessary to concentrate the crude liquor at 1 per cent strength from 1 ton of coal will be in the neighborhood of 150 lbs.

The ammonia still is very simple in theory, but requires constant care and supervision to operate properly. There is probably no other piece of apparatus used in gas works that requires as much attention as the ammonia still. A certain balance must be maintained between the temperatures at the outlet of the still, the strength of the crude liquor and the amount of ammonia found in the waste. If one of these things varies the others will vary, with possibly an accompanying loss of ammonia or an accompanying make of weak concentrated liquor. When this balance has once been determined, the condition should be maintained constant at all times. The following points are the ones to be looked after:

1. Keep the strength of the crude liquor constant.
2. Keep rate of crude liquor feed constant.
3. Keep the temperature constant at the outlet of still.
4. Keep temperature constant at outlet condenser.
5. Keep absorber at proper temperature.
6. Keep the proper amount of milk of lime flowing in at a constant rate and have this of constant good quality.
7. Eliminate all leaks.
8. Test waste liquor daily.

STRENGTH OF LIME WATER.

Degs. Twaddle.	Specific Gravity.	Pounds lime (CaO) in 1 gal.
0.....	1.00	0.00
2.....	1.01	0.11
4.....	1.02	0.22
6.....	1.03	0.33
8.....	1.04	0.45
10.....	1.05	0.56
12.....	1.06	0.67
14.....	1.07	0.78
16.....	1.08	0.89
18.....	1.09	0.91
20.....	1.10	1.14
22.....	1.11	1.26
24.....	1.12	1.37

In order to make a proper test for ammonia the first and most important thing is to obtain a proper sample.

To get a uniform sample of liquor from the well, tank or tank car, a piece of apparatus, commonly called a "gun" is used. For the construction of this gun, see illustration.

The procedure is as follows:

1. Slowly lower the gun to the bottom of the well or tank with the valve held open; allow the valve to close, then raise gun and discharge contents into some receptacle other than the same bottle. This is for the purpose of cleansing the gun and having the liquor that adheres to the sides of the gun of the same strength as the liquor in the well or tank.

2. Slowly lower the gun again, as before, and after carefully wiping off the outside of the gun, discharge the contents into the sample bottle.

TEST OF LIQUOR—NO. 1 METHOD.

This is the most customary method of making a test of the strength of ammonia liquor. The necessary apparatus is illustrated and marked "No. 1 Method." A list of the necessary chemicals and apparatus is as follows:

One Burette, 50 to 100 c.c.—preferably the latter. Milk back best. If ordinary burette is bought, it will require one Erdman's float. Glass stopcock best. Also:

One Bunsen burner; 2 retort stands, about 2 feet high; 2 retort rings for above—1 of 4 inches diameter

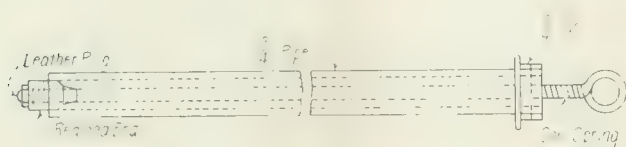


Fig. 1.—Sample Gun.

and one of 1 inch or 1½ inches diameter; 1 burette clamp; 3 Erlenmeyer flasks, 250 c.c.; 1 rubber stopper to fit above flask, with one hole to fit glass tubing; 1 porcelain dish, 6 inches diameter; 1 glass stirring rod, 6 inches long; 1 yard glass tubing, to fit inside tubing on funnel; 1 foot rubber tubing; two 2½ inch funnels; two 10 c.c. pipettes; 1 Twaddell hydrometer; 1 small glass beaker; 1 pound pot. hydrox. in sticks; 1 ounce methyl-orange; 1 ounce cochineal bugs; 2 small bottles, to hold indicators; 1 small mortar and pestle; one 100 c.c. cylindrical graduate.

The method is as follows:

With the 10 cc. pipette, draw a sample from the liquor to be tested. Allow this first sample to run to waste. Draw another sample and run into the Erlenmeyer flask. Put into latter about 50 cc. of distilled water and a piece of pot. hydrox. about the size of a pea. Put the rubber stopper, with the glass tubing inserted, into the top of the flask and lower the funnel into the evaporating dish about ½ full of distilled water until sealed. Add several drops of cochineal solution to the bath in the evaporating dish, for the indicator. Place Bunsen burner under the flask and start heating. As soon as burner is placed, read the burette and run into the evaporating dish from the

burette, standardized sulphuric acid at a rate that will keep the bath in the evaporating dish always of a straw color. Boil for about 20 minutes, and along towards the last add the acid very slowly, drop-by-drop, in just sufficient quantity to neutralize the ammonia vapors as they come over. This end reaction can be noticed a

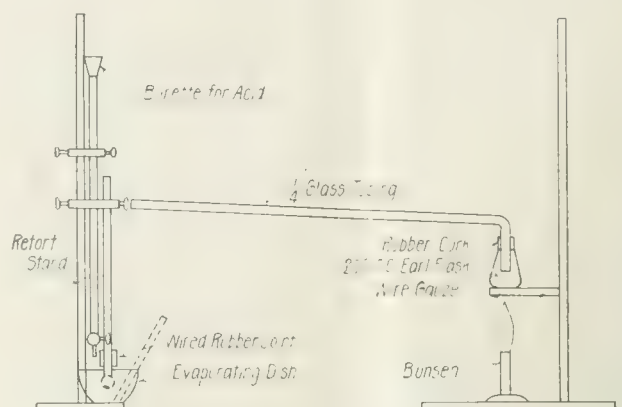


Fig. 2.—Testing Apparatus No. 1 Method.

little more closely if some methyl-orange is placed in a separate white dish and a drop from the evaporating dish brought over with the stirring rod from time to time. The methyl-orange serves as an indicator as to whether the bath in the evaporating dish is acid or alkaline.

When the last drop of acid necessary to turn the bath to an acid color is in, the number of cc. acid used is read from the burette and the computation is as follows:

$$\frac{\text{cc. H}_2\text{SO}_4 \times \text{factor} \times .017}{\text{cc. of sample} \times \text{sp. grav.}} \times 100 = \text{per cent NH in sample.}$$

The specific gravity of the sample is obtained by the use of a Twaddell hydrometer. To obtain the specific gravity from the reading of the hydrometer, divide the reading on the hydrometer by 2 and place in the

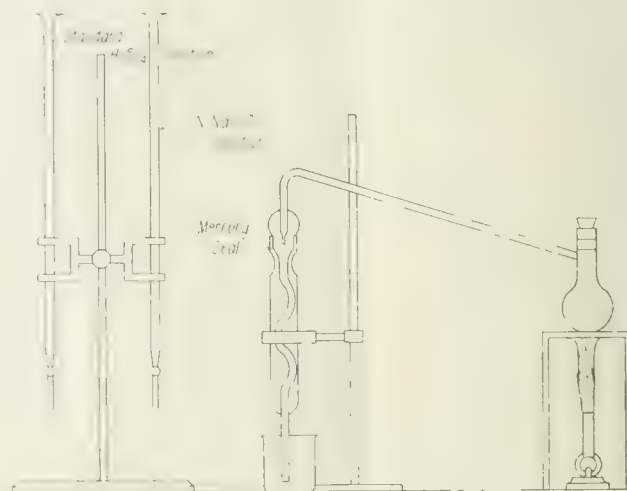


Fig. 3.—Testing Apparatus No. 2 Method.

second decimal place after 1; e. g., if the reading on the hydrometer is 18, the specific gravity will 1.09. If the reading on the hydrometer is 21, the specific gravity will be 1.05. In the formula above given, .017 is the grams of ammonia that 1 cc. of normal H₂SO₄ will

neutralize. In making the acid it is not necessary to have it exactly normal, and the factor in the above formula represents its relation to normal.

TEST OF LIQUOR, NO. 2 METHOD.

Another method which has some advantages over the first one described is that the end reaction is a little more easily observed. The apparatus for this method is shown in the illustration marked "No. 2 method." A list of the necessary chemicals and apparatus is as follows:

One ammonia determination flask with condenser, No. 1,668, E. H. Sargent & Co.; two 100 cc. burettes, milk back, glass cock; 1 double burette stand with 2 burette clamps; one 500 cc. beaker; 1 tripod, 12 inches high; 2 pieces $\frac{1}{4}$ -inch rubber tubing, 3 inches long; 1 bottle normal H_2SO_4 ; 1 pound can C. P., Na_2CO_3 ; 1 Twaddell hydrometer; 1 Bunsen burner; 1 pound potassium hydroxide in sticks, KOH; two 10 cc. pipettes; one cc. graduate, cylindrical; two $2\frac{1}{2}$ -inch funnels; 1 ounce methyl-orange; 1 ounce cochineal bugs.

No. 2 Method.—The method of procedure is as follows: Take the sample of ammonia liquor, place in the flask of distilled water and potassium hydroxide and submerge the end of the condenser in distilled water in the beaker, as in "Method No. 1."

Connect up the condenser to the water and start. Put in from the acid burette what you know to be an excess of acid. Boil for 20 minutes to $\frac{1}{2}$ hour, stirring the liquor in the beaker occasionally. When boiling is completed take the beaker over to the burette containing normal sodium carbonate Na_2CO_3 . Place in the beaker, 2 or 3 drops of cochineal solution for the indicator and add Na_2CO_3 until the bath shows alkaline. The computation is as follows:

$(\text{cc. H}_2\text{SO}_4 \times \text{factor} - \text{cc. Na}_2\text{CO}_3 \times \text{factor}) \times .017$

$\frac{\text{cc. of sample} \times \text{Spec. Grav.}}{\times 100} = \% \text{ NH}_3 \text{ in sample.}$

To make a test of the crude liquor or waste, proceed as above, using 50 cc. of sample instead of 10.

It is well, whenever new acid is obtained, to check with the old acid or with a normal alkali. Acid may be prepared by titration against the normal alkali. To prepare a normal alkali, weigh out 53 grammes of Na_2CO_3 , which has been heated in a crucible until the weight is constant. With these 53 grammes of Na_2CO_3 make a solution with distilled water up to 1,000 cc. at 39°F . One cc. of this normal alkali will neutralize 1 cc. of normal H_2SO_4 . cc. H_2SO_4 is added to distilled water and titrated against the Na_2CO_3 solution. The computation to find the factor

of the acid is as follows: $\frac{\text{cc. of Na}_2\text{CO}_3}{\text{cc. of H}_2\text{SO}_4} = \text{factor.}$

There were 417 mines producing gold, silver, copper, lead, or zinc in Idaho in 1910, compared with 324 in 1909. Of these 417 producing mines, 262 were placer mines, many of which made very small outputs.

THE MANUFACTURE OF C. P. HYDROCHLORIC ACID.*

Hydrochloric acid is usually made from sodium chloride and sulfuric acid according to the following formula: $2\text{NaCl} + \text{H}_2\text{SO}_4 = \text{HCl} + \text{Na}_2\text{SO}_4$. The HCl gas evolved is dissolved in water forming a solution when concentrated at 15°C containing about 38% HCl. The sodium sulfate formed at the same time is a by-product, but has a commercial value and is used extensively in the manufacture of glass. Hydrochloric acid can also be made synthetically from chlorine and hydrogen, and its production in this way is now being

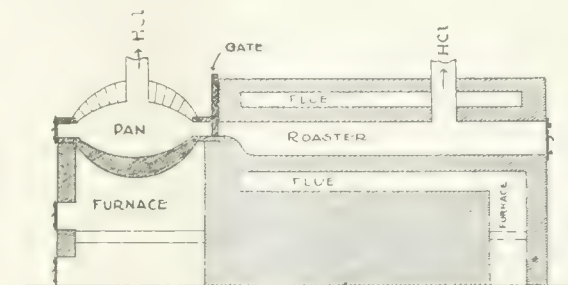


Fig. 1.

made to a limited extent as a by-product in connection with the electrolytic production of caustic potash.

Made from sodium chloride and sulfuric acid, the operation is performed in a retort essentially as shown in Fig's 1 and 2, Fig. 1 being a longitudinal section of Fig. 2. The "pan" is made of heavy cast-iron, 9 or 10 ft. in diameter, enclosed in brick, having a fire place below and a door on the side for charging, and arched over with an opening at the top for carrying off the HCl vapors. Connected with the pan is a "roast-

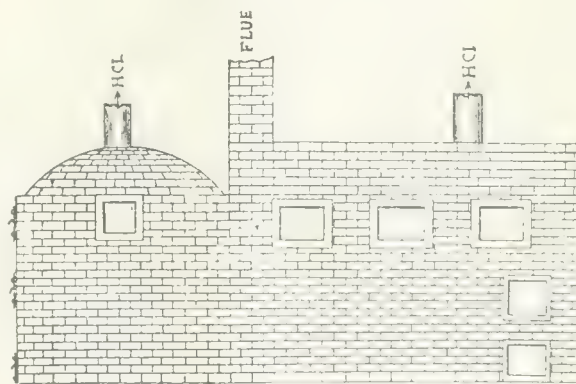


Fig. 2.

er," which is built entirely of brick, with fire flues above and below the doors along the side and at one end, and an opening on top to carry off the HCl vapors.

The charge consisting of a ton or more of salt is placed in the pan and the requisite amount of sulfuric acid run in with it; vapors of HCl are given off immediately, and as the temperature rises the mixture is occasionally turned over by means of a long-handled iron rake thrust in through the door. When the mass becomes pasty and begins to cake to the pan, it is

*From The Chemist Analyst.

scraped over into the roaster, where the temperature can be raised to a red heat necessary to complete the decomposition; the mass is turned over occasionally with rakes through the openings along the side, and the residue of salt cake is finally discharged through the door at the end.

The acid vapors evolved from the pan are comparatively cool and can be absorbed in water or weak acid in large stoneware receivers connected in series and air cooled, or in vessels immersed in the water-bath, similar to the arrangement used for cooling C. P. Acid, as shown in Fig. 3. The temperature of the vapors from the roaster are necessarily very high and are therefore conducted first through cooling chambers and then into a tower where they are absorbed in water or weak acid flowing over broken pumice stone with which the tower is filled.

The hydrochloric produced in this way has a strength varying from 1.16 to 1.18 sp. gr. and contains a considerable amount of impurities, mostly sulfuric acid and iron, the latter impurity causing the acid to have a decided yellow color. For making chemically pure

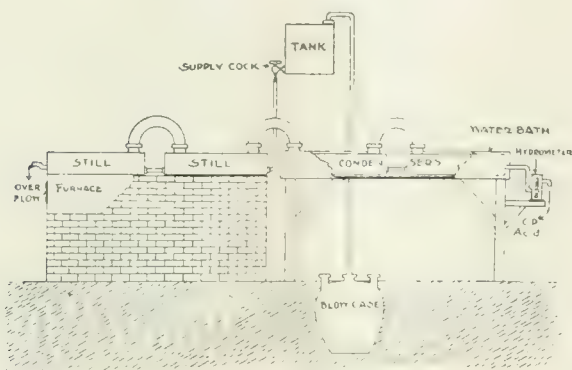


Fig. 3.

acid, these impurities are removed by distillation in an apparatus which will not be acted upon by the acid, such as glass or porcelain. The distillation of hydrochloric acid is peculiar in that it does not distil at a uniform degree of strength above 1.10 sp. gr.; above this strength HCl gas is driven off and the distillate will be stronger than the original liquid, while the latter becomes gradually weaker until it reaches a strength of 1.10 sp. gr., at which point it remains stationary and distills at a uniform strength. If we start with a strong acid, say 1.18 sp. gr., the boiling will commence with the first application of heat and at first HCl gas will be given off, which will gradually diminish until the strength is reduced to 1.10. To prevent the loss of HCl gas which distills off during the first part of the operation, means must be provided for absorbing it; this can be done by conducting the gas into a receiver containing water or weak acid and then changing the receiver when no more gas is evolved; the balance of the distillate will be only about 1.10 sp. gr., which in order to be marketable must be brought up to 1.19 sp. gr., saturating it with HCl gas from a subsequent distillation.

This intermittent process requires frequent changing and cooling of receivers to absorb the heat generated by the absorption and involves much labor and some loss of gas while the receivers are being changed. The process could be simplified by adding to the still a constant supply of strong acid as the distillation is going on, so that the distillate will always be of uniform strength, but the objection to this scheme is the accumulation of impurities, which are liable to distill over or be carried over mechanically with the acid when they become concentrated.

The last difficulty can be overcome by providing for a slight overflow from the still to gradually carry off the impurities, but to accomplish this an apparatus of special construction is required, substantially as shown in Fig. 3. This apparatus is designed to operate continually and automatically as far as possible. The acid to be purified is forced by air pressure from the blow-case set in the ground where it can be conveniently filled to the tank elevated above the apparatus, so that the acid will flow by gravity into the stills, the flow being regulated by a cock. The acid enters the still at the end farthest from the fire, and as it flows through the apparatus, the temperature of the liquid rises, so that at different points in the still there will be vapors distilling off at different strengths, all of which will combine again in the condensers to form an acid of full strength. As the acid flows through the apparatus, it becomes weaker until at the end opposite from where it enters, it is only about 1.10 sp. gr., at which point the impurities also tend to concentrate, and, with a constant overflow amounting to about 10% of the acid run in, these are drawn off with sufficient rapidity to prevent them from being carried over with the distillate.

The condensing vessels are placed in a bath of running water, which absorbs the heat of the vapors and the heat generated by their absorption, and the condensed acid flows out at the farther end through a glass jar containing a small hydrometer to show the strength. By this system a constant supply of chemically pure acid of uniform strength is produced, and by enlarging the number of vessels used, the output can be increased to any desired quantity, with a minimum amount of labor.

The number of deep mines producing metals in Montana in 1910 was 423, against 448 in 1909, and the number of producing placer mines was 157, against 170. The average total recoverable value per ton of ore produced increased from \$9.39 in 1909 to \$9.41 in 1910.

The output of lead in Idaho in 1910 increased 12,272,554 lbs. over that of 1909. The value of the lead produced in 1910 was 61.5 per cent of the value of the total metal production. Crude ore shipped to smelters contained 50,084,367 lbs., and concentrates (of which all but 704,052 lbs. came from Shoshone County)

M E T A L L U R G Y.

RECENT PROGRESS IN THE ROASTING OF SULPHIDES.*

By M. LEON GUILLET.

After several journeys in Belgium and Germany and visiting various plants in France the author has made this study of the roasting of sulphides, especially the roasting of galena and blende. Considerable progress seems to have been made in the industry during the last twelve or fifteen years, but development is still going on.

It is found necessary to review the articles which have already appeared in the *Revue*, and in doing this the following classification of roasting furnaces is made:

Furnaces with material in contact with fuel.	Heaps and stalls.	Relieve for lumps ores.	Free falling shafts.
	Shaft furnaces		Shafts with shelves.
		{ For fine ores.	Malétra-type, hand raked. Malétra-type, mechanical raked.
Reverberatory furnaces.	With shaft.		Hand raked.
		{ Fixed hearths.	Mechanical raked.
	Hearths	Portion of furnace moves.	
Muffle furnaces.		Whole furnace revolves.	
Converters.	{	Operated by blast.	
		Operated with suction.	

Heap roasting is rarely used and then only where sulphurous acid may be freely emitted and lost.

Stalls somewhat relieve the injury done by sulphurous acid.

Shaft furnaces are more used but desulphurization is incomplete, especially with blende. They are used in the Harz and at Mansfield.

Shelf furnaces have given only ordinary results.

The Malétra-type furnaces are used for roasting pyrite and making sulphuric acid and give better results. With blende the results are not as good. With these furnaces the original sulphur content must be high or the roasting will be liable to die out. In all cases considerable sulphur is left in the ore.

Reverberatory furnaces have been greatly improved and are discussed later.

Muffle furnaces give pure products but are very costly of construction, repairs and operation. They are

the only ones used in Europe for the roasting of blende.

Metallurgists and manufacturers have largely first paid attention to the hearth and reverberatory furnaces and the improvements will be enumerated.

Converters have become especially useful for roasting galena and have made a great improvement in the metallurgy of lead.

IMPROVED CONSTRUCTION OF ROASTING FURNACES.

The greatest efforts have been in the line of using mechanical in place of hand labor. In spite of this effort the old type of hand-stirred reverberatory still persists, as at Trail.

Plans of a recent furnace constructed by Fraser and Chalmers are given.

Besides mechanical rabbling the improvements in recent construction are in capacity, structural perfection, easier charging and improved heating, either by gas or regenerative.

The first mechanically operated furnaces were those invented by Black and Larkin, of Manchester, in 1880. In this year the furnaces were for calcining, only, but the next year, 1881, they took out a patent for desulphurizing furnaces as applied to copper and other ores. Only in 1883 did MacDougall first use his furnace in connection with lead chambers.

The original MacDougall had eight hearths and the ore was raked from the center of one hearth across to the edge and then on the hearth below from the edge back to the center of the furnace. In 1904, after a rather slow introduction, we find this furnace in widespread use.

Other furnaces of this general type, each with particular improvements, are the Herreschoff, Repath and Marcy, Schorr, and Kleptko. Their great improvement over the old Malétra type is in:

- Much less manual labor.
- Uniform roasting.
- Regular gas evolution.
- Higher per cent of sulphurous acid in gas.
- Better desulphurization.

The same improvements found in the shaft and hearth furnaces are also made in the reverberatory furnaces.

- Furnaces with fixed hearths.
 - Longitudinally dragged plows. O'Hara, Brown, Ropp furnaces.
 - Rabbles on vertical axis. Edwards furnace.
- Movable hearths and fixed rabbles.
 - Revolving hearth and fixed rabbles. Brunton and Heberlein furnaces.

*Translated and abstracted from the *Revue de Metallurgie*, 8, 8, August, 1911, pp. 561-605, by H. B. Pulsifer.

- b. Tilting hearths. Edwards furnace.
- C. Revolving furnaces.
 - a. Axis horizontal. Brückner cylinder.
 - b. Axis inclined. Oxland cylinder.

Muffle furnaces, especially for blende roasting, of the old Hasenclever hand-raked type, are still preferred in Europe. Other types are the Meyer furnace with fixed muffle and moving rabblers, the MacDouglas furnace with moving muffle, and the Zellweger furnace into whose muffle the rabblers are introduced at intervals.

A few details as to the capacity, mechanical moving, charging and heating of some of the above-mentioned furnaces are given. In the Oxland furnace at Laurium the heat generated by the gas firing was so strong that too much fusing of the charge and volatilization took place. These four furnaces are being replaced by some of the Brunton or Heberlein type.

As to the roasting of blende it is shown that the different requirements of European and American conditions have developed different types of furnaces. In Europe the need of complete roasting, costly fuel and sulphuric acid manufacture demands muffle furnaces and superimposed hearths; either several hearths may be in one muffle, or the muffles, each containing its hearth, may be placed one above another. Plans and details of the Delplace furnace, a type largely used in Belgium and France, are given.

In America costly labor, cheap fuel and the wasting of the gaseous products of the roasting has demanded a mechanically raked furnace of the Brown type. The Hegler furnace is, however, used if sulphuric acid is to be made.

For the roasting of stibnite and the production of antimony oxide the furnace of Chatillon, following the French method, has proved a great advance over previous constructions. The new furnaces at Blesles are described in detail with plans.

NEW METHODS OF ROASTING.

The distinctly modern methods as carried out for the roasting of lead and copper ores by the Huntington-Heberlein, Savelsberg, Carmichael-Bradford and Dwight-Lloyd processes are now described at length and illustrated with ample drawings.

At Pertusola, in Italy, experiments were conducted with nearly all types of roasting furnaces from 1875 until 1898, when "converters" were first successfully used. With this beginning the process spread rapidly to Braubach, Munsterbusch, and then to all parts of the world.

The Huntington-Heberlein process begins with roasting the ore or a mixture of ore and diluent in a revolving hearth furnace and then charging this pre-roasted material into a converter, where the process is completed. In the first roasting, the sulphur, which

was originally at from 14 to 18 per cent, is reduced to from 8 to 10 per cent. The limestone which was once considered necessary for the roasting reaction is now generally thought to be of physical benefit only, other materials replacing it to no detriment in both laboratory and practical operations.

The converters, proper, still used in many parts of Europe, are similar to the small ones used at Laurium, which resemble slag pots on trucks and hold from 1 to 3 tons. The more recent converters are of the pot type and hold up to 15 tons; those used in America holding from 8 to 10 tons. The latter converters are not mounted on trucks, but are swung on trunnions and dumped with cranes. These converters, or pots, are placed in rows with one blast pipe serving all, the flue above and tracks leading in from the pre-roasting furnaces placed at one side.

The product of the Huntington-Heberlein roasting may contain up to 3.5% sulphur from a charge originally of 16% sulphur; at Munsterbusch the product contains from 1.5 to 1.8% sulphur. The further advantage of the process is that it makes very little matte in the blast furnaces and that their capacity is doubled.

The Savelsberg process resembles Huntington-Heberlein converting, but dispensing with pre-roasting. Limestone and silica are mixed with the ore and this is charged on top of a fire in the converting pot; as the reaction progresses more material is added until the pot is full. At Munsterbusch one small unit will roast 6 tons in 18 hours.

The Carmichael-Bradford process is used only at Broken Hill, Australia, where gypsum is the diluent and sulphuric acid is made from the gas.

Roasting by suction has been introduced into Europe from America by the owners of the Huntington-Heberlein process, buying the European rights of the Dwight-Lloyd continuous machines. At Bindfeld-hammer, near Stolberg, three annular horizontal continuous machines are in operation. At the same time as the material is roasted the gas from the reaction is saved and sulphuric acid is made by the contact process. Pre-roasting, as for the Huntington-Heberlein process, is first carried out, the cooled material going to the bottom of the grate and other material still red hot from the furnaces is fed on top to start the combustion which progresses downward induced by the suction acting from below the grate. As the feeding is continuous and the discharge is also continuous the operation is easily and cheaply accomplished. Australian concentrates and German galena is diluted with limestone and silica for the roasting mixture, while the original 13% of sulphur is reduced to about 3.5% in the finished product.

The following table is interesting:

Process.	Material	Tonnage per 24 hours.	Ores.	Acid m'fg	Cost.
Continuous rabbling.	Raw ore.	10-30	All galenas.	No.	\$3.25-4.00
H.-H.	Pre-roast	30-35	All galenas.	No.	\$1.00-1.25
Savelsberg.	Raw ore.	12-15	Certain gals.	No.	\$1.00
Continuous suction.	Pre-roast	40-60	All galenas	Yes.	\$0.75

SMALL ELECTRIC FURNACE WITH HEATING ELEMENT OF DUCTILE TUNGSTEN OR DUCTILE MOLYBDENUM.*

By R. WINNE AND C. DANTSIZEN.

There are many uses for a simple furnace which can be run higher than the melting-point of iron, and in which substances can be heated without taking up either carbon or oxygen.

There have been a few special furnaces, electrically heated, which are capable of fulfilling these conditions. Of these, we may mention the following:

1. The iridium-tube furnace. This is very fragile and extremely expensive.
2. A carbon-tube furnace with a porcelain tube inside. This makes an excellent tube-furnace, but calls for special transformers.
3. The platinum foil or wire-bound body of a fire-clay or other refractory. This has been to the average man the most easily accessible and most generally used form of small furnaces for fulfilling the conditions. But it has its limitations. Most serious of these are the fact that the platinum is easily contaminated, and that, except for small sizes, the winding is very expensive. Besides this, much of the work for which these furnaces are used necessitates running the platinum perilously close to its melting-point.
4. The Arsem vacuum furnace¹, in which a graphite heater is used to generate the heat and in which the oxidation is prevented by the vacuum. This introduces carbon when very high temperatures are used.

We will show in the following how two simple types of furnaces have been made with a winding of ductile tungsten or molybdenum. It is now possible to produce these substances in the form of wire or ribbon, and their high melting-points and relatively low cost adapt them admirably to the purpose.

CRUCIBLE FURNACE.

This consists of a helix of ductile tungsten or molybdenum, supported by an alundum tube and protected from oxidation by a hydrogen atmosphere. The container, for the charge to be heated, is a small crucible which is placed inside of the alundum cylinder.

The diagram, Fig. 1, shows the furnace in vertical section. *W* is the heating wire, of ductile tungsten or molybdenum. It is wound on *G*, an alundum cylinder which is molded plain inside and with a helical groove on its outer surface. *A* is a Battersea crucible. *B* is an inverted Battersea of which the bottom has been cut off. *C* is the crucible bottom, used as a cover. The bottom of *A*, as well as the space between *A* and *B*, is filled with powdered silica. Hydrogen is constantly supplied to the furnace through the pipe *D*,

and burns as it escapes from the top of *B*. This gives a protecting atmosphere for both the tungsten or molybdenum winding and the object to be heated. Current is supplied to the winding through two large copper connectors, *E, E*.

An object to be merely heated, as for annealing, may be inserted directly into the alundum cylinder.

Material to be melted is placed in a small crucible, and this is let down into the alundum cylinder. An especially nice crucible to hold the charge is made of pure alumina.

A few dimensions and details of construction may be of interest.

In the apparatus shown in Fig. 1, the Battersea crucibles *A* and *B* are sizes *O* and *J* respectively. The alundum body *G* consists of two cylinders placed end to end. Each is 7.6 cm. high and 5.1 cm. inside diameter, while the distance from one convolution of the helix to the next is .95 cm. These alundum cylinders may be obtained from The Norton Company, of Worcester, Mass. The winding consists of 260 cm. of square molybdenum wire, 1.27 mm. on a side. This wire is first wound on a mandrel 3.8 cm. in diameter,

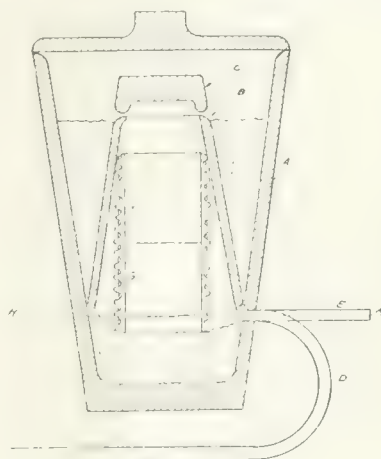


Fig. 1.

and wire and mandrel are then heated up to about 800° C. Upon then releasing the coil, it expands to the diameter of the alundum body, on which it can then be screwed. The copper connectors, *E, E*, are 0.8 cm. in diameter, and the ends of the coil are simply clamped in with set-screws. The copper connectors and the hydrogen inlet tube are held in place in the crucible wall to a mixture of powdered silica and water glass, which at the same time prevents loss of hydrogen.

This furnace can be safely run up to 1700° C. At this temperature it calls for 25 volts and 45 amperes.

We have melted pure iron and made many iron and other alloys in this furnace.

TUBE FURNACE.

This consists of a porcelain or alundum tube, wound with tungsten or molybdenum foil. Around the tube is a tight metal casing filled with powdered silica, and,

*Paper prepared for the 20th General Meeting of the American Electrochemical Society.

¹Transactions American Electrochemical Society, 9, 153-172 (1906).

J. Am. Chem. Soc., 28, 921-935.

to prevent oxidation of the winding, a hydrogen atmosphere is maintained in the casing.

Such a furnace is shown in the diagram, Fig. 2, in vertical length section. *A* is an alundum tube, 2.2 cm. inside diameter and 46 cm. long. The casing *D* is made of thin sheet iron and is oxy-acetylene welded. The winding is a molybdenum ribbon, 0.184 mm. thick, 2.54 mm. wide and 445 cm. long. *I, I*, are heavy copper leads, fastened to the ends of the coil by twisting the latter about them. *C* is the powdered silica packing, *F, F*, is some asbestos wool, used to prevent the escape of hydrogen at the ends of the casing. *H* and *G* are hydrogen inlet and outlet tubes respectively. *E* is a rubber stopper.

At this temperature it calls for 80 volts and 14.3 amperes.

The two furnaces above described are adapted to

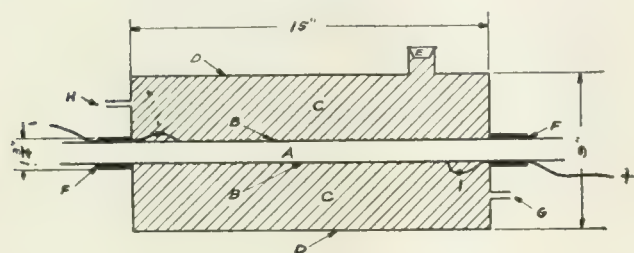


Fig. 2.

heating the charge in an atmosphere of hydrogen. In case it were desirable to heat in an oxidizing atmosphere, it would be necessary to replace the very porous alundum by porcelain, as otherwise the furnace winding would become oxidized.

A brochure calling attention to its high-grade steel has been issued by the Vulcan Crucible Co., Aliquippa, Pa. Both the crucible and open-hearth furnaces used by this company are operated with natural gas which, on account of its freedom from sulphur, is an ideal fuel for the melting of steel. The acid open-hearth steel which it makes does not contain more than 0.030 phosphorus and sulphur. The ignots are bloomed under a forging hammer and are rigidly inspected for any defects before finishing. The company manufactures chrome vanadium steel for all purposes requiring great strength and endurance, especially for automobile parts. Hecla vanadium steel is made for case hardening, for oil-hardened gears and for tools. The brand known as Wolfram high-speed steel contains a high percentage of vanadium and is carried in all sizes, annealed and unannealed. The brand known as Vulcan special vanadium steel contains a definite percentage of vanadium and is designed for all purposes where the finest grades of carbon steel are used, and is carried in all sizes, annealed. In addition to these classes of steel, the company manufactures tungsten magnet steels.

THE DIRECT PRODUCTION OF MOLYBDENUM STEEL IN THE ELECTRIC FURNACE.*

By E. T. DITTUS AND R. G. BOWMAN.

It is the intention of the authors to present in the following paper a brief resume of the properties, uses and methods of production of molybdenum steel, and to describe in detail a series of experiments on the production of molybdenum steel, direct from iron ore and molybdenite, in the electric furnace.

The use of a sulphide ore of so active a metal as molybdenum in connection with the manufacture of steel presents a number of problems. Foremost among these are the complete reduction of the molybdenum without serious loss, the diffusion of the molybdenum through the steel to form a homogeneous product and the elimination of the sulphur from the steel. The process employed was one based on a reaction described by F. M. Becket in U. S. Patent 855,157, and is believed to be new.

The authors wish to express their hearty appreciation and thanks to Professor W. G. Haldane and others for the encouragement and assistance they have given in carrying out the experimental investigations.

The desirable properties of molybdenum steel and the expense involved in its manufacture by the methods at present in use, led the authors to search for some reaction upon which could be based a process for its direct production from its most common ore. Without such a process any extended use of molybdenum steel, instead of tungsten steel, is very improbable. The reaction described by Becket appeared the most simple and was, therefore, made the subject of the investigations.

The production of steel direct from ore in the electric furnace is easily accomplished where the ore is pure. Certain impurities, particularly sulphur, are, however, very difficult to remove from the metal. Small amounts may be removed by the use of a basic slag but this method is limited in its application since the fusibility of the slag decreased rapidly with increasing basicity. For the reduction of molybdenite in the presence of molten iron it is necessary to have present some substance which has a greater affinity for sulphur than either molybdenum or iron. The compound formed by the desulphurizing agent and the sulphur must either pass into the slag or be volatilized as soon as formed. The two metals which seem best adapted for use as desulphurizers, under these conditions, are manganese and silicon.

Manganese in the form of metal or as the ferro alloy reacts with sulphides at high temperatures according to the following reaction:



*Paper presented at the Twentieth General Meeting of the American Electrochemical Society, in Toronto, Canada, Sept. 21-23, 1911. This paper includes extracts from a thesis presented to the Colorado School of Mines.

The manganese sulphide forms a slag resembling iron sulphide. If the above reaction takes place in a bath of molten steel or iron the resulting metal is apt to contain small included masses of manganese sulphide. Sulphur in this form has little effect on the properties of steel. This reaction might be applied to the production of alloy steels, such as molybdenum steel, by adding a mixture of molybdenite and ferro-manganese to the bath of molten steel just before tapping. This would result in the formation of ferro-molybdenum and manganese sulphide; the former alloying with the steel and the latter passing into the slag.

Silicon reacts with sulphides to form silicon sulphide, SiS_2 and liberates the metal of the original sulphide. This is the reaction described by Becket.¹ The reaction has been investigated by Fielding² who produced a yellow powder which sublimed at $1,500^\circ \text{C}$.

of the absence of slag, which in the furnace would remove small amounts of sulphur which might tend to pass into the steel.

The experimental work of this thesis was undertaken with a view to determine the feasibility of applying the Becket process to the production of molybdenum steel direct from iron ore and molybdenite in the electric furnace. The problems presented were: First, the design and construction of a suitable furnace; second, the production of a carbon steel by direct smelting from iron ore; third, the addition of molybdenum to the steel and the elimination of the sulphur of the molybdenite from the bath of metal. Of these the third presents by far the greatest difficulty.

To accomplish the results desired it was necessary that the furnace fulfill the following requirements: The crucible walls should be of a neutral or basic material, free from carbon; the furnace should be easily

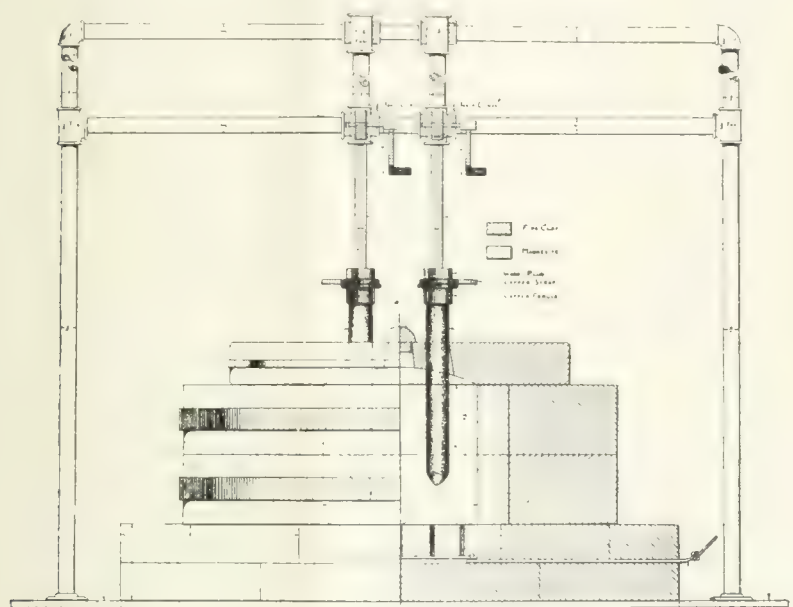


Fig. 1.

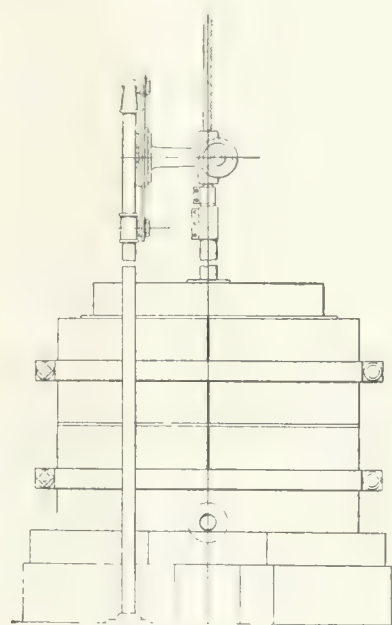


Fig. 2.

and which decomposed water with the formation of H_2S and silicic acid. This compound did not correspond to the formula SiS_2 . Sabatier³ describes a somewhat similar compound and suggests the formula Si_3S_4 . The heat of formation of SiS_2 is given by Sabatier³ as +40.4.

This reaction might be applied to the production of molybdenum steel in the same manner as the manganese reaction described above. The silicon, in the form of ferro-silicon, and the molybdenite should be reduced to powder, intimately mixed in the proper proportion to produce the reaction and added to the steel in the furnace before tapping. The mixture might be added in a soft iron tube or in small briquettes made up with a binder of sodium silicate. Addition in the ladle during tapping would probably result in raising the sulphur content of the metal on account

charged and operated; the electrodes should be capable of a variety of adjustments; the furnace should be capable of being readily dismantled for the purpose of relining or making alterations. Since the best method of heating could be determined only by experiment, it was thought best to design a furnace capable of being operated as a Girod, or with slight alteration, as a Heroult furnace.

With the above mentioned requirements in view, two furnaces were designed. These were essentially the same in principle but differed in the proportions of the crucible and in certain minor details. A detailed description of the design illustrated in Fig. 1 will suffice, since this was the furnace finally adopted and used in the investigations.

The crucible is elliptical in cross section and measures 6 in. x 9 in. x 8 in. (15 x 23 x 20 cm.) deep. The total volume of the crucible is about 360 cu. in. (5.5 litres), the volume of the smelting zone is about 90

³Comptes Rendus, 90, 819.

¹Elec. Chem. & Met. Industry, Aug. 1909.

²Bull. de Soc. Chem., Paris 2, 38, 134.

cu. in. (1.4 litres). The walls have a slight bosh to effect a concentration of heat at the base of the crucible. The crucible lining is of burned magnesite obtained by crushing magnesite brick to pass 10 mesh (2.5 mm.). The magnesite was mixed with tar to form a paste and rammed in hot around a central wooden form. The tar was burned out in the heating up of the furnace.

The walls are built up of eight fire-brick sections of special design, luted together with fire-clay and encircled by steel bands. These rest on a foundation made up of two courses of standard fire-brick, with the joints filled with fire-clay.

The cover for the crucible is a solid elliptical fire-brick section 17 in. x 20 in. and $2\frac{1}{2}$ in. (43 x 50 x 53 cm.) thick, encircled by a steel band. The electrodes enter through the slot in the center of the cover. This slot admits of considerable lateral or angular movement of the electrodes. The under side of the cover is slightly concave. An asbestos gasket is placed under the cover where it rests on the top of the furnace, and small asbestos washers encircle the electrodes where they pass through the cover.

The tap-hole is $1\frac{1}{4}$ in. (3 cm.) in diameter, and is well rounded at its point of entrance to the crucible. A small hole 1 in. (2.5 cm.) in diameter extends diagonally from the top of the outer wall to the smelting zone for the purpose of pyrometer readings and for the escape of gases.

The hearth electrode consists of six $\frac{1}{2}$ -in. (1.3 cm.) rods of Swedish iron screwed into a $\frac{3}{8}$ -in. (0.9 cm.) plate of the same material. The plate is embedded in the lining of the bottom of the crucible in such a position that the upper ends of the rods are flush with the inner surface. A copper strap $\frac{3}{16}$ in. (0.5 cm.) thick and 1.5 in. (4 cm.) wide, bolted to the plate, extends outside the furnace for connection to the source of current.

The two movable electrodes are supported on a frame above the furnace. The supporting frame is entirely independent of the furnace proper and is therefore unaffected by expansion or contraction of the walls. The electrodes are cylindrical graphite rods 1.25 in. (3 cm.) in diameter and 11 in. (28 cm.) long. Each electrode is attached to a brass rod by means of a collar clamp and cotter pin. The rod carries a rack which meshes with a pinion mounted on the electrode support. The electrode supports are mounted to slide from right to left on the supporting frame. This facilitates adjustment of the electrodes while the furnace is in operation, and also allows the electrodes to be pushed back out of the way when relining the crucible. The support is so constructed that the electrode may be swung about it as a center and inclined at any angle. Ordinary pipe fittings were used as much as possible in the construction of the frame and supports, since these give a light, strong construction, and require no special castings or forgings. An electrode

support built entirely of pipe and fittings is illustrated in Fig. 2.

The furnace illustrated in Figs. 2 and 3 differs from that described above in having a crucible of double the depth described, i. e., 16 in. (40 cm.) and a different arrangement of the rods in the bottom of the crucible. This furnace was designed to operate continuously with a column of cold charge resting on the molten material, as in a shaft furnace.

This type of furnace may be operated as a Girod steel furnace by connecting the two movable electrodes in parallel with one side of the circuit and the hearth

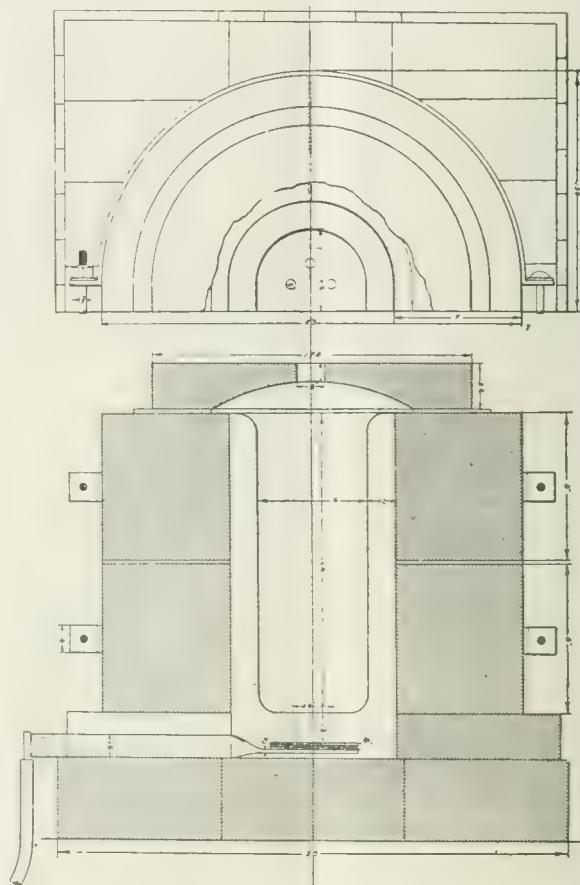


Fig. 3.

electrode with the opposite side. The movable electrodes may then be placed parallel vertically, thus forming two arcs, or set to converge to a single central point forming but one arc. To operate the furnace as a Heroult steel furnace, the points of the rods in the hearth electrode are covered with a layer of magnesite, and the two movable electrodes are connected in series with the circuit. The magnesite covering for the hearth electrode need not be of any great thickness, since there is but a slight tendency for the current to cross through the iron plate after the charge becomes molten.

The furnace is charged at the beginning of a run by removing the cover and distributing the charge with a trough or funnel. Additional material may be added through the opening in the cover while the furnace is running.

The small amount of coke in the charge used was found to be sufficient to make the cold charge slightly conducting, the resistance rapidly decreases with rise of temperature, and the furnace may thus be readily started on a cold charge, providing the walls of the crucible are hot. The electrodes are forced down through the charge to within a short distance of the bottom of the crucible at starting. As the charge becomes hot, the electrodes are gradually raised until the charge is reduced and the arcs play between the ends of the electrodes and the surface of the slag. With the current available it was found impossible to maintain arcs below both electrodes simultaneously for any length of time. The best results were obtained by raising and lowering the electrodes alternately at intervals of two or three minutes, which cause the arcs to alternate from one electrode to the other.

The capacity of the furnace as described is about 1,000 gm. of metal per charge, or approximately 2,000 gm. of raw charge. A charge of 1,000 gm. was employed in several runs. This yielded 500 gm. of metal, but this amount is too small to make clean tapping possible.

Single-phase alternating current at 60 cycles, 25 volts, was employed. The amperage varied throughout the run, but averaged 250 with the arcs running steadily.

An opening for pyrometer readings was provided in the furnace, but no readings were taken. In a furnace of this size a variation of temperature from cold ore to the maximum smelting heat may be found in a space of a few inches; such a variation would make pyrometer readings irregular and of little value.

The generator used in supplying current for the experiment was a General Electric alternating current composite-wound, having both slip rings and a rectifier commutator for supplying direct current to the series field. The name plate gives the following data: No. 2044, Type A, Class 10-30-1500. No load voltage 1040, Full load voltage 1155, Thompson-Houston System.

The transformers were made by the General Electric Co. No. 423052, Type H, Cycles 60, Form G, Volts 1,200, 2,400, 120,240. The transformation ratio was 10-1. The primaries were connected in series with the line and the two coils in parallel with each other, the secondaries all in parallel with the line. This combination gave the best results. The amperage ranged from 0-500 with an average of 250, the voltage averaged 25.

The switchboard consisted of two ammeters in parallel with each other and in series with the line, a voltmeter in parallel, a circuit breaker, and recording wattmeter. The ammeters were made by Westinghouse Electric Co. 7,200 alt. circuit, cap. 300 amps. The voltmeter was a Thompson Portable machine, the circuit breaker an I. T. E. type W. 500 amps. The wattmeter was a Thompson recording No. 104114, Type M, Class 100-200 Form D₃.

During all the work the generator was run with the series coil short circuited across the brushes. This protected the machine because any overload or short would cause the voltage to drop due to the armature reaction and reactance. A dead short would thus cause the voltage to drop to zero and thus no harm could be done to the machine.

The connections of the various pieces of apparatus are illustrated in the accompanying diagram, Fig. 4.

The iron ore employed was a good grade of hematite from the mines of the Colorado Fuel and Iron Co. at Sunrise, Wyo. Two separate lots were used, the analyses of which were as follows:

Fe	66.06	per cent
Al ₂ O ₃	1.52	"
SiO ₂	4.12	"
P	0.04	"
S	0.04	"

The ore was crushed to pass eight mesh (3 mm.) and the fines were retained.

A quantity of very pure molybdenite from Yorkés Peninsula, Australia, was obtained from the Foote Mineral Co.

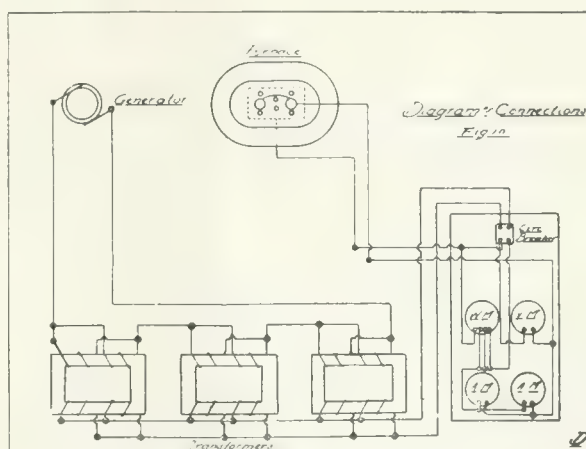


Fig. 4.

Analysis showed the material to be practically pure MoS₂ with only a trace of impurity in the form of copper sulphide.

Two lots of ferro-silicon were used, the analyses of which were as follows:

Fe	49.70	per cent
Si	50.00	"
S	0.005	"
C	0.20	"

The ferro silicon and molybdenite were ground to pass thirty mesh (0.85 mm.) and intimately mixed in the proper proportions to bring about the reaction:



The proper amount of this mixture to give the desired molybdenum content in the steel was enclosed in a small paper tube and added to the bath of metal just before reduction was complete.

The coke used was from Trinidad, Colo., and was obtained from the North American Smelting Co., of Golden, Colo., from whom the analysis was also obtained.

Analysis:

Fixed carbon.....	77.70	per cent
Vol. and combust. mat....	2.80	"
Ash	18.66	"
FeO	1.23	"
Al ₂ O	1.41	"
CaO	1.30	"
SiO ₂	6.59	"
S	0.47	"
Moisture	0.84	"

The coke was crushed to pass eight mesh (3 mm.) and the fines retained.

Ordinary builders' lime was used as flux. The analysis was as follows:

CaO	92.67	per cent
MgO	..	"
FeO, AlO	1.90	"
SiO ₂	1.31	"
P	0.06	"

The lime was crushed to pass eight mesh (3 mm.) and the fines retained.

RECORD OF FURNACE RESULTS.

Run No. 5.

Charge:

Iron ore	3,025	gm.
Coke	660	"
Lime	412	"
Calculated iron content	2,000	"
Length of run	1 hr. 30 min.	
Mean amperes	200	
Mean volts	25	
K. W.	5	
Metal obtained	1,700	gm.
Calculated molybdenum con.	2.5	per cent

Iron turnings were spread over the bottom of the crucible and covered by a layer of coke, lime and silica to form slag and protect the iron. The metal melted in 25 minutes but was kept molten in the furnace for 15 minutes after fusion in order to burn out the tar from new portions of the lining. This metal was tapped cleanly and a charge sufficient for 500 gm. of metal was added. This reduced in 20 minutes. Molybdenite-ferro-silicon mixture sufficient to give a content of 5 per cent Mo in the metal was added in a paper tube just before reduction was complete. The tap hole was opened but the metal was level with the tap and only slag flowed out. The tap hole was again plugged and a charge sufficient to give 1,000 gm. of metal was charged.

Reduction was rapid and the charge was entirely reduced in 30 minutes. Molybdenite mixture sufficient to give a molybdenum content of 2.5 per cent in the metal was added just before final reduction of the charge. Since the bottom of the crucible appeared to be sinking constantly, another charge equivalent to 500 gm. of metal was added and reduced in 15 minutes. This gave a total of 2,000 gm. of metal (calculated) in the furnace, with sufficient molybdenum to yield 2.5 per cent in the total.

The tap hole was opened and slag flowed out freely but no metal was obtained. The current was shut off and the furnace allowed to cool. A solid mass of metal level with the tap hole was found in the bottom of the crucible.

The furnace was taken down and the entire lining and bottom electrode removed. A solid mass of metal weighing 1,700 gm. was obtained. The metal was malleable and tough, and was broken with great difficulty.

When broken across the center the fracture was very fine grained and dense at the center of the mass and coarsely crystalline and full of small blow holes at its outer edges.

A section of the central portion of the mass, polished and etched with picric acid showed a network of a bright white constituent, probably a double carbide of molybdenum and iron. A section in the coarsely crystalline portion of the mass showed a coarsely granular structure:

Analysis:

C	0.62	per cent
Si	0.91	"
Mo	1.15	"
P	0.08	"
S	0.37	"

Run. No. 6.

Charge:

Iron ore	756	gm.
Coke	163	"
Lime	103	"
Calculated iron content	500	"
Calculated molybdenum content	2	per cent
Length of run	30	min.
Mean amperes	300	
Mean volts	25	
K. W.	7.5	
Metal tapped	—	

Description of run:

The furnace was rebuilt and arranged as in Run 5. The tap hole was given a slightly greater inclination.

Iron turnings were placed on the bottom of the crucible and were melted in 30 minutes. The tar in the new lining burned out very slowly and the molten metal was therefore allowed to remain in the furnace for one and one-half hours with the arcs running. When the greater part of the tar was burned out the metal was tapped and the charge added.

Reduction was rapid and molybdenum mixture sufficient to give a molybdenum content of 2 per cent was added. The walls of the crucible were crumbling and the crucible was rapidly becoming too large to heat properly. The magnesite from the lining made the fused charge thick and pasty. An attempt was made to tap the metal but nothing was obtained. The crucible was now too large to work properly but since the bottom seemed to be sinking it was thought advisable to fill the crucible up to the proper level by the reduction of a large charge of ore. A charge corresponding to 2,000 gm. of metal was added in small amounts with the arcs running. Reduction was rapid but not uniform owing to the increased size of the crucible. By continual poking of the charge the whole mass was finally fused but was not sufficiently liquid to tap. The furnace was allowed to cool and a mass of metal level with the tap hole was found embedded firmly in the bottom. A small portion of this metal was removed with tongs and proved to be very soft and malleable and easily sawn or cut.

A polished section etched with picric acid showed a network of bright white lines in a ground of pearlite.

Analysis:

C	1.14	per cent
Si	1.23	"
P	0.03	"
S	0.03	"
Mo	0.45	"

Run No. 7.	
Charge:	
Iron ore	1,512 gm.
Coke	326 "
Lime	206 "
Calculated iron content.....	1,000 "
Length of run.....	60 min.
Mean amperes	300
Mean volts	25
K. W.	7.5
Metal tapped	—

The crucible was re-lined. The electrodes were partially consumed and, since no more of the same size were at hand, were removed and replaced by 1½-in. (3 cm.) rods. The arcs were started on the mass of metal remaining from Run 6. When the furnace was hot and this metal was molten, a charge was added and quickly reduced. The tap-hole was opened, but only a portion of the metal was obtained. The charge was then found to have formed a scaffold across the crucible. It was found impossible to smelt down the semi-fused mass forming the scaffold with the small electrodes at hand; the furnace was therefore allowed to cool and the material picked out.

The metal obtained in tapping had a coarse crystalline structure and was extremely hard and tough.

A polished section etched with picric acid and magnified 180 diameters showed irregular branching figures of a bright white constituent in a ground of granular pearlite.

Analysis:	
C	0.71 per cent
Si	2.14 "
(Large amount of scale in sample.)	
P	0.004 "
S	0.025 "
Mo	1.95 "

Run No. 8.	
Charge:	
Iron ore	1,512 gm.
Coke	326 "
Lime	206 "
Calculated iron content.....	1,000 "
Calculated Mo. content.....	.5 per cent
Length of run	60 min.
Mean amperes	300
Mean volts	25
K. W.	7.5
Electrode consumption	65.0 gm.
Metal tapped	—

The furnace was started on a small amount of wrought iron punchings, covered by a little powdered coke. These melted rapidly, and a small charge was thrown in in small portions with the arcs running freely. When the metal was all melted, the entire charge was added and the electrodes embedded in the cold material. The small rods employed in Experiment 7 had been replaced by full-size electrodes.

Reduction of the charge was rather slow. At the end of 60 minutes an attempt was made to tap. Slag flowed out freely and filled the mould, but the metal would not flow. The tap-hole was closed and a strong arc started to melt the mass of metal. Upon tapping, a portion of the metal was obtained. This was tough and malleable and had a very fine-grained structure.

The molybdenum mixture was added as usual just before reduction was complete. When the arc was started on the surface of the metal after tapping the slag, a white sublimate formed on the electrodes and the under surface of the cover. This was probably

MoO₃, formed by the oxidation of the metallic molybdenum in the steel, which was unprotected by slag.

A polished section of the metal tapped, when etched with picric acid and examined under a magnification of 180 diameters, showed a coarse network of the characteristic bright white constituent in a mass of pearlite.

Analysis:	
C	0.92 per cent
P	0.043 "
Si	1.10 "
Mo	2.15 "
S	0.036 "

Run No. 9.	
Charge:	
Iron ore	756 gm.
Coke	163 "
Lime	163 "
Calculated iron content.....	500 "
Length of run.....	60 min.
Mean amperes	300
Mean volts	25
K. W.	7.5
Metal tapped	—
Calculated molybdenum content.....	0.6 per cent

The furnace was started on the metal remaining in the crucible. When the crucible was well heated, the charge was added, and reduction was rapid. At the end of 60 minutes an attempt was made to tap. Slag flowed freely, but the metal would not flow. This was due to a dam of metal formed just outside the tap-hole from a leak in the clay plug.

The furnace was allowed to cool, and a mass of metal weighing about 450 gm. was taken from the crucible with tongs. This metal was very malleable and tough and could be easily sawn and cut.

A polished section, etched with picric acid and examined under a magnification of 180 diameters, showed the characteristic network of double carbide and in addition a mass of needles of a white constituent resembling the carbide.

Analysis:	
C	0.54 per cent
Si	1.65 "
Mo	0.45 "
P	0.07 "
S	0.05 "

Run No. 10.	
Molybdenum steel from molybdenite concentrates.	
Analysis of concentrates (Woods Flotation Process):	
Mo	11.10 per cent
Co	0.72 "
S	15.00 "
SiO ₂	65.00 "

Molybdenum mixture (equivalent to 0.66 per cent Mo in calculated iron).

Concentrates	100 gm.
Si-Fe	9.28 "
Lime	65.00 "

Charge:
As in previous runs, equivalent to 1,500 grams calculated iron.

Length of run	40 min.
Mean amperes	250
Mean volts	25
K. W.	6.25
Metal tapped	1,370 gm.

Furnace started on metal in bottom. Charge added in two portions and reduced rapidly. A tap was attempted after the addition of the first charge, but the metal would not flow. The molybdenum mixture was added in two equal portions and was reduced almost instantly. The tap was clean and the ingot obtained was sound and free from large blow holes. The in-

crease in molybdenum content in the metal over that calculated was probably due to a concentration of the molybdenum in the metal reduced.

A polished section, taken from the center of the ingot, when etched with picric acid and examined under a magnification of 180 diameters, showed a fine network of the double carbide and small isolated patches of the same constituent all in a ground mass of pearlite.

Analysis:

C	0.70	per cent
SiO ₂	1.20	"
Mo	0.70	"
P	0.10	"
S	0.45	"

CONCLUSIONS.

Since the results of the experiments herein described, with the exception of runs Nos. 5 and 10, were qualitative rather than quantitative, further experiments would be necessary in order to determine the loss and distribution of molybdenum.

Since no analyses were made upon the slag, it is impossible to determine what proportions of the sulphur were removed by the ferro-silicon and the slag.

(1) Molybdenum steel can be made in the electric furnace by the direct reduction of iron ore and the addition of molybdenum in the form of molybdenite, MoS₂.

(2) Molybdenum steel of low sulphur content can be produced from molybdenite by the use of ferro-silicon as a desulphurizer.

(3) Molybdenum steel of low sulphur content can be produced from molybdenite in the form of low-grade concentrates by the use of ferro-silicon as desulphurizer.

With regard to the design of a furnace for small scale operations, the foregoing experiments would indicate that:

(1) The Girod principle as used was superior to the Heroult.

(2) A tilting furnace would be more effective than a stationary furnace.

(3) The tap hole on a stationary furnace should be made short, with a steep inclination. Ample provision for heating the tap hole should be made.

(4) Tar does not make a satisfactory binding material for crushed magnesite.

Sugar beets in Canada from the 12,000 acres this season are expected to yield 9,000 long tons of sugar, against 7,771 tons from 10,500 acres last year. Canada imported 574,108,164 lbs. of sugar, valued at \$15,110,740, in the fiscal year ended March 31, 1911.

The value of the mica exported from India during the fiscal year 1908-9 was \$634,170, as against \$774,890 in 1909-10 and \$944,915 in 1910-11. The mica mined in India belongs entirely to the variety known as muscovite.

As a result of the experiments which have been carried on in Europe and the Sudan during the last three or four years, says the Egyptian Gazette, the Sudan Government has definitely granted a concession, conferring a monopoly for the manufacture of solid fuel from sudd for 17 years. The Government is to receive 10 per cent rebate on all fuel supplied at the price charged to public consumers. The Government is further to receive a dead rent of \$1,250 per annum, or a commission of 5 per cent on the net profits of the company, whichever shall be the greater. The Government has allocated to the concessionaries the first 150 kilometers (93 miles) of the Bahr el Gebel, starting from Lake No, in which they are to have the sole right of cutting papyrus, um-soof, and other aquatic growth constituting what is commonly known as sudd. The Government further gives a site of 25 acres for the purpose of erecting a factory, etc., in any place which may be selected by the concessionaries, where such land may be available. The concession provides that the concessionaries shall supply the needs of all Government services in preference to private consumers, and stipulates for a minimum output of 25,000 tons per annum. It is understood that the plant to be put down is to have an output of double that amount.

The total value of the gold, silver, copper and lead from the state of Washington in 1910, according to C. N. Gerry, of the United States Geological Survey, was \$968,249 against \$448,966 in 1909. The output for 1910 was the largest in gold, silver, and lead and the smallest in copper for six years. The production of gold in 1910 was \$788,145 against \$362,051 in 1909; that of silver was 205,345 fine ounces, valued at \$110,886, against 79,488 ounces, valued at \$41,334; that of copper 86,918 lbs., valued at \$11,038, against 255,134 lbs., valued at \$33,167; and that of lead 1,322,287 lbs., valued at \$58,180, against 288,700 lbs., valued at \$12,414. The number of producing mines in Washington increased from 11 placers and 35 deep mines in 1909 to 21 placers and 55 deep mines in 1910. The placers produced, however, only \$3,859 in gold in 1910. The deep mines produced 59,209 tons of ore in 1910, an increase of 22,901 tons over the output of 1909. The Republic district, in Ferry County, produced the greater part of the gold and silver in Washington, and mines in Stevens County produced nearly all the copper and lead. The Republic district alone produced gold, silver, copper and lead valued at \$813,686 in 1910.

Rubber shipments over the Pan American Railway of Mexico during the fiscal year ended June 30, 1911, are given by Consul A. W. Brickwood, of Tapachula, as follows: To United States, 107,280 lbs.; United Kingdom, 15,182 lbs.; Germany, 127,623 lbs.; France, 12,667 lbs.



VOLUMETRIC DETERMINATION OF MERCURY.*

By CARL E. SMITH.†

Abstracts of a report by a committee of the Division of Pharmaceutical Chemistry of the American Chemical Society have appeared in recent journals (see *American Journal of Pharmacy*, 1911, v. 83, p. 186), the work reported on being a comparison of various methods for the determination of mercury in the medicinal compounds of this metal.

The purpose of these notes is to relate some recent experiences with two of these methods, which are probably the simplest and most reliable of those mentioned in the report, the Hempel method for mercurous and the Rupp method for mercuric compounds. It is desired also to call attention to one other, which is perhaps the best volumetric method available for some purposes.

The writer's experience with the Hempel method, in its application to calomel and mercurous iodide, corroborates to a considerable extent the results of the committee. The objectionable feature of it, noted by several of the members, that long-continued shaking is needed to bring the salt into solution, may be eliminated, it was found, by the simple change of adding the iodoine solution first, instead of the potassium iodide. When this is done and the mixture at once shaken vigorously in a stoppered flask, solution is effected very quickly. The final result is the same in either case, as comparative trials have shown. It seems advisable, also, to increase the quantity of sample, to lessen the experimental error, in the case of calomel to about 1 Gm. The most satisfactory results were obtained when working in the following manner: To about 1 Gm. of calomel, accurately weighed, contained in a glass-stoppered 300 c.c. Erlenmeyer flask, add 50 c.c. of $N/10$ iodine solution. Mix by rotating until the salt is thoroughly moistened, then add a solution of 2 Gm. of potassium iodide in 10 c.c. of water and at once shake the stoppered flask vigorously until solution is complete. Titrate the excess of iodine with $N/10$ sodium thiosulphate, using starch solution as indicator, adding the latter when the liquid is nearly decolorized. Each c.c. of $N/10$ iodine solution consumed corresponds to 0.02355 Gm. of mercurous chloride.

A sample of calomel, practically chemically pure, assayed by this method 99.5 to 100.0 per cent, the average of 6 determinations being 99.7 per cent, with

some variation in the working details. Practically the same figures were obtained with yellow mercurous iodide containing slight traces of impurities.

The criticism of the Rupp method, that reduction of the mercury is not complete within a reasonable time without the use of heat and that, when heat is employed, the mercury is not easily dissolved afterwards, is well founded, if Rupp's directions in outlining the method (*Berichte d. d. ch. Ges.*, 1906, v. 39, p. 3702) be taken literally as regards the quantity of alkali to be used. While his directions lead to the inference that only enough is required to combine with the acids formed by the reaction, his figures in the same paper show that he used a decided excess, not less than 10 c.c. of normal solution for 0.2 Gm. of mercuric chloride. The divergent opinions regarding this method by the members of the committee are doubtless due chiefly to differences in the quantities of alkali used. When the above mentioned proportions are taken, the reaction is almost instantaneous, unless the solution is excessively diluted, and may safely be regarded as complete within 5 minutes, without heating. It will do no harm to use a still larger excess of alkali or to let the mixture stand longer. Shaken in a stoppered flask with the iodine solution, the precipitate is then dissolved very readily. A large excess of acetic acid is to be avoided, as it tends to make the result too low, which is also the experience of Mr. L. D. Havenhill of the committee. The best results were obtained by carrying out the details as follows: Dissolve about 0.5 Gm. of powdered mercuric chloride, accurately weighed, contained in a glass-stoppered 300 c.c. Erlenmeyer flask, in a solution of 2 Gm. of potassium iodide in 10 c.c. of water. Add 25 c.c. of normal caustic alkali solution and 6 c.c. of 40 per cent formaldehyde solution. Mix by swirling the flask occasionally during 10 minutes, then acidulate with about 5 c.c. of 36 per cent acetic acid. Add 50 c.c. of $N/10$ iodine solution and shake vigorously in the stoppered flask until the mercury is dissolved. Titrate the excess of iodine with $N/10$ sodium thiosulphate solution, adding starch solution when the liquid is nearly decolorized. Each c.c. of $N/10$ iodine solution corresponds to 0.01355 Gm. of mercuric chloride.

Chemically pure mercuric chloride gave by this method 99.8 to 100.3 per cent, with an average of 100.1 per cent in 5 determinations. Similar results were obtained with mercuric iodide, oxide, ammoniated mercury and mixtures of mercuric chloride and ammonium chloride colored with aniline dyes. It is the official method of the German Pharmacopœia, 5th

*From the *American Journal of Pharmacy*.

†Laboratory of the Powers-Weightman-Rosengarten Co., Philadelphia.

revision, 1910, for the assay of mercuric chloride tablets and ointment of ammoniated mercury.

The third method alluded to above consists in the titration of mercuric compounds, in nitric acid solution, with sulphocyanate in exactly the same way as the titration of silver. It is not applicable in presence of chlorides and probably not in presence of other halogens. It was first made serviceable for accurate work by R. Cohn (*Ber. d. d. chem. Ges.*, 1901, v. 34, p. 3502) and simplified by Rupp and Kraus (*Ibid.*, 1902, v. 35, p. 2015). For illustrations of its application see below. The writer has no personal experience with this method, but his associates have found it accurate and useful for the assay of technical mercuric oxide containing iron. The German Pharmacopœia uses it for the assay of several galenical preparations. For the convenience of any readers of this journal interested in the subject, who may not have access to this book, the assay methods for mercury preparations prescribed therein are given here:

Mercuric Chloride Tablets.—Composed of equal parts of mercuric chloride and sodium chloride, colored with aniline dye. Dissolve 2 tablets of about 1 Gm. each, accurately weighed, in water and dilute to 100 c.c. In 20 c.c. of the solution dissolve 1 Gm. of potassium iodide, add 10 c.c. of a 15 per cent solution of potassium hydrate and 3 c.c. of a 40 per cent solution of formaldehyde with 10 c.c. of water. After one minute add 25 c.c. of 30 per cent acetic acid, 25 c.c. of N/10 iodine solution, and shake until the mercury is dissolved. Titrate the excess of iodine with N/10 sodium thiosulphate solution, using starch solution as indicator. Each c.c. of N/10 iodine solution consumed corresponds to 0.01355 Gm. of mercuric chloride.

Ointment of Ammoniated Mercury.—Consists of 10 per cent of ammoniated mercury and white vaseline. Heat about 5 Gm. of the ointment, accurately weighed, in a small flask with 25 c.c. of 12.5 per cent hydrochloric acid on a water bath. Mix frequently by rotating the flask during 10 minutes. Pour the acid liquid into a 100 c.c. flask, rinse the vaseline and flask repeatedly with water and dilute the combined liquid and washings to 100 c.c. Transfer 25 c.c. to a glass-stoppered flask, add 1 Gm. of potassium iodide and proceed further as directed for the assay of mercuric chloride tablets. Each c.c. of N/10 iodine solution consumed corresponds to 0.01257 Gm., approximately, of ammoniated mercury.

Mercuric Salicylate.—Contains about 55 per cent of mercury. Dissolve about 0.3 Gm. of the salt, accurately weighed, in dilute sodium hydrate solution, acidulate with acetic acid, and add 25 c.c. of N/10 iodine solution. Let the mixture stand in a closed flask for 3 hours at room temperature, rotating it occasionally. Titrate the excess of iodine with N/10 sodium thiosulphate solution. Each c.c. of N/10 iodine solution consumed corresponds to 0.0100 Gm. of mercury.

Mercury Plaster.—Contains 20 per cent of mercury; made with lead plaster, wool fat and yellow wax. Heat about 3 Gm. of the plaster mass, accurately weighed, with 20 c.c. of nitric acid, sp. gr. 1.38 to 1.40, in a flask having a wide neck and connected with a reflux condenser. Heat about 10 minutes or until mercury globules are no longer visible in the sandy deposit of lead nitrate, add 25 c.c. of water and heat again until the fat has separated, leaving the aqueous layer clear. Cool and pour the solution through a tuft of absorbent cotton into a 100 c.c. flask. Break up the disk of fat, rinse it and the flask with 4 or 5 portions of 5 c.c. each of water and to the combined liquids add potassium permanganate until permanently red or until brown flakes separate. Decolorize or clarify the solution by addition of ferrous sulphate solution and dilute to 100 c.c. To 25 c.c. of the solution add 2 c.c. of a 10 per cent solution of ferric alum and titrate with N/10 ammonium sulphocyanate. Each c.c. consumed corresponds to 0.01000 Gm. of mercury.

Mercury Ointment.—Contains 30 per cent of mercury; made with wool fat, peanut oil, lard, and mutton tallow. Proceed as directed for the assay of mercury plaster, using about 2 Gm. of the ointment and 20 c.c. of nitric acid.

Ointment of Red Mercuric Oxide.—Contains 10 per cent of mercuric oxide; made with white vaseline. Proceed as directed for the assay of mercury plaster, using about 5 Gm. of the ointment and 20 c.c. of nitric acid. Continue heating until the red color of the mercuric oxide has disappeared. Each c.c. of N/10 ammonium sulphocyanate solution corresponds to 0.0108 Gm. of mercuric oxide.

The factors given throughout these notes are based on the atomic weights having O=16 as the standard. If the volumetric solutions used are made by the H=1 standard, either the factors or the final results should be multiplied by 0.992.

The production of copper in Idaho decreased from 9,115,489 lbs., valued at \$1,185,014, in 1909, to 7,037,292 lbs., valued at \$893,736, in 1910, but the production of Custer County increased from 90,461 lbs. to 919,648 lbs., as a result of the operations of the White Knob mine. True copper ores yielded 5,912,392 lbs. and nearly all the remainder was derived from the lead ores of Shoshone County. The production of Blaine of the Gilmore and Pittsburgh Railroad, extending from Armstead, Montana, to Salmon, Idaho. The Yreka district, with a yield of 109,899,539 lbs., was much the largest producer in Shoshone County and in the state. The lead output of the Hunter district was 34,319,919 lbs., and that of the Leland district 72,840,963 lbs.

The production of gold, silver, copper and zinc in Tennessee in 1910 was valued at \$2,289,731, a decrease of \$305,619 from the corresponding value for 1909.

COLORIMETRIC TEST FOR CARAMEL.*

By C. F. HALE.

Pharmaceutical Chemist.

Within the past year the question of standardizing the color of caramel has been engaging the attention of pharmaceutical workers.

Dr. George A. Menge recently suggested the preparation of a standard solution of caramel by boiling on a water bath for five minutes one-half gram of sugar with 5 c.c. of a mixture of sulphuric acid 2 c.c. and water 10 c.c. The resulting mixture, partially cooled by the addition of 25 c.c. cold water, neutralized with potassium hydroxide solution and finally to 100 c.c. forms the standard color with which to compare commercial samples of caramel.

The standard that I have used for several years seems to me to be one that is more readily applicable, as the materials are always on hand and the method is a simple and quick one. The method consists in matching a given sample of caramel against a standard color consisting of a nesslerized solution of ammonia. For carrying out the test, make a stock solution of ammonia oxalate by dissolving .0417 gm. of monohydrated ammonium oxalate, in crystals, in 1 litre of distilled water. Prepare the standard color by taking 10 cc. of this stock solution, adding 38 c.c. of water and 2 c.c. of nessler's solution.

Match the standard color with the caramel prepared as follows: Dissolve 1 gm. of the caramel in water and make up to 1 litre. Run the solution from a burette into a nessler glass until, on dilution with distilled water to 50 c.c., it exactly matches the standard color.

As an arbitrary standard, consider the standard caramel as one of which 0.01 gramme (represented by 10 c.c. of the diluted solution made up to 50 c.c. with water) is required to match 50 c.c. of the color standard. Call this standard caramel 100 per cent. Caramel as found on the market will usually test around this figure.

To obtain the colorimetric value of any other caramel divide 100×10 by the number of c.c. of the diluted caramel solution required. For example, in a particular test, 20 c.c. of the solution of the sample of caramel were required to match the color standard. Then the colorimetric value of the caramel sample equals 100×10

———— = 50 per cent. In other words, this particular sample was one-half strength. This strength is a convenient one for making elixirs.

Among the advantages of this method I might mention that the tints of the ammonia solution and diluted caramel solution are practically identical. The materials for making the test are to be found in every laboratory and the apparatus required consists simply of a burette, pipettes, and two nessler glasses. The two

vials of liquid before you show how similar these solutions are in tint and illustrate the practicability of the method. This method is particularly valuable from the fact that it enables the operator to give a numerical value to any given sample of caramel, a point of importance in making purchases, as well as in the manufacturing laboratory.

APPARATUS AND METHODS FOR THE SAMPLING AND ANALYSIS OF FURNACE GASES.*

By J. C. W. FRAZER AND E. J. HOFFMAN.

The furnace conditions prevailing both in small and in large industrial establishments in this country are frequently far from satisfactory. If such conditions are to be improved, they must be more thoroughly understood, and means must be found to insure complete combustion of the fuel, and yet to permit operation with such an excess of air as will result in the greatest efficiency.

A very important problem is the determination of the small percentage of unburned combustible matter that escapes from the furnace in the flue gases. Under ordinary circumstances so little as .1 per cent of unburned combustible matter in a furnace gas is equivalent to about 1 per cent of the fuel used; and for the determination of such small percentages of gas more accurate and refined methods are required than have ordinarily been available heretofore.

It is the purpose of this paper to describe some apparatus and methods which have proved satisfactory to those engaged in the chemical work of the investigations mentioned, and because the methods are so very accurate, the abstractor thinks it advisable to reprint much of the article as it was published.

CONTINUOUS SAMPLING.

The sampling of flue gases can usually be accomplished satisfactorily by using a perforated iron pipe placed in the flue at the desired point. As the composition of the flue gases at any instant does not vary much at different points in a given cross section, the easiest method is to use a tube with 2 mm. perforations so shaped that it will best distribute the points from which the gas is to be drawn for analysis.

A common method of collecting a sample is to attach a large bottle filled with water to the outside open end of such a sampling tube and then to allow the water to escape at such a rate that the gas, which replaces the water in the bottle, is collected in the desired time.

The following method of sampling was devised to obviate the difficulty of the solubility of certain constituents, as well as certain other difficulties.

COLLECTION AND STORAGE OF THE SAMPLE.

The vessel should have a capacity of 150 to 250 cc. If, when the vessel is in the position shown and is filled

*Read before the annual Convention of the Minnesota State Pharmaceutical Association, Duluth, Minn., July 12, 1911.

*Abridged by B. B. Freud from Bulletin 12, U. S. Bureau of Mines.

with mercury, the stopcocks, and b, are opened the mercury will flow from the lower tube. The gas will then be drawn through the upper tube, enter the vessel at c, and collect above the mercury. So long as the surface of the mercury remains above c the same volume of gas will be collected in each equal interval of the sampling period, so that the sample obtained will be representative. The time required for a certain amount of mercury to run from the vessel can be varied from that taken when both (a) and (b) (see Fig 1), are completely open, to a period of from 8 to 10 hours, or even longer, by attaching at (d) by



Fig. 1.—Gas Sample Collector and Container.

means of a rubber tube a short piece of glass tubing drawn out to a smaller diameter. After the sample of gas has been collected the tube above (a) is filled with mercury from a small funnel whose stem has been drawn out to a capillary. The vessel is then inverted and, by means of a rubber tube attached to a mercury reservoir, the enclosed gas is put under a pressure of about 100 mm. of mercury. When the vessel is returned to its original position, as shown in the illustration, the stopcocks (a) and (b) are mercury sealed, and there is no danger of gas leaking into or out of the vessel.

When it is desired to remove the sample for analysis the vessel is again inverted and the tube above (b) is filled with mercury and attached to the burette. By means of the mercury reservoir previously used to put the gas under pressure, the desired volume of gas is forced out of the vessel into the burette. In case the gas is to be measured over water, the tube above the stopcock (b) is filled with water instead of mercury.

In order to facilitate the safe handling of these vessels it is necessary to mount them in a portable stand (see Fig. 2) and frequently it is desirable to arrange them in batteries of two to four each. The stand con-

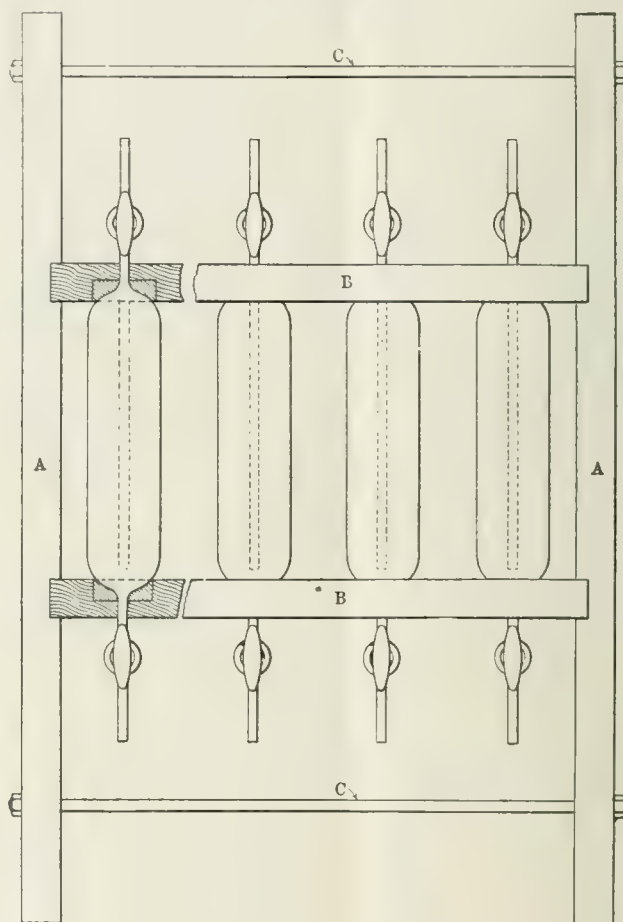


Fig. 2.—Portable Stand for Gas Sample Container.

sists of the two uprights (A), supporting the two shelves (B), and having iron rods (C) extending from side to side through holes bored about 3 cm. from each end of the uprights. The iron rods not only strengthen the stand, but are convenient handles. The shelves are divided longitudinally through the center and holes for the tubes are made along these lines of division. When filled with mercury the vessels should be carefully supported at both ends. Therefore the holes for the tubes are enlarged for a distance equal to about one-half the thickness of the shelf, to receive the tapering ends of the vessels, and the open spaces around these ends are filled with plaster of paris. This is done by filling the cavity in each half of the shelf independently, so that the front half of each shelf may be taken out at any time and a vessel removed without disturbing the remaining ones.

THE WATER-COOLED SAMPLING TUBE.

The portion of the sampling apparatus which is introduced into the furnace may be either a water-cooled metal tube, or better a water-cooled quartz tube.

A type of water-cooled metal tube that has been found satisfactory is illustrated by A, in Fig. 3. The inside tube, through which the gas is collected, is kept cool by cold water, which passes through the surrounding tube and returns through the inside annular compartment. When the inner tube is of quartz, the only difference in construction is the use of asbestos packing to insure a water-tight joint at each end. Connect the end of the tube with a $\frac{1}{4}$ -in. lead pipe, which is brought over to a table situated as near the furnace as possible. By connecting the other end of the lead pipe to an aspirator, a continuous current of gas may

INSTANTANEOUS SAMPLING.

Frequently it is desirable to ascertain the composition of gases at some point in a furnace at a certain definite instant.

It was for the purpose of collecting instantaneous samples that the device shown in Fig. 4 was constructed. The apparatus consists of a quartz sampling tube, A, of 100 cc. capacity, immersed in water contained in the steel tube, B, which is 1.2 meters in length and 10 cm. in diameter. At each end of the vessel terminates in a thick-walled quartz tube, 1-mm. bore, provided with a stop-cock as shown. One of the tubes (a) extends 150 mm. beyond the stop-cock and the open end projects beyond the end of B. An enlargement (b) 60 mm. from the stop-cock, gives a firmer

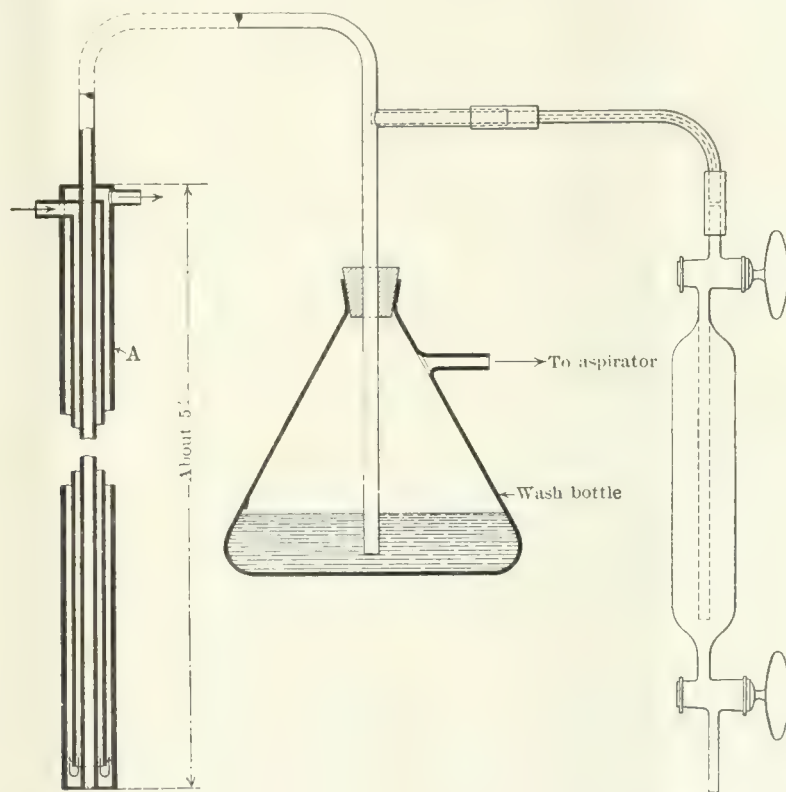


Fig. 3.—Arrangement of Apparatus for Sampling Furnace Cases.

be drawn from the furnace. At the table the leadpipe is perforated at a convenient point and a glass tube of small bore is inserted. This tube is connected directly with the mercury-filled sample receiver, and the sample is taken from the current of gas flowing through the lead tube.

It is advisable to use a tube with an inside diameter of about three-sixteenths of an inch. In order to be sure that the current of gas is flowing properly through the tubes, it is necessary to introduce a trap at some point beyond that at which the sample is taken from the lead pipe. The trap is simply a wash bottle containing water through which the gas bubbles on its way to the aspirator, the rate of bubbling roughly indicating conditions in the tubes. Fig. 3 illustrates the entire sampling system.

hold for the cement of litharge and glycerin with which the cavity in the cellar (c) is filled; in this way (c) is fastened permanently to (a). The device for opening and closing the sampling tube from the outside is made of brass and is supported by the two end pieces E of the steel tube (B). It consists of the brass frame C, in which is supported the mechanism for turning the stop-cock. This consists of the brass shaft (h) on which is set the wheel (i) and, beneath the frame, the brass plate (l) carrying four projections (g), which fit around the handle of the stop-cock as shown. In order to avoid straining the stop-cock in turning which might occur if (H) were not centered above the stop-cock, the pieces (g) are small rollers. The face of the wheel (i) is threaded to engage with the threaded end of the brass rod (r). The piece (k)

serves as a guide for the brass rod and affords a means of adjusting the threaded end of the rod to the face of (i). The adjustment is accomplished by having the hole in (k) through which the rod (r) passes eccentric to the bearing of (k) in C. A movable stop (m) can be set to limit the rotation of (i) and the extent to

bolts which bind it to B. The quartz tube is placed in position and the nut tightened. During this operation (c) is prevented from rotating by two small dowel pins which enter holes provided for them in (e). By trial (m) is adjusted so that when the rod (r) reaches it the handle of the stop-cock is rotated 90°. It is

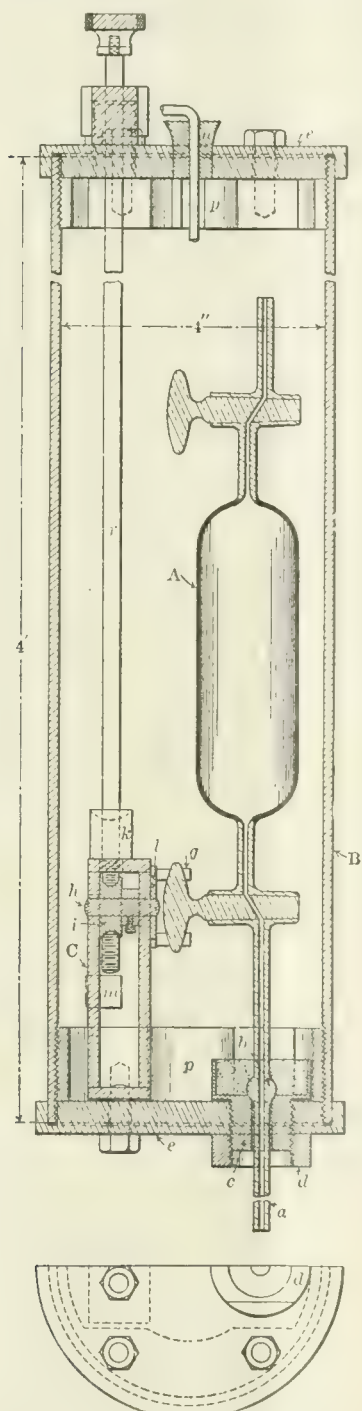


Fig. 4.—Apparatus for Taking Instantaneous Gas Samples.

which the stop-cock may be turned. The steel tube B is threaded internally at each end to receive the threaded steel pieces (p), which are tapped to receive bolts which secure the end pieces (e).

OPERATION OF THE APPARATUS.

Before collecting a sample, the end piece (e) which is to carry the vessel A, is removed by unscrewing the

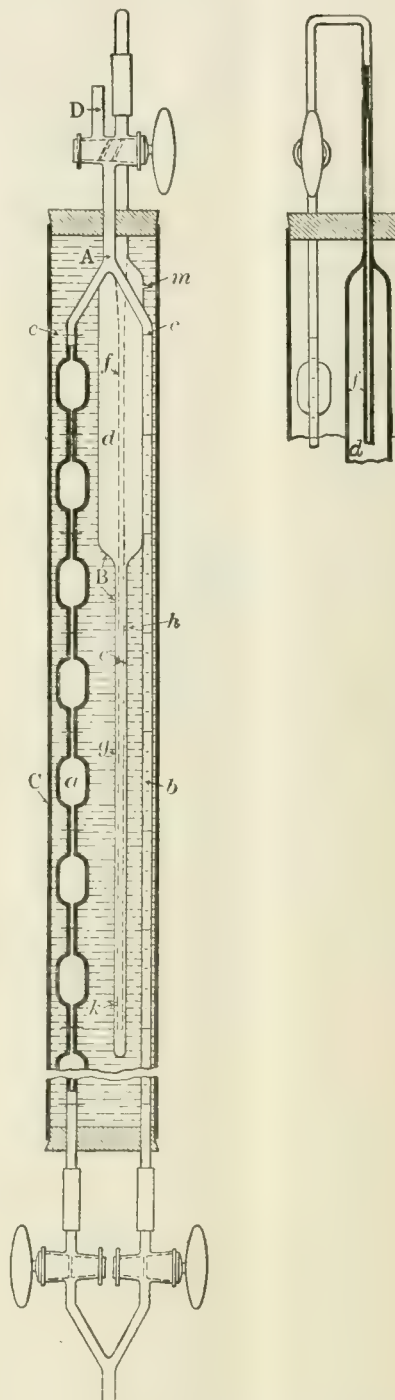


Fig. 5.—Apparatus for Exact Gas Analysis.

customary to adjust the stopcock so that communication with A is established when (r) is against (m) as shown. The withdrawal of (r) then closes the stopcock. Having the stopcock properly adjusted, the end piece (e) is bolted to B, and the latter is suspended in a vertical position by a handle clamped on the other end, and not shown in the illustration. While in a

brass bushing at the upper end and finds its way without difficulty into the hole in (k). The steel tube B is filled with water through (n), which is then closed by a perforated rubber stopper, through which passes a short glass tube bent at a right angle and having its projecting end directed upward when the apparatus is in use. This tube relieves the internal pressure when the temperature of the water rises.

The apparatus is then taken to a vacuum pump and placed in a horizontal position in two semi-circular rests which prevent its moving while it is connected with the pump. When the tube A is exhausted, and while it is still connected with the pump, (r) is withdrawn so far that the threaded end leaves (i) but remains in (k). This closes the stop-cock. The time at which (r) and (i) become disengaged can be readily determined by the increased ease with which (r) moves. The pump is disconnected, and as soon as convenient the apparatus is introduced into the furnace, the open end of the tube (a) being placed at the point from which the sample is desired. The stop-cock is opened by pushing in the rod (r) until it is in contact with (m) and after the short time required for (a) to fill the rod is withdrawn and the stop-cock thus closed. The sampler should remain in the furnace only about 30 seconds.

After removing the apparatus from the furnace, the water is drained from B, the rod is withdrawn, the nut (d) is started to insure its subsequent easy removal, and the end piece (e), carrying (a), is removed. Then (d) is taken entirely off, and after the two glass tubes have been filled up to the stop-cocks with mercury the sample in (a) is transferred to one of the holders illustrated in Fig. 1. Since it requires about 30 minutes for the complete operation of collecting a sample in this way, a series of samples is generally collected and stored in the holders before making the analyses.

With the apparatus illustrated in Fig. 4 the amount of water vapor accompanying a gas sample can be easily determined by absorbing the moisture and weighing it. Likewise, the presence of traces of nitric oxide in furnace gases has been shown by using a simple modification of the method just described.

ANALYSIS OF THE SAMPLE.

The sample having been collected, its analysis is performed most conveniently by the method of Hempel. While the ordinary Hempel burette is sufficiently accurate for most purposes it does not enable the observer either to detect or to determine changes in volume amounting to 0.1 cc. or less, since the error in reading the burette itself cannot be less than this volume.

It was to be able to measure all possible changes in volume that the apparatus described below was designed by the writers. Since this apparatus was devised entirely independently and used for some time before their attention was called to the work of Bleier and of White, and since it is believed to possess cer-

tain advantages over other similar forms of apparatus, its description is given here, but without making any claim of priority as to its essential principles.

The apparatus, illustrated in Fig. 5, consists of the burette A and the automatic compensating device B. The measuring portion of the burette A and the whole of the compensator B are inclosed in the water jacket C. The measuring portion of the burette consists of the two limbs (a) and (b) the graduated portions of which are 66 cm. long and united at the top in an inverted Y-shaped connection to which a Greiner-Friedrich two-way stop-cock is attached. Through this stop-cock communication can be made with either of two short, thick-walled tubes of small bore, one of which is connected with the compensating device. Outside the water jacket C, at the lower end, is a second Y tube, each of whose limbs is provided with a stop-cock and attached to rubber connections to the projecting ends of the limbs (a) and (b) as shown. To the lower end of the Y tube is attached heavy rubber tubing connected with a mercury reservoir not shown in the illustration.

The tube (a) consists of a series of 10 bulbs, each having a capacity of 10 cc. between the two graduation marks immediately above and below it. The straight glass tube (b) has an internal diameter of about 4.5 mm. and its graduated part has a total capacity of 10.1 cc. The beginning of the graduated portion of each limb of the burette is at (c). The compensator B, while utilizing the principle of Pettersson of counterbalancing the pressure of the gas to be measured with that of a constant mass of air occupying a constant volume, is arranged in a somewhat different form from Pettersson's device. The confined air, whose pressure at a constant volume is equalized by that of the gas to be measured is contained in the bulb (d) and above the mercury surface (e) in the tube (g) which forms the lower termination of the bulb.

The glass tube (f) is connected at its upper end with one of the communications through the stop-cock of the burette and is sealed into the top of the bulb (d). Its other end extends nearly to the bottom of (g) and opens beneath the surface of the mercury. It is centered in (g) at two points—h, which is 25 mm. above the surface of the mercury, and k, about 50 mm. from its lower end—by three small glass projections sealed on the outside of it at each of these points. The diameters of the tubes (f) and (g) are proportioned so that the distance from the inside of (g) and the outside of (f) is as nearly as possible equal to the internal diameter of (f). The distance from the surface of the mercury in (f) to the stop-cock of the burette should be as short as possible, since the gas in this portion of the tube must be drawn into the burette by decreasing the pressure. The depth of mercury in the compensator should be sufficient to require about one-third of an atmosphere excess pressure in the burette to force gas from the latter into the compensator, but the quantity of mercury

should not be so great that its expansion or contraction will cause an appreciable change in the volume of the enclosed air. The tube (f) should extend nearly to the bottom of the compensator in order to displace as much mercury as possible.

The compensator may be filled so that the readings on the burette are the correct volumes of the gas at 0° C. and 760 mm. pressure, or it may be closed under known conditions of temperature and pressure and the readings corrected to standard conditions. The latter method is sufficient for most purposes, and when it is used the compensator may be closed by replacing the seal at (m) by a small tube and stop-cock.

OPERATION OF APPARATUS.

To obtain a reading of the volume of a gas in the burette, draw nearly all the gas into the limb (a) and adjusting the pressure to approximately that of the atmosphere, the number of bulbs the gas will fill completely when at the pressure of the air in the compensator may be ascertained. Having determined this, the mercury in the burette is brought exactly to the level of the graduation beneath the last bulb completely filled in the trial experiment and the stop-cock at the bottom of the limb is closed. The remaining fraction of a bulb full of gas is then made to enter the limb (b) and the two-way cock is turned so to place the burette in communication with the compensator. The pressure of the gas in the burette is then adjusted, by means of the mercury reservoir, until the two surfaces of mercury in the compensator are on the same level. The pressure of gas in the burette is then equal to that of the gas in the compensator. The stop-cock at the bottom of (b) is then closed and the reading of the burette taken. To the reading is added the pre-determined capacity, K, of that part of the apparatus between the graduated portion of each limb of the burette and the mercury meniscus in the tube (f).

The gases are measured in the moist condition. With this apparatus a complete analysis of gas may be made in the usual way by connecting the burette in turn with various absorption pipettes. It is, however, more convenient and accurate to connect the burette permanently with the desired pipettes. By this arrangement errors due to the repeated making and breaking of connections are avoided. The entire system may be conveniently mounted on the same wooden support, and for the analysis made thus far such an arrangement has been employed.

A series of three pipettes is connected with the tube D by means of fine capillary tubing in the manner generally employed with the Orsat apparatus. These pipettes consist of a combustion pipette provided with an electrically heated platinum coil of the type recommended by L. M. Dennis, an absorption pipette containing a solution of potassium hydroxide for the absorption of carbon dioxide, and a pipette containing phosphorus for the determination of oxygen. In order to avoid error due to the capacity of the connecting tubes, these are made as short as possible, and their

internal diameter is not greater than 0.4 mm.

The limb (b) of the burette on which the readings are made is graduated in hundredths of 1 cc. and the tenths of these divisions can be estimated quite accurately. Experiments have shown that 50 to 100 cc. of gas can be measured accurately to 0.01 cc.

The above description of the manipulation of the apparatus involves the use of a leveling bottle attached to the lower Y tube. With care the pressures in the burette and the compensator can be equalized accurately by the use of the leveling bottle alone, but the equalizing can be accomplished more rapidly and with greater precision by making the final adjustments by means of a screw plunger fitting tightly in its bearings in a closed steel reservoir filled with mercury. The steel reservoir has two openings, one connected with the Y tube by means of a short piece of rubber tubing, and the other connected with the leveling bottle through a longer rubber tube. A glass stop-cock is placed between the reservoir and the leveling bottle, by means of which communication between the two is cut off previous to the final adjustment of pressure by means of the screw plunger. The steel reservoir also serves as a trap for much of the dirt which becomes loosened from the inner walls of the connecting rubber tubes, and would otherwise find its way into the burette.

The production of copper in Montana decreased from 312,058,011 lbs. in 1909 to 284,808,553 lbs. in 1910, a loss of 22,278,460 lbs. The Summit Valley or Butte district contributed all but 581,648 lbs. of the total.

There were 5,079,446 short tons of ore from Montana sold or treated in 1910, a decrease of 341,739 tons, or about 6 per cent. The production of copper ores, which amounted to nearly 90 per cent of the total tonnage treated, declined 448,684 tons. Only 266,689 tons were treated in gold and silver mills, and 616,596 tons were shipped crude to smelters, but 4,170,114 tons, or 82 per cent of all ore treated, were sent to concentrating mills.

Hardy S. Ferguson, Consulting Engineer, 200 Fifth Ave., New York, estimates the amount of wood required to make one ton of newspaper as follows:

Assumptions.—(1.) One cord of wood will yield 1 ton dry ground wood pulp. (2.) Two cords of wood will yield 1 ton dry sulphite pulp. (3.) In the paper mill 22 per cent of the sulphite is wasted. (4.) In the paper mill 8 per cent of ground wood is wasted. (a.) Paper containing 25 per cent sulphite. One ton paper requires — $25/98 \times 2 + 75/92 \times 1 = 1.32$ cords. (b.) Paper containing 22½ per cent sulphite. One ton paper requires — $22.5/98 \times 2 + 77.5/98 \times 1 = 1.30$ cords. (c.) Paper containing 20 per cent sulphite. One ton paper requires — $20/98 \times 2 + 80/98 \times 1 = 1.28$ cords.

The CHEMICAL ENGINEER

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NOTES AND COMMENTS.

The Use of Mine Rescue Breathing Apparatus in Chemical Plants.

A recent circular issued by the Bureau of Mines describes the formation of rescue parties, and the use of mine rescue breathing apparatus, in case of mine accidents.

So many of the instructions given in this circular for the work of rescue in mines would be applicable for the same purpose around many of our chemical plants. The quotations which are herewith given from certain portions of the circular show its applicability to chemical industries, and it is recommended that the superintendents in charge of chemical plants secure the circular—(Miner's Circular No. 4)—and familiarize themselves with its contents.

There are many times when occasions arise in a chemical plant due to a break in a pipe, a leak in a furnace, or furnace-wall, or some similar accident, when noxious vapors escape into the air endangering the lives of the men who have to make the necessary

repairs, or compelling a shut-down of this particular portion of the plant until the repairs are made.

It is the writer's opinion that in such instances a mine rescue breathing apparatus would be just the thing to have on hand. In addition to having the apparatus on hand, there should be a crew trained in the use of the apparatus just the same as the Bureau of Mines recommends for work at the mines.

We quote from the descriptive matter contained in this circular, and leave it to the reader to see how application of this can be made to work around the chemical plant.

The circular referred to is "The Use and Care of Mine-Rescue Breathing Apparatus," by James W. Paul, who has general charge of the rescue work. Mr. Paul makes the statement that the use of such apparatus for rescue work in mines is no longer an experiment, but has become an important factor in lessening loss of life and property from fires and explosions.

The circular describes the various types of apparatus used by the bureau and gives careful instruction as to the care this apparatus should receive when not in use. Then the author passes upon the qualifications of rescue men at the mines. Mr. Paul says: Men twenty-two to forty-five years old, in good physical condition who are temperate in their habits and naturally calm and deliberate are best suited for mine rescue work. Before a man undergoes training in the use of breathing apparatus he should be examined by a physician to ascertain his physical condition, especially the action of his heart and lungs and any defects of the nose or throat. Unless a man has a physician's certificate stating that his physical condition is good, he should not be permitted to take rescue training nor to attempt rescue work in a mine.

A rescue party should have not less than five, and better not less than six members. Only such persons should be allowed to join the party as have already been trained in the use of the apparatus and are equipped with rescue apparatus in good order and have agreed to follow the directions of the leader, who must have full charge. While working in unbreathable gases within a mine the men should keep close to one another and not separate under any condition.

To be efficient and successful a party must take every precaution for its own safety. If one person in a party faints or receives an injury he becomes a burden instead of a help, for the entire party must at once conduct him to the surface or to fresh air. One or two stretchers should always be at hand.

A relief station or base of operations should be established at the end of the good air and a relief crew with knapsacks should be stationed there, ready to put on their apparatus and start at a moment's notice. A patrol of all brattice and doors leading up to the relief station should be maintained to protect the relief crew from harm.

At each large mine there should be at least four crews, two outside and two inside crews, each of six men, including a captain and a lieutenant, and these crews should have practice once a week.

While working in dense smoke the members of a crew should hold a rope which leads to fresh air.

In case of total failure of an apparatus to supply unbreathable air, the wearer of the apparatus can throw away all parts but the oxygen cylinder, and breathe from the cylinder

through his mouth while endeavoring to reach fresh air with the rest of the crew.

Apparatus for giving oxygen to one who has been overcome with gases is an essential part of the equipment of a rescue party.

A telephone helmet is a convenience for shaft work, and its presence lends much confidence to a rescue party. Electric lamps, safety lamps, gas-analysis apparatus, thermometers, a pocket compass, and a map of the mine are necessary parts of the equipment.

At each training station a record should be kept showing the work done by the men and the difficulties encountered. A record of each apparatus should be kept also. If an apparatus fails to give proper service it should be subjected to the regular tests unless some injury is seen by inspection.

The United States Bureau of Mines has established a regular course of training in the use of mine-rescue breathing apparatus. This training is designed to give miners or other persons connected with mining a knowledge of breathing apparatus in general, and a confident familiarity with those types of apparatus which are most apt to be used in this country.

The purpose of the Bureau of Mines in establishing this system of training is to facilitate investigative work within mines in which disasters have occurred, and to make mine owners and miners acquainted with the value of breathing apparatus for rescue operations after mine disasters. It is hoped that, as a result of this work by the Federal Government, in the near future men familiar with such apparatus will be scattered throughout the coal-mining centers of the country, and be available on short notice to assist in rescue operations.

After a disaster, valuable time is often lost in training men at the mine before rescue parties can be organized. Furthermore, a man cannot work efficiently unless he has thorough confidence in the apparatus. To give a man this confidence, the course of training has been planned in such a way that he must do work in poisonous or unbreathable gases for periods of one and two hours at a time.

The British Indian Government six months ago employed a European expert paper manufacturer to come out to India for the purpose of conducting a series of investigations into India paper-making material at the Imperial Forest Research Institute at Dehra. His work has been so satisfactory and is considered of so much importance that his services have been retained for a further period of six months. The results of his investigations have not been made public, but probably will be about the first of next year.

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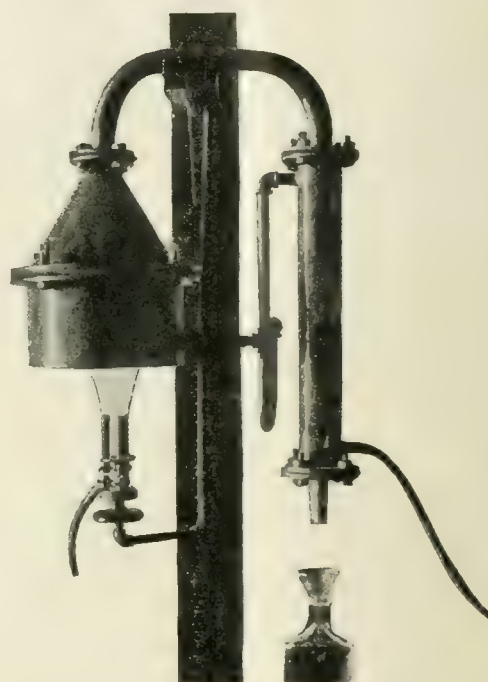
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No. 6.

A PLEA FOR REVISION OF THE RULES OF THE AMERICAN CHEMICAL SOCIETY GOVERNING THE PROXIMATE ANALYSIS OF COAL.*

By MARIUS R. CAMPBELL.

The recent publication by the West Virginia Geological Survey of all analyses of coals of that state that have been made in recent years affords an opportunity for the comparison of results obtained by different chemists in different laboratories. Such a comparison shows clearly that, although these laboratories may be well equipped and the chemists follow closely the rules laid down by the American Chemical Society for making proximate analyses, the results are very discordant and can not be relied upon for determining the comparative values of coals or for their classification. Heretofore the demands have not been such as to require great accuracy in coal analysis, but with the constantly and rapidly growing practice of buying coal on specifications and the need for a scientific classification of at least the high grade coals, accuracy and uniformity of results become of the utmost importance. The object of this paper is to call attention to these discordant results and to urge that the American Chemical Society revise its regulations, introducing such refinements that it will be possible for any chemist to duplicate, within a reasonable limit of error, the work of any other chemist, even though working in a different laboratory.

The West Virginia report in question contains not only all analyses made by the state, corrected and brought up to date, but also 140 analyses of Pocahontas and New River coals made at the Pittsburg laboratory of the United States Geological Survey. The former are analyses of individual samples, but the latter were made of composite samples and really represent the average of about 500 analyses.

In comparing these analyses it must be remembered that they were not made on check samples, but merely on samples obtained in the same mine but by different collectors and in different years. Some may urge that uniformity of results is not to be expected from samples collected in such a way, but in the present case

the striking feature is not the irregularity of the results, but the almost phenomenal regularity of the fixed carbon of the West Virginia analyses below those made by the United States Geological Survey.

Differences of this kind in such a large series of analyses effectually rule out the personal equation of the sampler, difference in method of sampling and differences resulting from samples being taken in different parts of the mine or coal bed, for errors of this kind would be variable—sometimes one way and sometimes another.

The discrepancies in the results, therefore, must be due to differences in laboratory equipment and methods of work. Recently it was found that the results obtained in the Pittsburg and Washington laboratories of the United States Geological Survey were not in accord, and close investigation showed that the difference was due to the fuels used in these laboratories, Pittsburg using natural gas and Washington using artificial gas. In the present case fuel does not seem to enter into the question, as natural gas was used in both laboratories. The writer has been assured by Dr. I. C. White, state geologist of West Virginia, that his chemist followed exactly the rules laid down by the American Chemical Society, and the same rules have been observed by the chemists of the United States Geological Survey. Therefore the discrepancies in the results are unexplainable at the present time, but doubtless they will be found to be due to some slight variation in practice or equipment, which apparently does not conflict with the regulations but which leads to very different results.

Table I shows the results obtained in the two laboratories on samples from 57 mines in the Pocahontas and New River districts, grouped according to coal beds. In order to facilitate comparisons the writer has recalculated all of the analyses to the moisture-and-ash-free basis, thus showing the actual composition of the coal substance as well as a proximate analysis can show it. Also, he has given the fuel ratio (fixed carbon divided by the volatile matter), as most American classifications of the high grade coals are based on this ratio.

From this table it will be seen that in every case, except four, the fixed carbon in the Pittsburg analysis exceeds that of the Morgantown analysis, and consequently the fuel ratios of the former exceed those of

*Published by permission of Director of the U. S. Geological Survey.

¹Levels and coal analyses: Bull. W. Va. Geol. Survey No. 2, 1910.

the latter. The average excess of fixed carbon for 29 analyses of Pocahontas No. 3 coal is 3.0 per cent, and the average excess of fuel ratios for the same analyses is 1.1. The same average for the four analyses of

TABLE I.—COMPARATIVE RESULTS OF ANALYSES MADE BY THE UNITED STATES GEOLOGICAL SURVEY AND THE WEST VIRGINIA GEOLOGICAL SURVEY.

Name of Mine	Coal	Bed.	U. S. Geo- logical Sur- vey, Pitts- burgh Lab oratory.				W. Va. Geo- logical Survey.				Excess U. S. G. S. over W. Va. G. S.			
			Fixed Carbon	Volatile Matter	F. C.	V. M.	Fixed Carbon	Volatile Matter	F. C.	V. M.	Fixed Carbon	Volatile Matter	F. C.	V. M.
Jed	Pocahontas	No. 3	87.4	12.6	6.9	84.9	15.1	5.6	2.5	1.3				
Empire ..	"	"	86.0	11.0	6.1	82.2	17.8	4.6	3.8	1.5				
Hiawatha ..	"	"	85.8	14.2	6.0	83.0	17.0	4.9	2.8	1.1				
Lick B'nch	"	"	85.8	14.2	6.0	81.8	18.2	4.5	4.0	1.5				
Upland ..	"	"	85.7	14.3	6.0	81.0	19.0	4.3	4.7	1.7				
Wenonah ..	"	"	85.7	14.3	6.0	84.0	16.0	5.2	1.7	.8				
Bottom Ck	"	"	85.7	14.3	6.0	82.0	18.0	4.6	3.7	1.4				
Shawnee ..	"	"	85.6	14.4	5.9	81.7	18.3	4.5	3.9	1.4				
Indian R'ge	"	"	85.6	14.4	5.9	81.5	18.5	4.4	4.1	1.5				
Keystone ..	"	"	85.6	14.4	5.9	82.3	17.7	4.6	3.3	1.3				
Elk Ridge.	"	"	85.6	14.4	5.9	83.5	16.5	5.1	2.1	.8				
Buckeye ..	"	"	85.4	14.6	5.9	81.8	18.2	4.5	3.6	1.4				
Roanoke ..	"	"	85.3	14.7	5.8	81.5	18.5	4.4	3.8	1.4				
Pulaski ...	"	"	85.2	14.8	5.8	82.6	17.4	4.8	2.6	1.0				
Arlington ..	"	"	85.2	14.8	5.8	82.0	18.0	4.6	3.2	1.2				
Peerless ..	"	"	85.2	14.8	5.8	82.4	17.6	4.7	2.8	1.1				
Shamokin ..	"	"	84.9	15.1	5.6	82.2	17.8	4.6	2.7	1.0				
Louisville ..	"	"	84.8	15.2	5.5	80.2	19.8	4.0	4.6	1.5				
Lynchburg.	"	"	84.7	15.3	5.5	83.0	17.0	4.9	1.7	.6				
Powhatan ..	"	"	84.7	15.3	5.5	83.4	16.6	5.0	1.3	.5				
Ashland ..	"	"	84.6	15.4	5.5	80.7	19.3	4.2	3.9	1.3				
Eureka ...	"	"	84.6	15.4	5.5	82.1	17.9	4.6	2.5	.9				
Turkey Gap	"	"	84.4	15.6	5.4	82.9	17.1	4.8	1.5	.6				
W. P'hont's	"	"	84.4	15.6	5.4	79.6	20.4	3.9	4.8	1.5				
Angle	"	"	84.3	15.7	5.4	80.1	19.9	4.0	4.2	1.4				
Coaldale ..	"	"	84.2	15.8	5.3	80.7	19.3	4.2	3.5	1.1				
Delta	"	"	84.2	15.8	5.3	82.9	17.1	4.8	1.3	.5				
McDowell ..	"	"	83.7	16.3	5.1	82.1	17.9	4.6	1.6	.5				
Goodwill ..	"	"	83.2	16.8	4.9	82.2	17.8	4.6	1.0	.3				
North Side	"	No. 4	85.7	14.3	6.0	83.0	17.0	4.9	2.7	1.1				
Slab Fork..	Beckley	"	85.7	14.3	6.0	82.9	17.1	4.8	2.8	1.2				
Raleigh No.3	"	"	84.2	15.8	5.3	83.3	16.7	5.0	.9	.3				
Piney No. 4	"	"	83.5	16.5	5.1	82.6	17.4	4.8	.9	.3				
Piney No. 2	"	"	81.6	18.4	4.5	80.6	19.4	4.2	1.0	.3				
Laurel Cr'k	Quinnimont	"	83.2	16.8	5.0	81.0	19.0	4.3	2.2	.7				
Beechwood ..	"	"	81.1	18.9	4.3	78.8	21.2	3.7	2.3	.6				
J.B.&B.No.2	Sewell	"	83.4	16.6	5.0	80.2	19.8	4.1	3.2	.9				
Tamroy ..	"	"	82.5	17.5	4.7	80.4	19.6	4.1	2.1	.6				
Lanark No.3	"	"	82.2	17.8	4.6	79.0	21.0	3.8	3.2	.8				
Sprague ..	"	"	82.1	17.9	4.6	80.9	19.1	4.2	1.2	.4				
Raleigh No.5	"	"	81.7	18.3	4.5	81.3	18.7	4.3	.4	.2				
Prosperity ..	"	"	81.5	18.5	4.4	81.5	18.5	4.4	0.0	0.0				
Derryhale ..	"	"	81.4	18.6	4.4	76.6	23.4	3.3	4.8	1.1				
Price Hill ..	"	"	80.7	19.3	4.2	78.9	21.1	3.7	1.8	.5				
Oswald ...	"	"	80.4	19.6	4.1	79.4	20.6	3.8	1.0	.3				
Wingrove ..	"	"	80.4	19.6	4.1	81.1	18.9	4.3	—	.2				
Sugar Cr'k ..	"	"	80.3	19.7	4.1	79.2	20.8	3.8	1.1	.3				
Sun	"	"	80.3	19.7	4.1	78.7	21.3	3.7	1.6	.4				
Star	"	"	79.9	20.1	4.0	78.0	22.0	3.5	1.9	.5				
Sherwood ..	"	"	79.9	20.1	4.0	80.4	19.6	4.1	—	.5				
Turkey K'b	"	"	79.6	20.4	3.9	77.8	22.2	3.5	1.8	.4				
Prudence ..	"	"	79.1	20.9	3.8	77.7	22.3	3.5	1.4	.3				
Harvey ..	"	"	78.4	21.6	3.6	76.4	23.6	3.2	2.0	.4				
K'neys C'k ..	"	"	77.5	22.5	3.4	73.7	26.3	2.8	3.8	.6				
Parra ..	"	"	77.5	22.5	3.4	76.2	23.8	3.2	1.3	.2				
Stuart	"	"	77.1	22.9	3.4	77.7	22.3	3.5	—	.1				
Smokeless ..	"	"	76.1	23.9	3.3	72.3	27.7	2.6	3.8	.6				
Av. for Pocahontas No. 3			85.1	14.9	5.7	82.1	17.9	4.6	3.0	1.1				
Av. for Beckley			83.8	16.2	5.2	82.4	17.6	4.7	1.4	.5				
Av. for Quinnimont			82.2	17.8	4.6	79.9	20.1	4.0	2.3	.6				
Av. for Sewell			80.1	19.9	4.1	78.4	21.6	3.7	1.7	.4				

Beckley coal are 1.4 per cent and .5, and for Quinnimont (Fire Creek) coal 2.3 per cent and .6. In the case of the Sewell coal the work of the two labora-

tories is more accordant, but even here in 17 analyses the fixed carbon of the Pittsburgh analyses is higher than in those made at Morgantown; three show the excess on the opposite side, and in one the same results were obtained in both laboratories. The average excess of fixed carbon for the whole number of analyses of this coal is 1.7 per cent and the average excess for the fuel ratio is .3. The average excess for the entire list of 57 analyses is 2.6 per cent for the fixed carbon and .7 for the fuel ratio.

Similar comparisons may be made of analyses from the various laboratories of the United States Geological Survey and the Morgantown laboratory of the West Virginia Geological Survey as shown by Table II.

TABLE II.—COMPARATIVE RESULTS OF ANALYSES MADE IN VARIOUS LABORATORIES.

Name of Mine	1. U. S. G. 2. U. S. G. 3. W. Va. G. 4. U. S. G.				S. Pitts- S. James- S. Morgan- S. St. Louis			
	Fixed Carbon	Volatile Matter	F. C.	V. M.	burgh Lab.	town Lab.	town Lab.	Lab.
Rush Run...	79.5	20.5	3.9	79.6	20.4	3.9		
Hemlock H'w.	83.7	16.3	5.1	81.3	18.7	4.3		
Piney No. 1.	82.3	17.7	4.6	81.7	18.3	4.5		
Raleigh No. 1.	84.5	15.5	5.5	82.8	17.2	4.8		
Sun	80.3	19.7	4.1				78.7	21.3
McDonald							77.8	22.2
Brooklyn							76.7	23.3
Star	79.9	20.1	4.0	79.2	20.8	3.8	78.0	22.0
Derryhale	81.4	18.6	4.4	80.1	19.9	4.0	76.6	23.4
Smokeless	76.1	23.9	3.2	74.5	25.5	2.9	72.3	27.7
Zenith	84.2	15.8	5.3					
Turkey Gap...	84.4	15.6	5.4	83.0	17.0	4.9	82.9	17.1
Lick Branch...	85.8	14.2	6.0	82.7	17.3	4.8	81.8	18.2
Average excess of analyses 1 above analy- ses 2.....								
Average excess of analyses 2 above analy- ses 3.....								
Average excess of analyses 3 above analy- ses 4.....								

In this instance the number of cases is less, but the figures given in the table, with the exception of those from West Virginia, are the averages of from three to ten analyses. Inspection of the table shows that the highest fixed carbon was obtained in the Pittsburgh laboratory, the second highest in the Jamestown laboratory, the third highest in the Morgantown laboratory, and the lowest at St. Louis. The difference between the average for the various columns 1, 2, 3 and 4 is remarkably regular, ranging, as given in the table, from 1.2 to 1.5, but in making this comparison only those coals common to the columns being considered have been taken into account. When, however, the other analyses are compared a similar difference will be found to exist. Thus, in the Rush Run analyses the difference between column 2 and column 4 is 3.5, whereas the average difference between these columns is 2.7; in the Sun analyses the actual difference between columns 1 and 3 is 1.6 and the average is 2.9; and in the Zenith analyses the actual difference is 4.1 and the average 3.8.

The figures given in Tables I and II, while not thoroughly conclusive, seem to indicate clearly that each laboratory has a different equipment, or that each laboratory staff interprets the rules of the American Chemical Society in a slightly different manner from that of another laboratory. The conclusion seems to be clear that the American Chemical Society should frame such a set of rules and regulations that differences like the above would be impossible and the work of any high grade laboratory could be compared directly and confidently with that of any similar laboratory in the country.

EFFLORESCENCE ON BRICK WORK.*

By A. F. GREAVES-WALKER.

Wherever brick are used for building purposes, brick walls will be noted which are covered or partially covered with a film of scum. This scum is generally white, but may be yellow or green.

It appears at its worst on red brick walls, but may be seen more or less plainly on brick of any color.

When the walls are new it appears as white blotches at various places. Later it may come out all over the building, giving the brick work a faded or dirty appearance. In a few years a red brick building so affected will take on a faded pink appearance which will stay with it forever.

The scumming is always worst near the eaves and downspouts, and under the window-sills and water table. This is caused by leakage of rainwater between the roof and trough and from the downspouts, and through the catching of the water by the sills and water table which consequently soaks into the brick beneath.

The scum is simply a film of soluble salts carried from the interior of the brick to the surface by the water which they absorb, and left there after it evaporates.

This scum is generally composed of sulphates, although it may be composed of any other soluble salt. Chlorides are found in seacoast clays and alkalies in clays of the western arid region.

In the clays of this country, the scum causing salts, in order of their abundance are:

CaSO_4	calcium sulphate.
MgSO_4	magnesium sulphate.
K_2SO_4	potassium sulphate.
Na_2SO_4	sodium sulphate.
FeSO_4	ferrous sulphate.
$\text{Al}_2(\text{SO}_4)_3$	aluminum sulphate.

Often all of these occur in a single clay, especially if it has been weathered and subjected to the attack of decomposing pyritiferous minerals.

All of the above salts are very soluble except calcium sulphate, which is soluble 1 : 400. But this is great enough to completely ruin face brick in which it occurs, as the surface is small in comparison with the weight. This particular salt, in fact, is probably re-

sponsible for half the ruined walls in this country.

The sulphates of soda, potash and magnesia will often cause scum when present in the most minute quantities. As little as 0.01 per cent of any one of these salts has been known to ruin face brick.

There are four ways in which salts are incorporated into the brick:

- (1) Soluble salts in the clays or other materials used.
- (2) Soluble salts in the water used for tempering the clay.
- (3) Soluble salts formed in kilns by oxidation of minerals in clay, or by reaction of sulphurous kiln gases on clay.
- (4) Soluble salts introduced into brick after burning, principally from mortar.

A great majority of clays contain soluble salts, some large quantities, others but a trace. Surface clays are more likely to have a high content than the older deposits. Trouble is encountered in all parts of the world from this source.

Probably 75 per cent of the scumming comes from this cause.

Often materials are mixed with clays that carry soluble salts into them. Examples of this are the brick made around London, Eng., in which ashes are used to reduce shrinkage, and brick made along the Hudson in which coal slack is used to aid in burning. In both of these cases the soluble salts in coal add to, if they do not entirely produce, the scum.

In clays that vitrify scumming can be overcome by burning them to a vitrified body, in which case the salts form silicates and become harmless.

The water used for mixing and tempering mud brick often contains soluble salts. This, of course, produces the same effect as if the clays contained them. Many good clays are injured in this way.

The formation of soluble sulphates in the kiln is a very common trouble. It is often encountered with clays that contain practically no troublesome ingredients before being fired.

In many cases there is really no excuse for trouble at this stage. It is simply a case of ignorance on the part of the brickmaker or poorly designed kilns and stacks.

When a kiln is set and the heating up process started, the ware contains considerable moisture. If good judgment is used and the fires are kept at such a stage that only so much of the moisture is being driven off as the sluggish draft of a cool stack will carry off, trouble is not likely to occur; but if the moisture is driven off more rapidly than the stack can handle it, it is precipitated on the ware again, when it absorbs sulphur from the gases and forms sulphuric acid. This acid attacks the carbonates and other less soluble salts and forms soluble sulphates.

There is no doubt that considerable trouble is caused by the mortar in which the brick are laid. Poorly mixed mortar is dissolved by the rain water and the

*From the Journal of Industrial and Engineering Chemistry.

salts carried into the brick, only to be brought to the surface later. It is therefore always important to take great care with the mortar.

Methods of Overcoming Scum.—Many methods are employed to overcome or prevent scum. It is possible to do this in nearly every case, but is often impossible on account of the low selling price of the ware. Where facing brick or other wares are made by the wet process or the ware is vitrified, there is practically no excuse for trouble.

The preliminary treatment is the principal method of overcoming or preventing scum. This is done by treating the clay or treating the water used.

Treating the Clay.—Clays that contain soluble salts can be washed before using and the salts leached out. This may be done by treating piles of clay, after mining, with water and then allowing them to dry. This method is only possible where high-priced ware is made. Clays are often allowed to stand in piles for from six months to a year, in which case the rain and snow do the same thing.

In many clays the above methods will only cause the trouble. This would be the case where a clay did not contain soluble salts originally, but upon exposure to the weather insoluble salts were converted into soluble salts. This would be the case with pyritiferous clays.

Chemical treatment is very often resorted to. This is accomplished in clays made up by the wet process, by adding, to the water used, such chemicals as will precipitate the soluble salts and form harmless by-products, such as BaSO_4 . Barium salts are the favorites with which to accomplish this reaction, but others can be used. Barium chloride and carbonate are most commonly used. When these barium salts come in contact with soluble sulphates, barium sulphate is formed, which is practically insoluble in water.

In case soluble sodium compounds are present, the addition of the above-mentioned barium salts would form either Na_2CO_3 or NaCl , both of which are soluble. However, they are so soluble that they would be readily washed from the surface of the ware by the rain as soon as they formed.

According to Gerlach, a clay containing 0.1 per cent CaSO_4 , which is 0.4 gram per pound, would require 0.6 gram of BaCO_3 to perform the reaction. However, 6 or 7 grams should be added to every pound of clay for safety. This would mean practically 100 pounds per 1,000 brick. If BaCl_2 is used, but 26 pounds are required per 1,000 brick. However, when BaCl_2 is used, close to the theoretical amount must be added, for if added in excess it will form a scum itself.

This method cannot, of course, be used in the dry press process, where no water, or at least very little, is added, as the barium or other salts must be used in solution to do the required work.

Treating the Water.—When it is necessary to use a water that contains injurious soluble salts, it is simply necessary to precipitate them before adding the water to the clay.

Many good clays are being ruined by bad water, when it needs but a few hours of a chemist's time to rectify the trouble.

Preventing Scum by Correct Drying.—Scumming is often caused by incorrect drying of the ware. If ware containing soluble salts is placed in a drier and allowed to sweat slowly, the chances are all in favor of scummed ware, whereas, if the water is carried away from the surface of the ware as fast as it appears, this will not often be the case.

It is a remarkable physical fact, not yet fully explained, that where a clay contains soluble salts, they are not deposited on the surface at all, or at least to a less degree, where the drying is steady and rapid, than where it is slow and discontinuous. It seems that rapid drying causes the water to come through the pore channels of the clay as through a filter and a sort of osmosis is set up, by which the saline matters are retained inside the mass of the clay, while the water escapes as vapor from the outside. But where the drying is slow the salts get through this barrier and arrive on the surface plentifully.

It is therefore only necessary that brickmakers, in order to avoid trouble from this source, build good driers and install in them hygrometers by which their operation can be controlled.

Many other methods are used to prevent this trouble. Some of them, however, have but temporary effects.

One of these is to paint the surface of the scummed ware with oil, which for the time being covers up the scum. This is really practicing a fraud, for while the ware does not show scum when put into the building it shows up as soon as the oil evaporates.

Another method is to paint one surface of the unburned brick with a heavy oil or tar. This prevents evaporation from that surface, the salts being carried to the other surfaces. The oil or tar is burned off in the kiln, leaving a clean face. This, too, is only temporary unless the brick is vitrified, as otherwise the salts will be drawn to the clean surface after a few soakings.

According to an official publication of the Dominion of Canada forestry branch of the department of the interior, the original timber area, omitting semitreeless lands, was approximately 1,900,000 square miles; 98,000 square miles have been cleared for settlement, and 100,000 square miles have been cut over by lumbermen, leaving a timbered area yet untouched of 1,702,000 square miles. Assuming the average of 3,000 ft. per acre, there should yet remain 3,279,000,000 ft. of timber in Canada at a very conservative estimate. The highest estimate that has been made hitherto, that given by the conservation commission, places the amount of saw timber and pulp wood in Canada at 494,600,000,000 ft. and 1,100,000,000 cords, respectively.

45TH MEETING OF THE AMERICAN CHEMICAL SOCIETY.

The forty fifth meeting of the American Chemical Society will be held at Washington, D. C., December 27 to December 30. The headquarters for the meeting will be at the New Raleigh, corner Pennsylvania Ave. and 12th St. The meetings of the society will be held in the McKinley Manual Training School, corner 7th St. and Rhode Island Ave. The following program has been announced:

TUESDAY, DECEMBER 26.

Meeting of the Council at 2:30 p. m. at New Raleigh Hotel. Meeting of Council at 8:00 p. m. at New Raleigh Hotel.

WEDNESDAY, DECEMBER 27.

9:30 a. m.—Meetings of the Biological, Fertilizer, Pharmaceutical and Organic Divisions.

Joint meeting of the Section of Chemical Education and the Physical and Inorganic Chemistry Divisions. General Assembly Hall.

A. A. Noyes, Chairman. "The Teaching of Physical Chemistry."

W. D. Bancroft, "Physical Chemistry in the Introductory Course."

H. C. Jones, "The Introduction of Physical Chemical Conceptions in the Early Stages of the Teaching of General Chemistry."

Louis Kahlenberg, "The Teaching of Physical Chemistry."

J. Howard Mathews, "Some Applications of Color Photography in the Teaching of Physical Chemistry." (Illustrated.)

2:30 p. m.—General meeting of the Society. General Assembly room.

Address by President Frankforter, Section C, American Association for the Advancement of Science. Title, "The Resins and their Chemical Relations to the Terpenes."

Address by Chairman H. P. Talbot, Division of Physical and Inorganic Chemistry. Title, "Privileges and Responsibilities of the Chemical Analyst."

8:00 p. m.—Complimentary Smoker, New Raleigh Hotel.

THURSDAY, DECEMBER 28.

9:30 a. m.—Meetings of Divisions and Sections. Agricultural and Food Divisions, Room 120, First Floor.

Division of Industrial Chemists and Chemical Engineers, General Assembly Room.

Fertilizer Division, Room 103, First Floor.

Organic Division, Room 108, First Floor.

Pharmaceutical Division, Room 106, First Floor.

Physical and Inorganic Division, Room 212, Second Floor.

1:30 p. m.—Excursions.

7:30 p. m.—Address by Alexander Smith, President American Chemical Society. Title, "An Early Physical Chemist."

Lecture by Frank B. Kenrick and H. E. Howe.

Title, "Lantern Experiments on Reactions in Heterogeneous Systems."

FRIDAY, DECEMBER 29.

9:30 a. m.—General Business Meeting, American Chemical Society.

10:30 a. m.—Meetings of Divisions and Sections.

Division of Agricultural and Food Chemistry.

Biological Chemistry Section.

Division of Physical and Inorganic Chemistry.

Section of India Rubber Chemistry.

Division of Pharmaceutical Chemistry.

Symposium on Drug Assaying.

Division of Industrial Chemists and Chemical Engineers.

Special order for Friday morning is a Symposium on "Mineral Wastes and Conservation." This will consist of informal discussions of losses in the preparation and utilization of mineral products, and of wastes, inefficient methods and proposed improvements of processes in the chemical technology of mineral substances. Topics will appear on final program as well as names of specialists who will lead the discussions. Any member of the Society knowing of any instance of valuable mineral material being inefficiently used, produced or lost, or of any new use rendering mineral material available is requested to write the Secretary before the meeting, especially if he himself cannot attend.

2:30 p. m.—Continuation of morning meetings.

The Section of Biological Chemistry will hold a joint meeting with the Society of Biological Chemists.

7:00 p. m.—Subscription dinner, Hotel Raleigh.

SATURDAY, DECEMBER 30.

9:30 a. m.—Adjourned meeting of Divisions.

1:00 p. m.—Excursions.

The chief papers of interest to Chemical Engineers will probably be the following:

DIVISION OF AGRICULTURAL AND FOOD CHEMISTRY.

Fernand Brunschwig. "The Presence of Lead and Copper in Cream of Tartar and Tartaric Acid. Technical Methods to Purify These Products."

J. F. Small. "An Electrical Conductivity Test for Purity of Maple Syrup."

DIVISION OF FERTILIZER CHEMISTRY.

Paul Rudnick, Chairman's Address. "Fertilizer Chemistry—A Report of Progress."

F. B. Porter. "A Method for Testing Out Problems in Acid Phosphate-Manufacture."

Report of Committees:

Paul Rudnick, for the Committee on Nitrogen.

G. A. Farnham, for the Committee on Phosphoric Acid.

J. E. Breckenridge, for the Committee on Potash.

A. M. Peter, for the Committee on Phosphate Rock.

F. B. Carpenter, for the Committee on Fertilizer Legislation.

SECTION OF INDIA RUBBER CHEMISTRY.

Wilhelm A. Ducca. "Scientific Tests in Methods for Rubber Contents in Raw and Vulcanized Rubber."

S. P. Thacher. "On Mineral Compounds Used in Rubber."

Victor Hanzlik. Title in final program.

Also discussion on the following subjects:

"Synthetic Rubber."

"Effects of Oils and Other Adulterants in Reclaiming Rubber."

"The Treatment of Crude Rubbers."

"Uses of Ceylon Rubbers."

It is expected that other papers will be announced on the final program.

DIVISION OF INDUSTRIAL CHEMISTS AND CHEMICAL ENGINEERS.

H. C. Sherman, D. A. Bartlett and N. E. Weatherless. "Relation of Ultimate Composition to Calorific Power in Coal."

H. C. Sherman and S. H. Regester. "Relation of Proximate Composition to Calorific Power in Coal." (Preliminary Report.)

G. A. Burrell. "New Forms of Gas Analysis Apparatus." (Illustrated by Lantern Slides.)

F. M. Williams. "New Forms of Apparatus for Gas Analysis."

J. T. Baker. "Problems in the Manufacture of C. P. Acids."

J. R. Cain and J. C. Hostetter. "A Rapid Method for the Determination of Vanadium in Steels, Ores, etc., Based on its Quantitative Inclusion by the Phosphomolybdate Precipitate."

Ernst Bidtel. "Valuation of Fluor-Spar."

L. B. Lockhart. "The Quality of Commercial Kerosene."

John P. Simmons. "The Effect of Filtration Upon the Physical Properties of Petroleum Oils."

Franklin Peale Summers. "The Product Patent."

H. H. Willard. "The Preparation of Perchloric Acid."

Chas. H. White. "A Calorimeter for Rapid Work With Widely Varying Standards."

Irving Langmuir. "The Melting Point of Tungsten."

J. I. D. Hinds. "Rapid Precipitation of Arsenic, Antimony and Tin."

As a result of its investigations into the alcohol industry of the Philippines, through Dr. H. D. Gibbs and others, the Bureau of Science of the Philippine Government has called the attention of the business world to the probability that sugar can be made from the nipa palm more economically in the Philippines than from sugar cane, and that the possibilities of sugar production from this palm constitute a matter of surpassing importance to the islands. The conclusions and the data of the investigations upon which they are based are included in a series of reports printed in the Philippine Journal of Science for April and for June, 1911. The researches involved have covered the greater portion of the past two years, and have represented some of the best work of the Bureau of Science.

FOURTH ANNUAL MEETING OF THE AMERICAN INSTITUTE OF CHEMICAL ENGINEERS.

The fourth annual meeting of the American Institute of Chemical Engineers will be held at Washington, D. C., December 20 to December 23. The headquarters of the institute will be at the New Willard Hotel. The program of the meeting is as follows:

Wednesday, December 20, 1911.

9:30—Meeting at New Willard Hotel.

Address of Welcome.

Business Session:

Canvass of Ballots for Officers.

Reports of Officers and Council.

Report of Committees.

10:30 a. m.—Reading of Papers:

Advances in Testing Explosives—Clarence Hall, expert in charge of Explosives Section, Bureau of Mines, Pittsburg, Pa.

Distribution of Power in Portland Cement Manufacturing—Richard K. Meade.

Manufacture of C. P. Acids—J. T. Baker.

Combustion of Pulverized Coal—L. S. Hughes.

Manufacture of Gelatin—Ludwig Thiele.

The Natural Bituminous Rocks of the United States—S. F. Peckham.

Wednesday, December 20, 1911.

1:30 p. m.—Excursion.

Visit to Bureau of Standards. It will not be possible to inspect all departments of this bureau in a single afternoon. The members of the Institute and their guests will be divided into three parties, making the following inspections:

1. Inspection of the adjustment of pyrometers, thermometers, pressure gauges and similar instruments.

2. Inspection of the standardizing of weights and measures.

3. Inspection of liquid air apparatus which will be in operation for the benefit of the visitors.

8 p. m.—Meeting at New Willard Hotel.

Address of Retiring President—F. W. Frerichs.

Some Problems in Chemical Engineering Practice.

A—Manufacture of Chloroform from Alcohol.

B—Construction of Laboratory Apparatus.

C—Manufacture and Testing of Shipping Cylinders for Liquid Ammonia Gas.

Manufacture and Testing of Carbonic Acid Cylinders—John C. Minor, Jr.

Thursday, December 21, 1911.

Excursion. Visit to U. S. Proving Grounds at Indian Head, Md.

Immediate notice by those who desire to attend should be given to the secretary, stating nativity, citizenship and business connection, as the list of those who desire to attend must be submitted for approval.

The proving grounds comprise plants for contact, sulphuric, nitric and mixed acids, the nitration of cellu-

lose and recovery drying; powder presses, mines and heavy artillery.

7 p. m.—Subscription Dinner at New Willard Hotel, \$3.

Friday, December 22, 1911.

9:15 a. m.—Meeting at New Willard Hotel.

Installation of Officers.

Business Session.

10 a. m.—Symposium on the United States Patent System.

The United States Patent Office—E. B. Moore, Patent Commissioner, U. S. Patent Office.

Patent Office Procedure.

Practice of Patent Law up to the Issue of the Patent—Mr. Edw. T. Fenwick, of Mason, Fenwick & Lawrence.

Protection of Inventions by Patents. Existing Defects and Remedies Therefor—Walter D. Edmonds, of Edmonds & Peck.

The United States Patent System—R. N. Kenyon, of Kenyon & Kenyon.

Discussion.

Friday, December 22, 1911.

2:30 p. m.—Reception at White House.

The President will receive the members of the Institute in the East Room of the White House.

3 p. m.—Visit to Patent Office.

8 p. m.—The Adaptation of the Centrifugal Pump to Chemical Service. Illustrated with lantern slides—F. G. Wheeler.

A complete discussion of the theory of this pump, all designs and makes and their behavior under operating conditions.

A Course in Chemical Engineering Education, prepared in accordance with the vote of the Institute on the Questionnaire of December, 1910—J. C. Olsen.

Discussion.

Business Session.

Saturday, December 23, 1911.

Excursion.

Alternative. Steel Plant at Sparrows Point, Baltimore.

Cement Plant of Tidewater Portland Cement Co.

The production of sugar in Brazil has been practically stationary for many years. The annual output during the last decade has been approximately 250,000 tons. Of this total 30,000 to 50,000 tons have been exported annually. In 1909 the exports amounted to 69,500 tons, and this figure is practically a maximum one for the exportation possibilities of the sugar crop in Brazil. The size of the crop varies from year to year, largely depending upon the weather conditions and but little upon the efforts of the planters. The development of the industry has been very slow, and methods of production are at the present time almost exactly as they were a century ago.

PHYSICAL AND CHEMICAL PROPERTIES OF PETROLEUM OF THE SAN JOAQUIN VALLEY, CAL.

The United States Bureau of Mines, in following out its investigations into petroleum, has just issued Bulletin No. 19, by Irving C. Allen and W. A. Jacobs, on the Physical and Chemical Properties of the Petroleum of the San Joaquin Valley, California.

In detailing the investigations which are to be conducted by the Bureau of Mines, the authors say in their introduction:

Realizing the great importance and wide application of petroleum and its products for fuel, lighting and lubrication, and the absence of authentic and comprehensive information in the literature on the commercial value of American petroleum, the United States Geological Survey has for a number of years been making analyses and other examinations of these oils and their products. In 1907 the technologic branch of the survey began a study of certain economic problems having to do with the natural mineral oils. The prime object of the investigations was to determine the economic value of the petroleum and natural gases in the United States with special reference to the supplies in the public lands, and to ascertain how these resources could be utilized in the industries with the least waste and greatest efficiency.

This work, which is now being conducted by the petroleum section of the Bureau of Mines under authority of the act of May 16, 1910, will include the collection at the wells and refineries of information as to the following details:

Facilities for handling and storage at the wells. Facilities for transportation, whether by pipe line, rail, or water.

Distance to the nearest port or center of industry and time necessary for delivery. Facilities for storage at such point of delivery. Suitability of the crude oil for fuel, or the treatment necessary to convert it most economically into the best fuel; best processes of refining or of converting the crude oil into its many products; the examination of these products; the utilization of the by-products and residues of the refineries; the collection of adequate and reliable samples, and the complete chemical and physical examination of these samples in the laboratory.

It is hoped that the information thus obtained will render possible the establishing of suitable specifications for gasolines, kerosenes, certain lubricants, and fuel oils. In the near future a study will also be made of the problems involved in the most economical utilization of the heavy crude oils in the various types of internal-combustion engines.

The gas-producer problem, because of its importance, will also be investigated, as a producer capable of converting the heavy oil residues directly and completely into a combustible gas would be of paramount value to the oil industry.

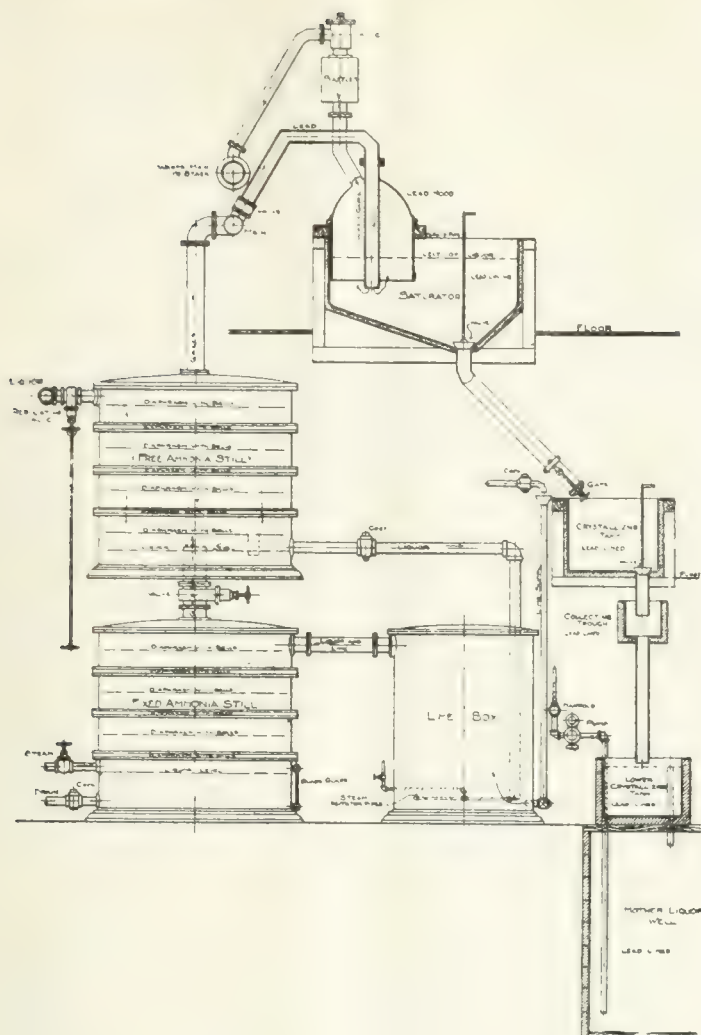


Fig. 2—Elevation Showing Parts of One Set of Apparatus.

liquor before evaporation, to obtain a more neutral sulphate; but this procedure largely increases the amount of liquor to be evaporated, and it is dangerous to the operator, because of the liberation of large quantities of hydrogen sulphide.

Acid is received in tank cars of 50 tons capacity. They are unloaded by air pressure into an underground tank, a rubber hose with brass connections being used for the discharging connection. From this tank the acid flows by gravity to an air-operated pump that elevates the acid to storage tanks under the roof of the building. The discharge from the pump is into a lead drum above the tanks, to provide for the air discharge with the acid, thus preventing any splashing. The outlets of these storage tanks are in the bottoms, and can be closed by a plug in case of emergency. Below these plugs in the pipe line is a valve for each tank to facilitate repairs to the line and tanks. The acid runs by gravity to each saturator with a valve for each outlet, and, further, a valve on a low point on a pipe for draining in case of repairs. All of these valves are of cast lead of the globe valve type.

The operation of the saturator is as follows: The box is filled with acid and mother liquor, or, not having sufficient mother liquor, with water to the proper height,

making the strength 38° to 40° B. This strength should vary according to the salt capacity of the box, and should be such that the box is not choked with salt before the acid is neutralized. After turning on the gas, as soon as the bath gets near the boiling point the operator collects the scum as previously described. For the next 2 or 3 hours no attention is necessary, other than to see that the proper temperature of the bath is maintained. This is best done by observing the temperatures of the inlet gases. If they rise above 95° C. there is danger of steam condensation in the box, which increases the volume and may cause an overflow. When the salt begins to deposit freely, it is necessary for the operator to push back the salt from the curtains to assure a circulation of the acid, otherwise the salt will be blackened or "burnt," as it is called. When the strength of the bath drops to 32° B. the gas is shut off, as that is almost the neutralization point. If the bath is allowed to become alkaline it will turn black from sulphides and spoil the color of the salt.

The salt and mother liquor are now dropped into the settling box through the outlet pipe, the mass being allowed to remain for several hours to cool, in order that as much salt as possible will crystallize out of the mother liquor. It is then drained, shoveled into a cart, taken to the centrifugal machine and dried, then wheeled to the storeroom.

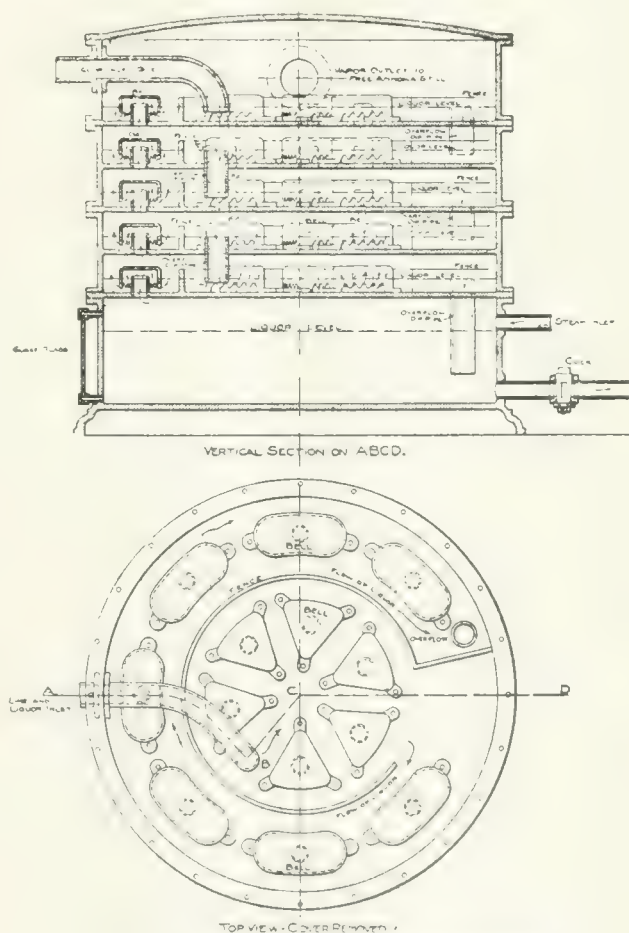


Fig. 3—Diagram of Fixed Ammonia Still.

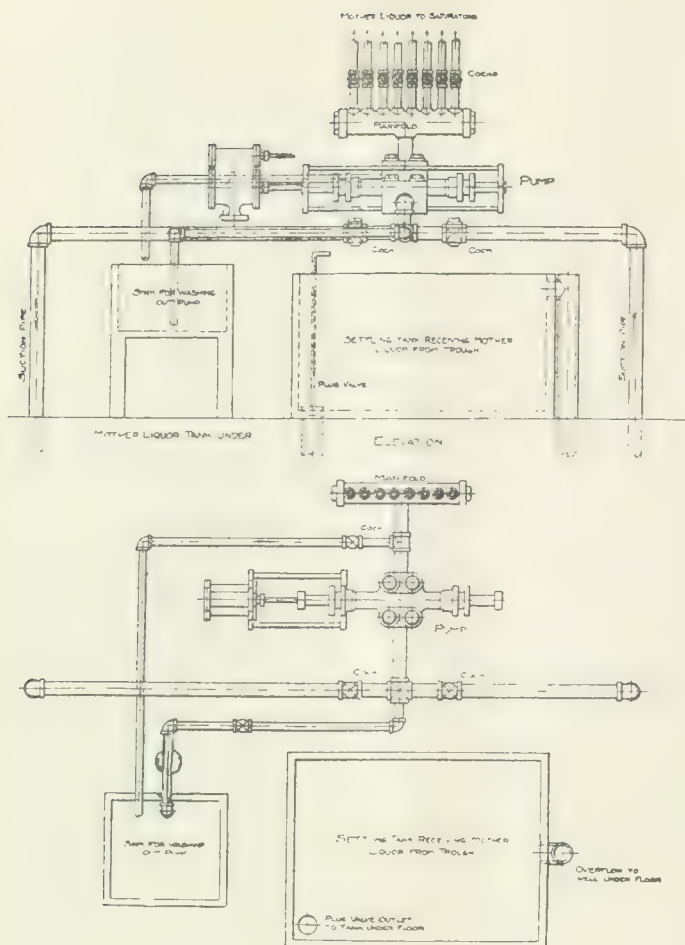


Fig. 4—Arrangement of Pump and Connections for Molten Liquor.

The handling of the mother liquor is somewhat difficult, because as it cools, salt crystallizes out very freely and quickly closes the pipes. All the mother liquor from settling boxes, centrifugal machines, etc., collects in lead-lined cisterns under the main floor of the building, several catch-boxes being interposed on the way to remove the salt that is carried along mechanically, and thus keeping down the accumulation in the cisterns.

The pump and pipe connections for mother liquor are shown in Fig. 4. A brass pump is used for elevating the mother liquor from the cisterns to the saturator, the piping arrangements being shown on the drawing. A gum rubber suction hose is used in the cistern and connects above the floor to brass pipe. This makes a flexible arrangement for the varying amounts of salt and liquor in the cisterns. The manifold drum on the discharge is to prevent the crystallization of the mother liquor in the pipes when not in use, as each pipe is drained after being used. The pump and connections are frequently washed out with water to prevent accumulation of salt, the water being pumped from the sink through the pump and back into the sink.

The general arrangement of the plant is shown in Fig. 1, and a diagram elevation of one set of apparatus is shown in Fig. 2.

It will be noticed that the stills arrangement is three of lime to each two free stills. The reason for this is largely because for several years the coal used in the plant gave a high percentage of mixed ammonia compounds, requiring the use of a large amount of lime. This, of course, necessitated frequent cleaning of the stills, and with this arrangement two stills could be out of commission all of the time. The fixed ammonia still is shown (in plain and suction) in Fig. 3.

The liquor is supplied to the free stills by gravity from an elevated tank, giving a practically constant head. Feed valves are of the all-iron style with long extension stems to enable operator to reach them from the floor. Steam for the stills is supplied principally at the bottom of the lime, some in the lime box for agitation, with a small auxiliary supply at the bottom of the free still. In both cases boiler pressure steam is used, being throttled as admitted. The amount is governed wholly by the operator's tests of the waste liquor and the temperature of the distilled gases. The saturators are placed on an upper floor. This has many advantages, principally that the handling of the salt is largely by gravity; it also prevents boiling over of the stills when being forced, and thus prevents iron salts from being deposited as ferrocyanides in the sulphate, with their well known blue color.

The saturators (shown in Fig. 2) are of a wooden frame, square type, with double square bells. Practically all of the lead work is $\frac{1}{2}$ in. thick and is serv-

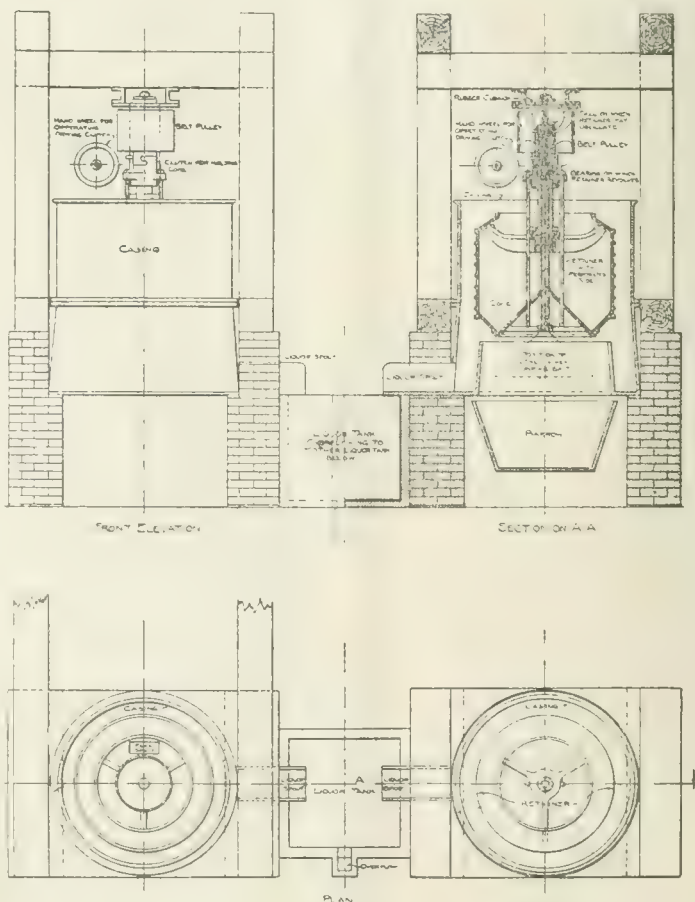


Fig. 5—Centrifugal Device for Sulphate of Ammonia Plant.

iceable without repairs for at least five years, while small repairs will add three more years to the box before relining. The capacity of the box is about 5,000 lbs. dried salt.

The plug shown in the bottom of the box is merely for the safety of the operator in removing the valve on the bottom of the outlet pipe. This valve is simply a flat plate with a gum rubber disk held by pressure against the end of the pipe. The salt, after draining, is shoveled into a square two-wheeled wooden cart, which is wheeled to the centrifugal machine, being dumped through a brass chute into the basket. The basket is run at slow speed until the charge (of about 800 lbs.) is all poured in. This is necessary because of the sticky nature of the sulphate, as fast running

the paper is to point out some of the many troublesome points which are not usually mentioned in textbooks, yet which are of prime importance in the maintenance and operation of an ammonium sulphate plant.

The Bengal, India, Chamber of Commerce, according to a U. S. Consular Report, recently took up the question of watering jute to excess, and it was agreed that a certain percentage of water was allowable at certain seasons of the year to comply with the terms in the contract that the jute be in a "sound, dry, storing condition." The following memorandum on the subject was agreed to: (1) That in the months of July and August a fair allowance for moisture would be

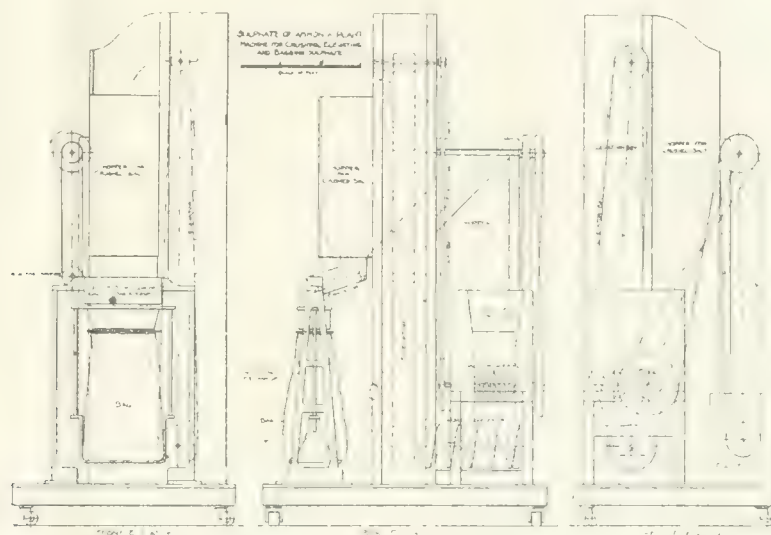


Fig. 6—Sketch of Machinery for Crushing, Elevating and Bagging Sulphate.

will cause uneven loading, with possibly disastrous results.

A drawing of the centrifugal machine is shown in Fig. 5. The basket has a drop bottom that is raised by a lever and secured by a clutch. The basket, bottom, shaft, etc., are all of brass, and have a life of from $1\frac{1}{2}$ to 2 years. The envelope and liquor collector around this basket are made of sheet lead, which forms a very satisfactory and safe casing. It is essential that the sulphate should be allowed to remain in storage for four or more days before shipping, to prevent shrinkage in transit, so accordingly it is piled on a wooden floor. From storage the salt is loaded into bags with an automatic weighing machine, shown in Fig. 6. The salt caking in storage is broken up by picks and shoveled into a crusher that disposes of all the lumps. It is elevated by a bucket conveyor into a hopper over the bagging machine, into which it is fed by gravity. The capacity of the machine is from $3\frac{1}{2}$ to 4 tons per hour, which is the capacity of the crusher.

The drawings give many details that have not been included in this description. The primary object of

12 per cent; (2) that in the months of September and October a fair allowance for moisture would be 10 per cent; (3) that in the remaining months of the year moisture should not exceed 8 per cent; (4) that the following should be the method to be adopted for ascertaining moisture, viz., the jute selected for testing should be accurately weighed and then exposed to the sun for two hours, and should be accurately reweighed immediately thereafter. If the moisture evaporated should prove to be more than is allowable for the time of the year, according to the foregoing scale, it would mean that the jute is not in accordance with the requirements of the memorandum.

It is estimated that at the close of 1910 there were in Ceylon 185,000 planted acres of rubber and 12,000 bearing acres of rubber. A local writer has estimated that 220,000 acres will be the maximum area planted with rubber in the island, and that, with 140 trees to the acre average, and $1\frac{1}{2}$ lbs. yield per tree per annum, this will give an export of about 20,000 long tons by 1920.

AUTOMATIC REGULATING DEVICES FOR WATER PURIFICATION PLANTS.

Many industrial plants using a surface water supply have installed water softening or purification plants. One of the essential details of these water purification plants where chemical treatment is employed is the obtaining an efficient control of the chemical solutions used. In an interesting article in *The Industrial World*, Mr. Thomas Fleming, Jr., of Chester & Fleming, Hydraulic Engineers, Pittsburgh, Pa., points out a few methods for handling this essential detail. Indirectly Mr. Fleming's article also gives an insight into one of the serious problems of the art of purifying water, which should warn the novice against attempting the construction of costly works without expert advice. Mr. Fleming's article follows:

The chemicals most frequently used in water purification are sulphate of alumina, soda ash, lime, sulphate of iron, and hypochloride of lime. These chemicals must be placed in solution, and fed to the water in definite quantities. Some plants are equipped to handle a combination of one or more of these, while at many only one is needed. The chemicals to be used depend to a great extent on the nature of the water to be treated and the results desired. There are some instances where one or more methods might be used successfully and under these conditions the combination selected is largely a matter of judgment.

The most common treatment is the addition of sulphate of alumina to the water in small quantities, varying from a fraction to several grains per gallon. Where there is not sufficient alkalinity in the water to combine with the alum, lime or soda ash is added. Frequently hypochloride of lime is used with alum during certain periods of the year. Iron sulphate and lime are more suitable for the coagulation of some turbid waters, especially those of the middle West, but are sometimes used at plants where the turbidity varies widely as an alternative to the alum lime treatment. It is not unusual to find a plant where all these chemicals are used at different times of the year. The amount used varies with the turbidity, color and chemical composition of the water to be treated and where used as a disinfectant with the bacterial count. Some waters are quite stable in composition and require little change throughout the year in the quantity of chemical used for treatment; others vary widely. The entire flow of the Monongahela river, which is the most difficult water in America to treat, is used for domestic and industrial purpose, over six times in the dry season before reaching Pittsburgh. During this period it runs strongly acid while in the wet season and during high stages of the river, it is strongly alkaline. In addition at times after protracted droughts it is occasionally colored with vegetable matter that gives it a tea color. This is reported to be due to wild laurel which has been steeped in swampy places near its headwaters. This river also has a wide variation

in the amount of suspended matter and it can be seen that under these conditions the amount and character of coagulant used will have a wide variation and the chemical treatment plant must be equipped with very elastic apparatus.

The success of a mechanical filter plant depends upon the proper coagulation of the raw water and an efficient setting out of the suspended matter in the preliminary settling basins. To obtain this coagulation economically the chemicals must be added so as to entirely combine in forming the flocculent precipitate which settles out the suspended matter in the settling basin and forms the gelatinous "schmutzdecke" on the surface of the filter. This can only be obtained by efficient apparatus where the chemicals can be mixed in solutions of known strength and can be accurately fed to the water to be treated.

There are two methods of feeding the coagulant to the water to be treated, viz.: by gravity or by pumping. These two methods divide the apparatus into two distinct types, gravity feed and pressure feed. In both types the apparatus may be further sub-divided into the mixing and solution tanks and the regulating apparatus. The method to be used depends upon local conditions. It is considered preferable to use the gravity feed where possible, but there are sometimes cases where this is out of the question.

If the chemicals were easily dissolved in water and would remain in solution indefinitely the problem of mixing would be a simple one. Such, however, is not the case. While there is a variation in the solubility of the various substances, yet generally speaking it can be said that none of them are perfectly soluble in water to any degree of concentration without agitation. For large plants it is therefore necessary to provide mechanical or air agitators for the solution tanks to prevent a precipitation of the chemical in solution. For small installations it is customary to omit agitators on account of the extra expense and often on account of the difficulty of providing power and to allow for a certain amount of precipitation by feeding the chemical at a slightly higher rate. Where hot water or steam is available the practical operator will arrange to heat the solution. This simple expedient is of great assistance as it raises the point of solubility and in the case of slaking lime, greatly assists the chemical reaction. This great secret in the successful design of mixing and solution tanks is to furnish ample capacity for mixing the chemical in a weak solution. This is especially valuable in small plants where a concentrated solution in addition to the difficulty of mixing, will furnish such a small quantity in the outlet pipes that it will frequently clog up and will be very difficult to preserve a uniform flow.

For chemicals that are easily dissolved, such as alum, the mixing and solution tank are one and the same. The chemical is placed in a small compartment near the top of the tank and is dissolved as the tank is slowly filled with water. If the plant is of any size

compressed air is blown through the mixture from a grid in the bottom or a mechanical agitator constantly stirs it.

Lime or hypochloride of lime should be mixed in a compartment separate from the solution tank to secure the best results, if there is a considerable amount used. It is customary to slake the lime in a separate tank and to drain off the saturated milk of lime solution to a solution tank. Hydrochloride of lime must be thoroughly mixed by hand and then thinned out to a dilute solution for the solution tank. The solution tanks as well as the mixing tanks should be constantly agitated to preserve a uniform solution and to prevent precipitation.

The regulation apparatus that controls the feed from the solution tanks may vary from a simple gate valve to an orifice automatically controlled by floats or other devices. It is usually located below the bottom of the solution tanks and is connected to them by small piping. This piping should be sufficiently large to permit cleaning in case of clogging and should be of such material as will resist the corrosive action of the chemical to be handled. In some instances it has been found cheaper to use ordinary galvanized iron pipe and replace it as it wears out than to purchase brass pipe or lead lined pipe with a little longer life. This is especially the case with lime solutions. Lead pipe seems to be the best adapted to hypochloride of lime solutions and brass pipe to alum solutions.

The details of the regulating apparatus will depend to a great degree on local conditions. If there is a uniform amount of water to be treated with a constant flow it is usual to feed the solution through an orifice, either varying the strength of the solution as the condition of the water to be treated changes or varying the size of the orifice or both. The capacity of orifice may be varied by using a variable orifice or by varying the head of the orifice. In this case the flow of water to be treated varies, it may be necessary to further control the flow of the solution so that it can automatically vary in proportion to the flow. This can be done by floating orifices or by an orifice controlled by floats. In pressure feeds under a varying flow of the water to be treated, a feed pump is attached to the pump handling the water to be treated. This feed pump is so attached that its strokes will be in proportion to those of the larger pump and the solution is very accurately introduced in proportion to the water pumped.

APPARATUS FOR CHARLEROI FILTER PLANT.

The apparatus recently installed by us in the new Charleroi Water Co. filter plant is of the gravity type. The water to be treated is Monongahela river water which, as previously stated, is very variable. The filter plant has a capacity of 3,000,000 gals. and is equipped with a preliminary settling basin and six 500,000 gal. gravity filter units. The coagulant house is located at one end of the filter house. It has been made quite elastic and ample space has been provided for meeting all emergencies. The coagulant house is

a three story brick structure. It is 12x40 in plan and 30 ft. high. Only half being used for the laboratory and reception room. The solution tanks rest on the second floor with their tops flush with the top floor which is used as a charging floor. There are five of these tanks. They are constructed of reinforced concrete and are $\frac{3}{4}$ -in. by 4 ft. in plan and 9 ft. deep. Two of the tanks are used for alum, two for lime or soda ash and one for hypochloride of lime. The mixing is done in small compartments at the top of the tanks which are really a part of the main tank and the solution is kept constantly agitated by mechanical agitation. This consists of a set of horizontal paddles located approximately 2 ft. from the bottom of each tank and attached to a vertical steel shaft which is driven by a belt connected to a water motor located on the floor below. Each vertical shaft is independently controlled by a clutch. The agitators in the lime tanks are further provided with four chains 2 ft. long which hang vertically to the bottom of the agitator and prevent the lime from depositing on the bottom. The third floor in front of the tanks is reserved for charging and operating. The balance is used for storage. A hand elevator is used to bring the material up from the ground floor.

Each tank is provided with independent water connections. The outlets are located near the bottom of the tanks on the second floor and are controlled by standard orifice boxes of enameled iron. Each tank is provided with an independent orifice box. These boxes have a circular orifice in the bottom controlled by a cut-off plate which is so arranged as to vary the size of the opening. This is controlled by a graduated dial and by setting the dial any desired rate of flow can be obtained. The inlet from the solution tank is controlled by a copper float and valve which keep a uniform head on the orifice. From the orifice box the solution flows by gravity to the supply main leading to the settling basins.

The chemicals at this plant are mixed once each day. The duplicate tanks permit this to be done thoroughly while the solution is being fed from other tanks. Although the water is difficult to treat and makes rapid changes in composition, yet it has been possible for one man to operate this filter plant and the nearby pumping station and boiler house and to obtain excellent results. This is to a large extent due to the ample facilities for mixing and handling the coagulants. This arrangement of the apparatus is similar to that installed a few years ago by Mr. Chester for the filter plants at Portsmouth, Va., and South Pittsburgh, Pa. It is the result of a long study of gravity feed of chemicals at various filter plants and its success cannot be attributed to any one plant.

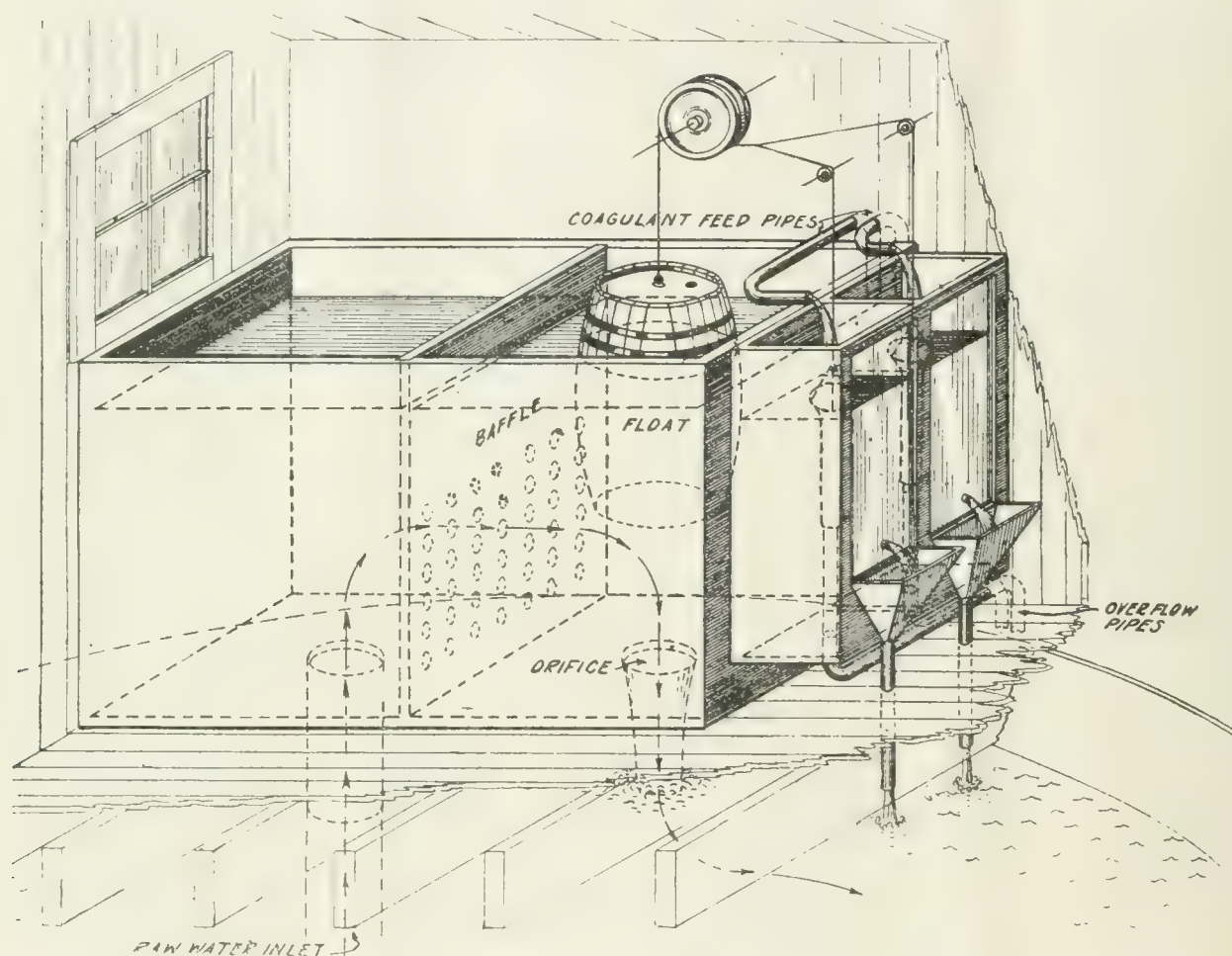
APPARATUS FOR MONESSEN FILTER PLANT.

The apparatus for the Monessen Filter plant comes under the pressure type of feed and involves some unique features which are the result of a study of peculiar conditions to be met. The filtration plant is

located on a hillside above Monessen and the location is such that a pressure feed must be used. The coagulant house is a two-story structure. The solution tanks are located on the first floor and the second floor is used for storage. The main tank used for mixing is 10x10 feet in plan and 10 feet deep and the top is flush with the charging floor. From this tank the coagulant is drawn off by gravity with smaller tanks on the lower floor. These tanks in turn supply a water lifter which discharges against a head of 40 ft. into the raw water main leading to the settling basin.

The inlet pipe enters and is baffled to prevent too much disturbance to the water in the compartment. A large float consisting of a whiskey barrel completes this part of the apparatus.

The coagulant pump lifts the solution to a set of feed boxes located sufficiently high above the raw water compartment to discharge by gravity. Each of these boxes is 8 in. by 8 in. by 8 ft. deep. It has a plate glass front and a small orifice drilled in the front near the bottom. In the center of the compartment is a horizontal movable orifice connected to a return



Sketch Showing Arrangement of Coagulant Feed Apparatus at Monessen Filter Plant.

As the pumps are located on the Monongahela river bank a mile distant, it is important to have an accurate control for the coagulant pump as otherwise coagulant would be fed to the settling basins when the main pumps stop and vice versa.

An automatic float control arrangement was designed for this purpose. A cubical compartment 4x4x4 feet deep was built on the upper end of the raw water pipe where it enters the settling basin. This compartment is placed above the flow line of the basin and a circular orifice is cut into one side at a point 18 ins. above the bottom. This orifice is ten inches in diameter, which is sufficiently large to discharge the maximum pumpage without overflowing the com-

partment. The inlet pipe enters and is baffled to prevent too much disturbance to the water in the compartment. A large float consisting of a whiskey barrel completes this part of the apparatus. The coagulant pump lifts the solution to a set of feed boxes located sufficiently high above the raw water compartment to discharge by gravity. Each of these boxes is 8 in. by 8 in. by 8 ft. deep. It has a plate glass front and a small orifice drilled in the front near the bottom. In the center of the compartment is a horizontal movable orifice connected to a return

APPARATUS FOR THE PENNSYLVANIA WATER COMPANY.

The Wilkesburg filter plant of the Pennsylvania Water Company is equipped with chemical apparatus

very similar to that used at Charleroi. The plant has a capacity of 10,000,000 gallons per 24 hours, in comparison to the 3,000,000 gallon capacity of the Charleroi plant. However, Allegheny river water is handled and this is partly clarified by a crib in the river, so that the conditions to be met are much more favorable, and very little more solution tank capacity is needed than in treating the smaller quantity of Monongahela river water.

The main feature of this apparatus is the method of control. Instead of standard orifice boxes, a special form of box was designed which has proven very flexible. This consists of a reinforced concrete compartment approximately 2x2 in plan and 5 ft. deep. The front of this compartment is plate glass and to this is bolted a circular brass plate, the bolt being through the center of the plate and controlled by a thumb screw. Near the circumference of this plate are a series of circular orifices ranging in size from a fraction of an inch in diameter to several inches in diameter. On the same radius in the plate glass front of the orifice box there is an orifice slightly larger than the largest one in the plate. By revolving the plate any one of the orifices in it can be brought over this opening, thereby changing the discharge. The plate glass and the ground metal surface of the brass plate prevent any leakage around the orifice. There is a large copper float in the compartment which is adjustable and controls the inflow pipe from the solution tanks. By adjusting this float further elasticity in control of the solution can be obtained by raising or lowering the head of the orifice.

APPARATUS FOR VALLEY CAMP.

The Valley Camp equipment is different from the types previously described. It is to be used to disinfect the effluent from a sewage plant and handles hypochloride of lime only.

It is located in a small frame building near the outer end of the disposal plant and consists of two wooden solution tanks and the control apparatus. The solution tanks are to be located on a platform several feet above flow level, so as to give sufficient room to feed by gravity through the control apparatus which is located on the floor.

These tanks each have a capacity of one day's maximum supply of hypochloride solution. The hypochloride is to be mixed by hand in a bucket and then diluted in the solution tank. Each tank is provided with running water connections. The outlet is located near the bottom and is connected by a one-inch valve connection to a galvanized iron feed pipe which leads to the control box. This box is located on the floor at one side of the tanks. The inlet from the tanks is controlled by a float valve and the outlet consists of an adjustable orifice which feeds through a funnel to the disposal plant effluent in the sewer below the flood. The flow of this sewage is very variable, and in order to prevent any undue wastage of chemicals it is neces-

sary that it should be automatically controlled so as to feed at a given rate in direct proportion to the amount of sewage.

The adjustable orifice is therefore arranged as follows: It consists of a $\frac{1}{4}$ -in. glass tube bent so as to discharge downward, and connected by a rubber tube to an opening of the same size in one end of the feed box. This tube is attached to a brass clamp which runs in a vertical slotted groove along one side of the box by which is attached a graduated scale. The clamp is connected by a small steel wire to an overhead pulley wheel. This wheel is in turn connected by a wire to a float which is adjacent to the outlet sewer under the floor of the building. There is a triangular notched weir in this sewer so that the heights of sewage in the float chamber can be made to accurately correspond to the different rates of flow of the sewage. The pulley wheel is differential and in this case at a point just below the flow line of the control box for a zero reading over the weir in the sewer. Then for any increase in the sewage flow the orifice will drop four times the distance that the float rises and with the drop the flow of chemicals will increase. The ratio to be used depends on the amount of chemical to be used and size of orifice. This method of feed has been found by the writer to be the most successful for handling hypochloride of lime in small plants where the amount of solution used is very small. Standard control boxes are as a rule failures as the orifices quickly clog up. A comparatively large orifice such as the above with a variable head has proved very satisfactory and is simple in adjustment.

In the general estimates for appropriations for the fiscal year 1912, which begins July 1, 1912, Secretary of the Interior Walter L. Fisher has recommended the following items for the Bureau of Mines: For the investigation as to the causes of mine explosions, methods of mining, especially in relation to the safety of miners, the appliances best adapted to prevent accidents, the possible improvement of conditions under which mining operations are carried on, the use of explosives and electricity, the prevention of accidents, and other inquiries and technologic investigations pertinent to the mining industry, \$360,000; for the investigation, analyzing and testing of the coals, lignites and other mineral fuel substances belonging to or for the use of the United States, \$135,000; for investigations into the treatment of ores and other mineral substances, with special reference to the prevention of waste in the mining and utilization of important mineral resources, \$100,000; for the investigation of the coals of Alaska, with reference to their mining, transportation and utilization, \$50,000.

Rubber exports from Ceylon this year up to Oct. 2 were 4,064,180 lbs., or just about double those in the same period last year. Direct shipments to the United States were 1,303,889 lbs., an increase of 368,712 lbs.

IMPROVED SULPHURIC-ACID CHAMBERS.

On October 10, 1910, three of the nine chamber plants of the Union Fabrik Chemischer Produkte at Kratzwieck, near Stettin, were destroyed by fire. In planning for rebuilding the director decided to adopt the essential features of the so-called Moritz system of chamber construction, as not only promising economy in future cost of production, but also offering exceptional facilities for rapid replacement of the missing units, even during the winter season. He was able to use the first series of chambers on March 15 and the second on April 4 of this year. The construction of the third unit did not begin until July. Thomas H. Norton, the U. S. Consul at Chemnitz, Germany, has recently visited these works, and in a recent Consular and Trade Report, he gives the following description of the new chambers:

The Moritz system was brought to the attention of technical chemists about four years ago. It was first introduced at Wasquetal, northern France, by the Société anon. des Établissements Eyken et Leroy, with which the inventor, R. Moritz, is connected. The results obtained in a chamber of 4,500 cubic meters (cubic meter = 35.314 cu. ft.) were so satisfactory that several French and Belgian works utilized the new method in increasing their chamber capacity.

At present a large system of 8,000 cubic meters is under construction at Hainaut, Belgium, a smaller one of 4,000 meters at Elberfeld, Germany, and the third unit of the "Union" works (5,200 meters).

As a rule, in chambers constructed according to prevalent methods, the sides, the top, and the bottom of a lead chamber are supported by strong wooden beams. These timbers, while showing externally no change, are seriously affected by the radiation of heat from the adjacent lead walls. The temperature of the walls may attain 90° and even 100° C. (194° and 212° F.), and as a result the wood rapidly loses its strength, becomes brittle, and is warped. This leads to a dangerous strain upon the flaps along the walls. They are frequently torn off and the wall loses its shape. This tendency is heightened by the great difference between the coefficients of expansion of lead and wood. The thoroughly dry timber offers also an undesirable amount of very combustible material, as shown by the extensive fire at Kratzwieck in 1910 and the conflagration at the Wasquetal works in 1907.

Much more serious, however, is the increased corrosion of the lead chambers, due to the use of wooden supports. Those portions of the lead walls adjacent to the timber supports are prevented from radiating heat freely. This retards the oxidizing reaction in the immediate vicinity, while allowing the lead to be corroded more rapidly. Careful measurements of the thickness of the wall in different parts of a lead chamber, after the fires mentioned, showed that the portions of wall screened by a timber were on an

average half a millimeter (0.01969 in.) thinner than other sections of the chamber. This was in spite of the fact that the timbers were separated by spaces of 8 to 10 centimeters (3.15 to 3.94 ins.) from the lead walls.

An additional disadvantage results from the more rapid corrosion of the lead behind a beam, as leaks occur thereby most frequently at points where the repairs are difficult to execute. In some few works iron instead of wooden beams have been introduced. This obviates the distortion of the chamber walls, due to the warping of the wood, but does not diminish the trouble arising from rapid corrosion and the difficulty in carrying out repairs.

With regard to the bottom of chambers, observation shows that corrosion is very pronounced along the lower edge, where the lead is bent upward. Here a zone of about 6 ins. is peculiarly liable to rapid attack; and the same is true of the lower part of the side walls, which form the hydraulic joint. The direct cause of this quick corrosion is the thick flooring of wood necessary to support the weight of the acid, but retarding the radiation of heat from the surface of the lead.

The tops of chambers are ordinarily flat. The lead sheets are attached by flaps to overhead beams or hung upon iron rods or tubes. In both cases the same disadvantages exist as have been previously mentioned. A flat roof also favors the accumulation of dust, which in turn retards radiation. As a rule it is a matter of great difficulty thoroughly to clean the ordinary flat roof of an acid chamber.

The prevalent method of attaching the various sections of a lead chamber to its supports prevents almost entirely any uniform expansion of the different parts. In consequence of the heavy weight of the lead, and of the uneven expansion, there is a constant tendency toward strain about the flaps, and cracks appear along the soldering lines that are exceedingly awkward to mend.

All of these defects of the customary construction seem to be avoided in the Moritz chambers. The essential feature of such a chamber unit is a strongly built skeleton of structural iron, inclosing the chambers, which are suspended from overhead beams. The skeleton serves also as a support for the roof.

The main columns of this skeleton, which bear the weight of the roof, have been constructed in some cases of concrete or masonry. Preference is, however, given to iron, as involving less time in preparation and assuring greater strength. A tile roof is used. The outer walls of the structure are built of brick, laid between the upright columns. The bricks are placed edgewise, so that the walls are exceedingly light, 2.36 ins. in thickness. No windows are required. An abundant use of panels of wire-net glass and of ordinary glass tiles in the roof affords all needed light. Some additional light and abundant ventilation are assured by numerous series of openings in the brick walls, the

bricks being arranged in a latticelike manner at distances of 3.937 ins. from each other. Such an arrangement suffices to keep out snow and rain, while affording a maximum of air circulation for cooling purposes.

At Kratzwieck two units are located side by side, separated from each other by an aisle about 4 ft. wide, and under a single roof. In other forms of construction there is a separate roof over each unit.

Great ingenuity is displayed in the manner of suspending the chambers from the skeleton framework. At intervals of 31.5 ins. along the top edges (27.5 ins. on the end sides) hooks of half-inch iron are attached firmly to the lead sheets. These hang by iron rods (1.57 by 0.24 ins.) from the T girders overhead, and thus insure a complete, unhindered exposure of the side walls to the cooling action of the air. There is, however, a further addition necessary, in order to obviate any danger from expansion or contraction. Along the walls, likewise at distances of 31.5 ins. (or 27.5 ins. on the end walls) vertical flaps are soldered on tightly. Light iron rods, round or flat, pass through these, and are connected above with the pendant supports, while below they are attached, by means of spring hooks, to the upturned edge of the chamber floor. They give a ribbed appearance to the chamber walls and increase the cooling surface. While imparting a certain amount of rigidity, and preventing undue swelling outward, or inward depression, they still permit of free play for ordinary contraction, or the reverse, without any undue strain falling upon the soldered fastening of the ribs. The effectiveness of this rib construction is heightened by numerous horizontal connections, by means of iron rods, with the skeleton frameworks.

A similar principle is applied in the case of the top of a chamber. Here there is also a ribbed construction, and hooks at frequent intervals are connected by round iron bars with the girders overhead. All exposed iron parts are carefully protected from corrosion by a special paint.

In the shape of the chamber top we encounter one of the most striking innovations of the Moritz system. It is of a semicylindrical form to prevent the accumulation of dust, while avoiding "dead corners" inside the chamber, and their liability to frequent repair, and assuring a more perfect circulation of air about its upper surface.

The lead floor of the chamber rests upon an under floor of iron plate 0.197 in. in thickness, and this in turn is supported by brick columns about 8 ft. high. In some instances the constructors have used wood or ferroconcrete. In all cases, however, the upturned edge of the floor consists of iron plate, bent underneath to a distance of 9.8 ins. This is to insure the most rapid cooling at the weakest point in the construction of a chamber, where corrosion is most pronounced, and repairs involve a complete stoppage of manufacture. An identical method is used in supporting the lead mantels of the Glover and Gay-Lussac towers. Details

are naturally much simpler, on account of the smaller size of the towers. Each tower shows eight vertical ribs.

There are certain theoretical considerations connected with the use of the rounded roof that deserve mention. In addition to the advantages attendant upon this particular shape, the inventor claims that it is a powerful factor in equalizing temperature, and consequently in insuring uniform chemical activity and production in all parts of a chamber. While there is little movement of air over a flat roof, there is, on the contrary, an active current over the surface of a curved roof, thus bringing about a more rapid cooling. In a chamber of rectangular construction the hotter gases accumulate at the top. With a curved roof the volume of such gases is reduced to a minimum. They are constantly drawn off by the gyratory movement of the chamber's contents, due to external cooling, and descend steadily downward along the walls. An instance is cited from the observations made at Wasquetal upon the temperatures at three different points in the first chamber of a plant. Under similar conditions of external temperature, with the same pyrites, the same ovens, and the same towers, the temperatures observed at the three points in a rectangular chamber were 90°, 88° and 86° C. (194°, 190.4° and 186.8° F.). In the chamber with curved top, which replaced the former, they were 75°, 75° and 74° C. (167°, 167° and 165.2° F.). The more rapid cooling and the close approach to uniformity are noteworthy. The director of the works at Kratzwieck is, however, of the opinion that a more gently curved roof is quite as effective as one of semicircular shape and more desirable from several standpoints. In the plant now under construction he plans to modify this feature.

At the outset it must be noted that the cost of construction, per unit of cubic chamber space, is higher than by the old method. At Kratzwieck the additional cost per cubic meter was found to be about 33 per cent. This figure would vary according to local factors. Where timber is expensive the ratio would be less. Where timber is cheap and iron costly it would naturally increase.

There seems to be no doubt of the manifest economy in space, frequently an important factor where chemical works are hampered in their development by lack of sufficient available area for building. At Kratzwieck the two units already built on the Moritz system occupy but two-thirds of the area formerly covered by three separate units before the fire. Yet these two units produce more sulphuric acid than the three units that were destroyed.

The next important economy is in the amount of nitric acid required to produce the maximum effect in the lead chambers. The newly erected plant of the "Union" yields daily about 7 kilograms (15.432 lbs.) of acid (50° Baumé) per cubic meter (35.314 cu. ft.) of chamber space. Formerly, at this rate of production, 0.7 to 0.8 kilogram of nitric acid (36° Baumé)

was consumed for each 100 kilograms of chamber acid of the above strength. This consumption has now fallen to very nearly 0.5 kilogram. It is to be noted that 0.9 kilogram is a very common figure in German acid works. In 1900 the average consumption was 1.4 kilograms in American acid works burning pyrites. This loss of nitric acid is an important item in the cost of producing sulphuric acid. In Germany it constitutes 4 to 8 per cent of the total cost. It is obvious that a saving of 1 or 2 per cent in this direction constitutes a notable economy. The inventor guarantees a maximum loss of 0.6 kilogram where chambers are producing 7 kilograms daily per cubic meter. It may be mentioned in this connection that in the "Union" system water in the form of spray is admitted to the chambers in summer and in the form of steam during the winter.

Apart from the direct saving on the daily cost of production, the director of the "Union" works is confident from his study of the plant during the few months that it has been in operation that there will be ultimately a very material saving in the item of depreciation, through the prolonged life of the chambers, and the ease and simplicity with which any necessary eventual repairs can be executed. When the time arrives to replace a lead chamber there will no longer be need of a new timber framework, as is ordinarily the rule at present, and the change can be effected in a very short time. At the most a few iron rods or bands used for suspending the chambers may require renewal.

In 1900 Sweden consumed 253,132 metric tons of artificial fertilizers, of which one-half were produced at home. In the course of a decade the industry has developed so rapidly that there is now a surplus of superphosphate available for export, amounting to 17,000 tons in 1909.

Patents have been taken out in Mexico, the United States and Cuba for an invention for utilizing the residue of the maguey, after the fiber has been taken, for the manufacture of alcohol. In the process of extracting the fiber the "flesh" of the leaf is scraped off by machinery and this, with the exception of a small portion used in the manufacture of packing paper, has been heretofore simply thrown on the rubbish heap. The inventor, after years of experiment, is stated to have found that this waste, together with the juice which escapes during extraction of the fiber, will produce a good, merchantable alcohol, as high as 40° Cartier. The raw material is placed in tanks with water and allowed to ferment for two days, after which it passes into a specially arranged still. Approximately, the waste of 1,000 leaves gives 400 liters (liter equals 1.05 quarts) of juice, which in turn produces about 80 liters of alcohol of 40°. The improvements to the ordinary still for correcting the taste and color have been protected by patents.

DEFLOCCULATION.

By EDWARD G. ACHESON, Sc. D.

It is with much diffidence I come before you to speak on a subject that has not yet emerged from the embryonic state. My latest experimental researches had to do with it; I believe it will rapidly grow in importance in the scientific and industrial world; and finally, much work of a strictly scientific character remains to be done to reduce the fragmentary knowledge we now have of it to an exact science.

In my labors devoted to working out and developing industrial and commercial projects, I have upon several occasions found reactions, conditions and results that did not harmonize with the accepted theories and formulae of scientific men. Being an earnest believer in publicity, in order that any possible benefits that might accrue to the common welfare may the more quickly be enjoyed, I shall lay before you a detailed account of my experiments on the deflocculation of inorganic bodies. It will become very evident as my story unfolds, that throughout the series of experiments described and the working hypothesis employed, I was wholly disregarding the prevailing theories, and that I unconsciously entered the field of colloids.

Having worked out the problems involved in the manufacture of graphite from coal and other carbonaceous materials, I undertook, in the summer of 1901, the introduction of this artificially made graphite into the crucible trade. My first efforts were devoted to the making of a satisfactory crucible of my graphite, using as a binding material American clays. Many failures were met with, and I found it difficult to locate the cause of the failure, whether with the graphite or with the clay. I soon learned the manufacturers of crucibles in the United States invariably used as a binding agent for the graphite in the crucible body clays imported from Europe. I secured samples of these imported clays, and found them much superior to the American ones in plasticity and tensile strength.

Chemical analysis failed to disclose the cause of the physical differences existing in the clays. The question involved interested me greatly, and I decided to endeavor to determine what produced the variations. I found it generally stated in the books that residual clays were non-plastic, and sedimentary clays were more or less plastic. Here was the starting point. Plasticity was developed by or during the act of transportation from the point of formation to the final resting place of the clay. I did not believe there was anything in the simple act of the suspension in water that would produce the effect noted, and therefore looked for the cause of the foreign matter carried by the water. It seemed the most likely agents were the organic substances washed from the forests into the running waters. With this idea in view, I made a few experiments with those substances I thought likely to be found in the washings of vegetation. One of my early experiments was to treat kaolin with a solution of tannin, and I at once noticed less water was needed to

produce a given degree of fluidity; also that the tensile strength and plasticity were increased.

Tests were made on the increased tensile strength of clay, as the result of treatment with organic matter, and it was found that briquettes made of Harris kaolin and dried at 120° C. would break with a load of 5.73 kilograms per square centimeter, while the same clay, after treatment with 2 per cent of catechu for a period of ten days, then formed into briquettes and dried at 120° C., would not break until the load was increased to 19.75 kilograms per square centimeter—an increase of more than 244 per cent.

I now began to wonder whether or not the effect I had discovered was known, as it might have much value to an industry of such colossal dimensions and antiquity as clay working. Moreover, it would be amazing if it should not be known, in view of the tremendous amount of experimental work that had been done on that art. I searched for some record of the addition of organic matter to clay during its working, and only one instance could I find, that of the Egyptians in brick-making, as recorded in the fifth chapter of Exodus. The accepted theory of using straw fiber as a mechanical binding agent had never appealed to me. Straw, however, contains no tannin, and the effect I had found had always been produced with tannin, or a substance containing tannin. I procured some straw, boiled it with water, decanted the resultant reddish brown liquid, and mixed it with clay. The result was like that produced with tannin, and equal to the best I had obtained. It now seemed likely that the Egyptians were familiar with the effect I had discovered, and believing this was why they used straw in making brick and were successful in substituting stubble for the straw, I called clay so treated "Egyptianized clay."

The effect of organic matter, as typified by tannin, in producing deflocculation and a resultant colloidal state of clay, is very readily shown; for instance, I have here some powdered kaolin, a small quantity of which I will place in a test tube, add water, and after shaking, set aside. Another portion of the kaolin I will put into a beaker, and moisten with a water solution of tannin, to which a small amount of ammonia has been added. After a thorough mixture has been made, using a glass rod to stir with, to eliminate as much as possible any grinding action, I will add more water and divide the contents of the beaker between two test tubes. To one of the tubes containing tannin-treated clay I will add a little common table salt. The three tubes I will place here before you, that we may examine them later.

In the summer of 1906 I succeeded in making artificially a high-grade graphite which I wished to make applicable to all kinds of lubrication. To meet the various demands, it would be necessary to have it remain diffused in liquids lighter than itself; for instance, water and petroleum oils. Recalling the effect of tannin on clay, which caused it to remain diffused in wa-

ter, I treated a sample of my graphite with tannin, and found the same effect occurred. The graphite being black, it makes a better lecture demonstration than the clay I have shown you, and I will repeat my experiments, using graphite. I have here a sample of artificially made graphite, which has been disintegrated to a fineness that will permit it to pass through a sieve having 40,000 meshes to the square inch. I will introduce a small quantity of it into a test tube, add water, and after shaking, set aside. Another sample I will place in a beaker and moisten with a solution of organic matter, and after thoroughly stirring with a glass rod, I will add water and divide it between two test tubes, to one of which I will add table salt. These three tubes I will place beside those holding the clay, to be examined later.

The actual amount of deflocculating effect produced on the graphite in the beaker is very small indeed. In commercial work considerable mastication and time are required. I have here a bottle containing water having two to three tenths of 1 per cent of its weight in deflocculated graphite, the deflocculation having been produced by a treatment similar to that I have applied to the graphite in the beaker, and a little of it being poured on a filter, you see the black liquid running into the test tube below the filter. The paper retained none of the graphite on its upper surface, all of it having passed into and through the paper. I will now add two or three drops of acid to the black liquid in the tube, and after warming over a spirit lamp, will throw it on another filter paper, and you now see a clear, colorless liquid descending into the tube below the filter. This is the water in which the graphite in a deflocculated condition was diffused; the graphite having been deflocculated by the acid is now retained on the upper surface of the filter paper. The effect I have produced with the acid could have been produced with a solution of salt, lime water, or any one of that large list of substances known as electrolytes, even so weak an acid as carbonic acid if caused to bubble up through water carrying deflocculated graphite will cause flocculation and sedimentation.

Upon being deflocculated, the graphite is diffused through the water in a colloidal state, and I have samples of deflocculated graphite in water which have stood for more than two years without showing any settling, notwithstanding the graphite is two and two tenths times heavier than water.

I have been able to obtain this same effect on natural graphite, amorphous alumina and silica, lamp black, and my manufactured product—Siloxicon, which is an amorphous body having the formula $\text{Si}_2\text{C}_2\text{O}$. The effect can be produced with a long list of organic bodies; for instance, tannin or organic substances containing tannin, also with solutions containing the gum of the peach and cherry tree, or extracts from straw and grass. The offal from the barnyard proved to be very efficient. I speak of these organic substances as agents when used to produce defloccula-

tion, and they act as protective colloids to the deflocculated body.

Some minutes have now passed by, and we will examine the tubes containing the clay and the graphite. We find the clay that had been mixed with plain water has settled. The mixture of clay, water, organic matter and salt has also cleared, while the tube containing the clay, water and organic matter remains muddy. In like manner the tube containing the disintegrated graphite in water has cleared; the second one containing water, graphite and organic matter remains black; while the third tube, which was set up the same as the second, but had a little salt added to it, has cleared. Apparently a great affinity existed between the organic and the inorganic substances introduced into the water. The inorganic body abstracted the organic from the water, and in doing so, was deflocculated. Each particle as it was thrown off was enveloped in an aqueous jelly of the organic agent, or at least such was the working hypothesis I followed to arrive at my results, and I find it difficult to think of this breaking up stopping short of the final subdivision with the resultant separation into individual molecules, or the smallest particles into which a body can be subdivided without loss of identity.

As I have already stated, I deflocculated clay in the year 1901 and graphite in 1906 and immediately afterwards a number of other bodies. I early understood I was producing colloidal conditions of these bodies, but not until the summer of the present year, 1911, did I read any treatise on colloids, being familiar with this state of matter only in a very general way. During the summer I procured a copy of the book "Colloids and the Ultramicroscope," as written by Dr. Richard Zsigmondy and translated into English by Jerome Alexander, of New York. I found the book extremely interesting, and at once wished to have a sample of my deflocculated graphite subjected to ultramicroscopic examination. Mr. Alexander kindly undertook the examination. He found the graphite in the deflocculated condition to be in a true colloidal state, the particles being in rapid motion, and he estimated their average size in linear dimensions to be 75 millimicrons. Seventy-five millimicrons are seventy-five millionths of a millimeter, and it would require slightly more than 13,000 of the particles to extend one millimeter. Now, the particles of disintegrated graphite, used as the crude material from which to produce deflocculated graphite, pass, as I have stated, through a sieve having 40,000 meshes to the square inch and their maximum linear dimensions such that it would require 13 of them to extend one millimeter. Hence, the particle of disintegrated graphite is one thousand times greater in linear dimension than the deflocculated one. These are figures that certainly test our powers of appreciation.

I have been asked, "Why don't you speak of the graphite as colloidal?" Knowing now that it is in a colloidal state, I speak of it as being colloidal, but when

I am speaking of my process I am talking of a method of producing deflocculation. When does that process of deflocculation stop? Is it short of the final subdivision and the throwing off of the molecule? I think not. I believe we are here dealing with molecules. Their size may not agree with what they should be as computed in accordance with accepted theories, but, nevertheless, I cannot conceive the subdivision once started, in the presence of sufficient deflocculating agent, will stop short of the final, with the freeing of the molecule and the creation of the colloidal state.

How did all this I have been telling you come to happen? The following quotation aptly tells how:

"It's generally the fellow who doesn't know any better who does the thing that can't be done. You see, the blamed fool doesn't know it can't be done, so he goes ahead and does it."

An International Hygiene and Maritime Exposition will be held in Genoa, Italy, from October, 1912, to July, 1913. The exposition is intended to illustrate progress made in matters relating to hygiene and maritime affairs in general. The program established by the committee comprises, in the section devoted to hygiene, exhibits covering developments made in practically all branches of the science. The maritime exhibits are to be very broad in scope, covering practically every department of navigation and maritime affairs. Besides the exhibits shown, numerous lectures will be given to explain them by persons notably well qualified to speak on the subjects assigned them, thus assisting the exhibitors in a more widespread diffusion of popular knowledge regarding these important matters. Detailed information as to the exposition may be secured by addressing the Comitato Esecutivo della Esposizione Internazionale di Marina e di Igiene, Piazza Corvetto No. 1, Genoa.

The production of iron ore in south Russia centers at Krivoi-Rog and Kertch, the first named predominating in importance. In 1909, owing to the mines having put out rather more than market requirements, there accumulated quite a quantity at the pits' mouths. On January 1, 1910, this accumulation of ore at the Krivoi-Kertch mines was 595,800 and 108,000 tons, which decreased to 563,400 and 81,000 tons, respectively, on July 1, 1910. Increasing demands further decreased these accumulations to 450,000 and 45,000 tons on January 1, 1911, and to 363,000 and 7,200 tons by July 1 this year. The present strong demand is indicated by the output for the first half of 1911—2,466,000 tons, against 2,136,600 tons for the same period in 1910 at Krivoi-Rog, and at Kertch 138,400, against 160,200 tons. The demand was principally for ores of no less than 62 per cent iron. The price of ore this year averages from 71½ kopecks (kopeck equals ½ cent) to 8 kopecks per pood (pood equals 36 lbs.) at the pit's mouth, this being 1½ to 1¾ kopecks per pood more than last year.



A NOTE ON THE ASSAY OF FORMALDEHYDE.*

By ELIAS ELVOVE.†

It has been shown by Williams¹ that the two different types of methods for estimating formaldehyde, namely, those dependent on the oxidation of the formaldehyde and those dependent on its forming condensation products, do not yield concordant results. In explanation of this discrepancy, Williams advances the view "that either the condensation reactions are not complete or a small part of the formic acid is oxidized farther." He also reaches the conclusion that the hydrogen peroxide method is the most satisfactory for strong, impure solutions; while "the potassium cyanide method is recommended for dilute, impure solutions." Recently, the writer has had occasion, in connection with the study of embalming fluids² which has been in progress in this laboratory, to make a comparatively large number of determinations of the quantity of free formaldehyde in these fluids. Owing to the possible presence in such fluids of substances other than formaldehyde which on oxidation will yield acids (thus rendering the hydrogen peroxide method inapplicable), or other reducing substances (thus invalidating the iodometric method), it was found necessary to resort to one of the condensation methods, and the potassium cyanide method was chosen. It became of interest, therefore, to look into the points raised by Williams as to the discrepancy of the results obtained by the oxidation and condensation methods and the possible incompleteness of the condensation reactions.

The methods investigated by Williams were those which are most generally recommended for the determination of formaldehyde, namely, the ammonia method of Legler,³ the hydrogen peroxide method of Blank and Finkenbeiner,⁴ and the iodometric and potassium cyanide methods of Romijn.⁵ His conclusion that the results obtained by the condensation methods are lower than those obtained by the oxidation methods, is in harmony with the previous work of Smith,⁶ who also found that, as compared with the Legler method, "the hydrogen peroxide method almost invariably gave higher results." Smith also seems to hold the view that the potassium cyanide method is applicable to dilute solutions only, since he refers to the

KCN method as a "method which is applicable to solutions containing but small quantities of formaldehyde"; and while he concludes that "the iodometric and potassium cyanide methods give good results on dilute solutions," he adds that "it should be remembered that in diluting strong solutions to the range of these methods, a small error in weighing may be considerably multiplied." Apparently influenced by this suggestion of Smith, many authors on analytical chemistry do not recommend the potassium cyanide method except in the case of dilute solutions. Thus Leffmann and Beam,⁷ referring to the work of Smith, state that for the determination of formaldehyde the choice of the method "will depend on the strength of the solution," recommending the iodine method of Romijn in the case of moderately strong solutions and the potassium cyanide method for dilute solutions. Likewise, from Schimpf,⁸ who also refers to the work of Smith, one gains the information that "the Blank and Finkenbeiner method is very satisfactory for strong solutions" and "the iodometric and potassium cyanide methods give good results on dilute solutions." Similarly, Leach⁹ omits the potassium cyanide method as a suitable method for determining formaldehyde in the commercial preservative, although listing the iodometric method,¹⁰ the method of Blank and Finkenbeiner and the ammonia method; while the potassium cyanide method he recommends only in the case of very dilute solutions such as are met with when determining formaldehyde in milk.¹¹ The impression, therefore, which one obtains from the literature on the subject as has been referred to, is that while the potassium cyanide method is admittedly a very good method for determining formaldehyde in dilute solutions it is not recommended in the case of the concentrated or moderately strong solutions, and that in such cases it is preferable to use one of the other methods, of which the hydrogen peroxide method is the most favored. An examination of the leading pharmacopœias as to the methods chosen for assaying commercial formaldehyde solutions also seems to confirm the conclusion that the potassium cyanide method is not generally regarded as a method suitable for use in such cases. Thus the U. S. Pharmacopœia (1905) adopts the hydrogen peroxide method, the German Pharmacopœia (1910) gives the acidimetric sodium sulphite method, the

*Ber., 16, 1333-1337 (1883).

†Ber., 31, 2979-2981 (1898).

²Zelt. anal. Chem., 36, 18-24 (1897).

³Jour. Amer. Chem. Soc., 25, 1028-1035 (1903).

⁴Leffmann and Beam: Food Analysis, 2d ed., p. 84 (1905).

⁵Schimpf: Manual of Volumetric Analysis, 5th ed., pp. 644-645 (1909).

⁶Leach: Food Inspection and Analysis, 2d ed., p. 819 (1907).

*From the American Journal of Pharmacy.

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¹Jour. Amer. Chem. Soc., 27, 596-601 (1905).

²It is expected that the results of this study will shortly be published as a Hygienic Laboratory Bulletin.

British Pharmaceutical Codex (1907) recommends the ammonium chloride and iodometric methods, while the French Pharmacopœia (1908) agrees with the U. S. Pharmacopœia in giving preference to the hydrogen peroxide method. Finally, we may mention that according to Fresenius and Grünhut¹² there really are only four methods in use for assaying commercial formaldehyde, namely, the ammonia method of Legler, the method in which the formaldehyde is treated with sodium hydroxide in pressure bottles, the hydrogen peroxide method of Blank and Finkenbeiner, and the iodometric method of Romijn.

On the other hand, the meaning of the objection advanced by Smith against the use of the potassium cyanide method for determining formaldehyde in strong solutions, namely, "that in diluting strong solutions to the range of these methods, a small error in weighing may be considerably multiplied," is difficult to understand. For supposing that 1 gm. of the formaldehyde solution is weighed out, diluted with distilled water to 100 c.c. and 25 c.c. of the resulting solution taken for the analysis, and supposing that the delicacy of the balance used is only ± 0.001 gm., then the actual amount taken for the analysis would be $0.250 \text{ gm.} \mp 0.00025 \text{ gm.}$, involving therefore a possible error of ± 0.1 per cent. Now if the step of diluting and taking for the analysis an aliquot portion were omitted and the total amount weighed out were used for the analysis, the amount so taken would be $1. \pm 0.001 \text{ gm.}$, the possible error involved would therefore be exactly the same as before, namely, ± 0.1 per cent. It is possible that "error in weighing" is an oversight, the intention having been—error in *measuring*. If this was the intention, we can readily see how an error might be introduced through an error in measuring. For supposing that instead of using for the analysis the entire amount weighed out, it be diluted to a definite volume and 5 c.c. of this solution taken for the analysis. In measuring out the 5 c.c. there might be an error, say of ± 0.05 c.c. This would introduce an error of ∓ 1 per cent, which would have been avoided if the analysis had been carried out directly on the weighed amount of formaldehyde solution. But even in this case the error would not be "multiplied," unless more than one dilution be made. Moreover, if instead of using only 5 c.c. of the diluted solution a larger volume be taken, say 25 c.c., an equal absolute error in the measuring

would, of course, relatively be only one-fifth, or reducing the possible error from ± 1 to ± 0.2 per cent. In other words, a procedure which would involve the dilution of a certain weight of the strong formaldehyde solution to 100 c.c. and the use of 25 c.c. of the diluted solution for the analysis, would ordinarily involve no greater error than the inherent errors of volumetric analysis. And since all the methods compared are volumetric, any such errors may be considered equally possible in all of them. For supposing that in measuring out the 25 c.c. of the diluted formaldehyde solution there might be an error of ± 0.05 c.c. or ± 0.2 per cent, might there not also be a similar error when the same measuring instrument will be used for measuring out the 25 c.c. of twice normal NaOH (according to Smith) in the case of the hydrogen peroxide method? Besides, in taking 25 c.c. of the 100 c.c. of the diluted formaldehyde solution it is not necessary that the measuring instrument used should measure out exactly 25 c.c. but only that the volume so measured out be one-fourth of the total, and whether this is the case can be readily and conveniently ascertained by using the same measuring instrument for filling the empty 100 c.c. flask with water and seeing whether four fillings will fill it exactly to the mark. This will prove the accuracy of the instrument for this purpose or enable the operator to apply the proper correction. It would seem therefore that if the potassium cyanide method is an excellent method for determining formaldehyde in dilute solutions it may be used with equal advantage in the case of strong solutions by suitably diluting the latter with water.

That such is really the case may be seen from Smith's own results. Using the potassium cyanide method for strong formaldehyde solutions, in which case "all solutions containing more than one per cent were diluted," the three results given in the case of the 37 per cent formaldehyde solution are: 37.12, 37.07, and 37.18 per cent. Likewise, the results of Williams, when using the potassium cyanide method for determining the formaldehyde in the concentrated solution, show that very concordant results are obtained by this method. Thus six determinations yielded the following percentages: 35.15, 35.06, 35.23, 35.01, 35.21, 35.07, or a maximum difference from the average (35.12) of only 0.11 per cent. When using the hydrogen peroxide method, four determinations yielded the following percentages: 35.82, 35.92, 35.74 and 35.78, or a maximum difference from the average (35.81) of 0.11 per cent. It is thus seen that the potassium cyanide method can be applied to strong formaldehyde solutions by suitably diluting them, the results thus obtained being equally concordant as those obtained by the hydrogen peroxide method. It therefore remains only to clear up the point as to whether or not the lower results obtained by the potassium cyanide method as compared with those obtained by the hydrogen peroxide method may be due to possible

¹⁰Ibid. It may be noted in this connection that both here and in the first edition (p. 665) there seems to be an error in some of the figures given. For if we take 10 c.c. of a 3 per cent formaldehyde solution (it is directed that the solution is to be diluted "to a strength not exceeding 3 per cent") we will be operating on about 0.3 Gm.; and since "two atoms of iodine are equivalent to one molecule of formaldehyde," 0.3 Gm. HCHO will require 200 c.c. of $N/10$ iodine, whereas according to the given directions only 25 c.c. of $N/10$ iodine are to be taken. The same error is found also in Leffmann and Beam: Food Analysis, 2nd ed., p. 110.

¹¹Ibid., p. 181. There seems to be also an oversight in this connection, since it is directed to use ferric chloride as the indicator in titrating the excess of silver.

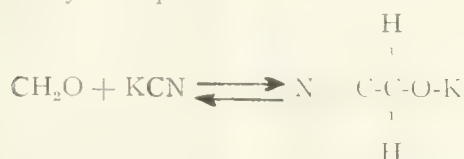
¹²Zeit. anal. Chem., 44, p. 13 (1905): "Doch sind es—soweit unsere Kenntnis reicht—im wesentlichen nur vier Verfahren, deren man sich in der Technik zur Betriebskontrolle, sowie zur Wertbestimmung des Handelsproduktes, bedient: die Ammoniak-Methode von Legler, die Oxydation mit Natronlauge in Druckflaschen, die Wasserstoffsperoxyd-Methode von Blank und Finkenbeiner, und endlich die jodometrische Methode von Romijn."

incompleteness of the reaction between the KCN and the formaldehyde.

If we will suppose that the reaction between the KCN and the formaldehyde is not complete, we must assume that the value of $c_1 p_1 q_1$ in the general question

$$V \frac{c_1}{c_1 + c_2} = c_1 p_1 q_1 - c_1 p_1 q_1,$$

which is the fundamental equation underlying all chemical dynamics,¹³ is not negligible but has a value which would account for the lower results obtained by the KCN method. That is, in the reaction between the formaldehyde and the KCN, which may be represented by the equation



we must assume that the velocity of the reaction which tends to decompose the condensation product back into formaldehyde and KCN has an appreciable value. From the law of mass action it would follow, therefore, that if we were to vary the mass of one of the reacting substances the point at which equilibrium is established should also vary; just as in the classical instance of an incomplete reaction, namely, the reaction between acetic acid and ethyl alcohol, in which case Berthelot and Pean de Saint Gilles¹⁴ found that by varying the amount of alcohol with respect to that of the acid the amount of ester produced also varied, being only 66.5 per cent of the theory when one equivalent of alcohol was used but increasing to 82.8 per cent when two equivalents of the alcohol were added. In order, therefore, to determine whether any such variation occurs in the case of the reaction between formaldehyde and KCN, the following experiments were carried out on mixtures of formaldehyde and KCN, in which the amount of KCN added varied from approximately one to three equivalents.

MODE OF PROCEDURE.

A formaldehyde solution was prepared by diluting 21.1879 gms. of a sample of U.S.P. solution of formaldehyde to 2000 c.c. with distilled water. Portions of 15 c.c. of this solution were mixed at room-temperature (22-25° C.) with varying amounts of approximately N/10 KCN, the minimum amount of which was just a little in excess of that theoretically required to combine with all of the formaldehyde taken, while the maximum amount of KCN used was about two equivalents more than necessary for a complete equimolecular union. The KCN solution was prepared from a sample of Kahlbaum's potassium cyanide, and its value in terms of tenth-normal silver nitrate was determined simultaneously with each series of experiments, the determination being carried out in a manner similar to that used for determining the uncombined cyanide in the experiments. The mixing of the formaldehyde and potassium cyanide solutions

was generally effected in Erlenmeyer flasks; and immediately after mixing (the latter operation consuming about half a minute), the resulting solution was added to a mixture of a known amount of silver nitrate and nitric acid in a 200 c.c. measuring flask. The amount of silver nitrate used was sufficient in every case to combine with all of the excess of KCN and calculated to leave in the filtrate, for the subsequent titration, the equivalent of about 2 c.c. of N/10 KCNS per 100 c.c. of the filtrate. The amount of nitric acid used was 10 or 20 c.c. of U.S.P. dilute (10 per cent) nitric acid. After thoroughly washing the Erlenmeyer flask in which the KCN and formaldehyde solutions were mixed and adding the washings to the flask containing the acidified silver nitrate solution, the whole was made up to 200 c.c. with distilled water, thoroughly shaken, filtered through a dry filter, and 100 c.c. of the filtrate titrated for the excess of silver by means of N/10 KCNS, using 2 c.c. of a 10 per cent ferric alum solution as indicator. The results obtained are given in Table I.

TABLE I.—EFFECT OF VARYING THE EXCESS OF KCN.

No. of experiment.	Amount of formaldehyde solution taken (c.c.)	Amount of KCN added (c.c.)	Amount of N/10 AgNO ₃ used (c.c.)	N/10 KCNS required for one-half of filtrate (c.c.)	Found equivalent of the formaldehyde solution in terms of N/10 AgNO ₃ (c.c.)
1	15	19.7	4	1.50	18.45
2	15	20.0	4	1.45	18.65
3	15	21.0	5	1.55	18.84
4	15	22.0	6	1.67	19.07
5	15	23.0	7	1.75	19.21
6	15	24.0	8	1.75	19.20
7	15	25.0	9	1.80	19.29
8	15	27.0	11	1.85	19.36
9	15	30.0	14	1.83	19.29
10	15	40.0	24	1.90	19.30
11	15	50.0	34	1.98	19.34
12	15	60.0	44	2.02	19.29

The results given in Table I show that the maximum result was reached when the excess of KCN was about one-half of an equivalent (exp. 8), while further increasing the amount of KCN up to approximately three equivalents did not appreciably alter the result. The reaction may therefore be regarded as complete, at least for all practical purposes, when the excess of KCN is as much as one-half of an equivalent. On the other hand, when the excess of KCN is less than one-fourth of an equivalent (exps. 1-6), the reaction is incomplete; and when the amount of KCN added is only slightly in excess of that required to combine with all of the formaldehyde present, the result is about 3 or 4 per cent too low (exps. 1-2). That the excess of KCN must be above a certain minimum in order to insure a complete reaction is not brought out either

¹³Jones: Elements of Physical Chemistry, 3rd ed., p. 529 (1907).
¹⁴Ibid., p. 522.

*The KCN solution was only approximately tenth-normal, 48 c.c. being equivalent to 47.4 c.c. N/10 AgNO₃.

in the original paper of Romijn or those of Smith or Williams. In fact, none of the literature examined has any reference to this point; while from the statement of Romijn that the method is based on the property¹⁵ of formaldehyde to *immediately* add itself to potassium cyanide and his unmodified general statement¹⁶ that when the KCN is in excess so much of the KCN becomes combined as to correspond to one molecule of KCN for every molecule of formaldehyde, one might suppose that the relative amount of the excess of KCN to be added does not matter. This conclusion would gain further support from the statement of Smith¹⁷ that the determination is carried out by mixing the formaldehyde "with a known quantity of potassium cyanide, the latter being in excess."

In order to determine the effect of varying the time or the temperature, two series of experiments were carried out. In one of these, the mixtures of formaldehyde and KCN were allowed to stand for different intervals of time at the constant temperature of 15° C. and the residual cyanide then determined in the usual way. In the other, the time was constant but the temperature was made to vary between 5° C. and 40° C. In those cases where the mixtures had to stand for a considerable length of time, the flasks containing same were tightly stoppered by means of glass or rubber stoppers and the value of the KCN solution was controlled by letting it stand under similar conditions and determining its strength at the end of the experiment. The experiments designed to show the effect of temperature were carried out as follows: The formaldehyde solution was placed in one Erlenmeyer flask while the KCN solution which was to be added to it was placed in another similar flask and both were then warmed or cooled to the desired temperature by being set in a large water-bath which was kept about a degree above or below the desired temperature, respectively. The time thus consumed was about half an hour. The KCN solution was then poured into the flask containing the formaldehyde solution and the mixture quickly transferred back again into the flask which contained the KCN solution and mixed in the usual way, while the flask containing the solutions was still immersed in the water-bath. This mixture was then quickly poured into a mixture of silver nitrate¹⁸ and nitric acid, in a 200 c. c. measuring flask, and mixed with it. Both flasks were then thoroughly washed with distilled water, the washings added to the silver solution, and the whole made up to 200 c.c. From this point the procedure was the same as that already described. Each of the different conditions of time and temperature was studied with three different KCN excesses, namely, a KCN excess of approximately one-hundredth, one-fourth, and one-half of an equivalent, respectively. The results obtained are given in Tables II and III.

From the results given in Table II it is seen that when the mixture of KCN and formaldehyde is allowed to stand 42 hours the result obtained is practically the same as when the mixture is immediately added to the silver solution. From the results given in Table III it is seen, however, that a variation in the temperature does have an appreciable effect on the results obtained, especially when the KCN added is only slightly in excess of that required to combine with all of the formaldehyde present. That this is so may also be seen from the following experiments: To each of four Erlenmeyer flasks there were added 25 c.c. of a dilute formaldehyde solution and 33 c.c. of an approximately N/10 KCN solution and the flasks tightly stoppered by means of rubber stoppers. From the known value of the formaldehyde and KCN solutions¹⁹ the amount of KCN added was calculated to be approxi-

TABLE II.—EFFECT OF VARYING THE TIME.
Temperature: 15° C.

No. of experiment.	Time mixture stood before added to silver solution. Hours.	Amount of formaldehyde solution taken. (c.c.)	Amount of N/10 KCN added. ^b (c.c.)	Amount of N/10 AgNO ₃ used. (c.c.)	N/10 KCNS required for $\frac{1}{2}$ of filtrate. (c.c.)	Found equivalent of the formaldehyde solution in terms of N/10 AgNO ₃ . (c.c.)
1	0*	25	33	5	1.80	31.33
2	0*	25	40	12	2.30	32.27
3	0*	25	48	20	2.38	32.36
4	1	25	33	5	1.80	31.33
5	1	25	40	12	2.30	32.27
6	1	25	48	20	2.40	32.40
7	18	25	33	5	1.82	31.37
8	18	25	40	12	2.25	32.17
9	18	25	48	20	2.35	32.30
10	42	25	33	5	1.85	31.43
11	42	25	40	12	2.30	32.27
12	42	25	48	20	2.35	32.30

mately one-hundredth of an equivalent in excess of that required to combine with all of the formaldehyde present. All these flasks were then set in a large water-bath, the temperature of which was kept at about 40° C., and allowed to remain in this bath for half an hour. At the end of this time, the contents of flask No. 1 was added to a mixture of 5 c. c. N/10 AgNO₃ and 20 c. c. of 10 per cent nitric acid in a 200 c. c. measuring flask and the analysis completed is in the former experiments; while flasks Nos. 2, 3, and 4 were taken out of this bath and set in another large water-bath, which was kept at about 5° C. After remaining in this bath for half an hour, No. 2 was treated as No. 1, while Nos. 3 and 4 were taken out and placed again in the bath at 40° C. After this second warming to 40° C., No. 3 was treated as were Nos. 1 and 2, while No. 4 was taken out of this bath and set again in the bath at 5° C. After this second cooling to 5° C., No. 4 was treated as those preceding it. On titrating half of the final filtrate with N/10 KCNS, the following results shown in Table II were obtained:

*Added immediately after mixing.

^b48 c.c. of the KCN solution was equivalent to 47.56 c.c. N/10 AgNO₃.

¹⁹The amount of AgNO₃ varied as shown in the tables, while the amount of HNO₃ was the same in all cases, namely, 20 c.c. of 10 per cent nitric acid.

¹⁵Zeit. anal. Chem., 36, 18 (1897): "die Eigenschaft des Formaldehyds, das Cyankalium sofort zu addiren."

¹⁶Ibid. "wenn ein Ueberschuss von Cyankalium vorhanden war, wird von dem Cyankalium so viel gebunden, dass auf ein Molecul Formaldehyd ein Molecul Cyankalium kommt."

¹⁷Jour. Amer. Chem. Soc., 25, 1032 (1903).

On now repeating these experiments in exactly the same way except that the excess of KCN was increased²⁰ to approximately one-half of an equivalent (using 48 c. c. of N/10 KCN, whose factor was 0.9875), the following results shown in Table III were obtained:

These results, therefore, like the results given in Table III, show that when only a comparatively small excess of the KCN is used, the effect of a considerable variation in the temperature manifests itself quite markedly in the results; but when the excess of KCN is increased to about one-half of an equivalent, the results obtained at 40° C. are only slightly lower than those obtained at 5° C.; while the results given in Table III show that a variation of the temperature between 5° C. and 30° C. had no appreciable effect on the found value of the formaldehyde solution when the excess of KCN was about one-half of an equivalent.

TABLE III.—EFFECT OF VARYING THE TEMPERATURE.
Temperature: 5°C.

No. of experiment.	Amount of formaldehyde solution taken (c.c.)	Amount of N/10 KCN added (c.c.)	Amount of N/10 AgNO ₃ used (c.c.)	N/10 KCNS required for 1/2 of filtrate (c.c.)	Found equivalent of the formaldehyde solution in terms of N/10 AgNO ₃ (c.c.)
1	25	33	5	1.90	31.50
2	25	40	12	2.30	32.23
3	25	48	20	2.38	32.32
Temperature: 15°C.					
4	25	33	5	1.83	31.32
5	25	40	12	2.30	32.18
6	25	48	20	2.40	32.30
Temperature: 30°C.					
7	25	33	5	1.60	30.86
8	25	40	12	2.20	31.98
9	25	48	20	2.40	32.30
Temperature: 40°C.					
10	25	33	5	1.35	30.36
11	25	40	12	2.15	31.88
12	25	48	20	2.30	32.10

lent. We may conclude, therefore, that the ordinary changes in room-temperature during the year will not appreciably affect the results when the excess of KCN is as much as one-half of an equivalent.

A closer examination of the potassium cyanide method seemed to show also that there is nothing inherent in the basic principle of this method which would prevent its application directly to the concentrated formaldehyde solutions. That such is really the case may be seen from the following experiment: To 0.5505 Gm. of a sample of U.S.P. formaldehyde, weighed out in a closely fitting glass-stoppered Erlenmeyer flask of about 150 c.c. capacity, there were added 100 c. c. of an approximately N/10 KCN solution²¹ and the two solutions well mixed. This mixture was then added to a mixture of 35 c.c. N/10 AgNO₃ and 20 c.c. 10 per cent nitric acid in a 200 c.c. measuring flask, the Erlenmeyer well washed and

the washings added to the silver solution, the whole made up to 200 c.c., thoroughly shaken, and filtered through a dry filter. 100 c.c. of this filtrate were found to require 2.00 c.c. N/10 KCNS. This would make the formaldehyde in the 0.5505 Gm. of the sample equivalent to 67.75 c.c. N/10 AgNO₃ which would correspond to 0.20325 Gm. HCHO or 36.92 per cent. On analyzing a diluted solution of this sample, the result obtained corresponded to 36.98 per cent. It is thus seen that not only is the potassium cyanide method applicable to concentrated formaldehyde solutions by previously diluting them with water but that it may even be applied to the concentrated solution directly, just as in the case of the hydrogen peroxide method.

It has already been pointed out above that in the case of the concentrated formaldehyde solution the H₂O₂ method seems to be most generally favored and that it is the official method of the present U. S. Pharmacopœia. We may therefore ask, What advantages has the H₂O₂ method over other methods, such as the iodometric or Legler method, which entitle it to this preference? The answer to this may be found partly in the results of Williams²² who found that in the presence of small amounts of ethyl alcohol, paraldehyde, methyl alcohol, or acetone, the H₂O₂ method gave normal results whereas the results obtained by the iodometric method were abnormal. Similarly, in the case of a formaldehyde solution which contained a small amount of acetaldehyde, the result obtained by the Legler method was 47.8 per cent instead of 34.87 per cent. in the absence of the acetaldehyde, thus increasing the result in percentage by 12.93; whereas by the H₂O₂ method, under similar conditions, the increase in the percentage was only 2.78 (from 35.82 to 38.6). The results of Williams, however, while showing that the H₂O₂ method is more advantageous than the iodometric or Legler methods, also show that the KCN method is even more reliable than the H₂O₂ method. For not only were the results by the KCN method normal in all cases where the H₂O₂ method gave normal results but even in the presence of acetaldehyde which, as already noted, increased the results by the H₂O₂ method from 35.82 to 38.6 per cent., the result obtained by the KCN method was only slightly higher than in the absence of the acetaldehyde, being 35.12 per cent. in the absence of the acetaldehyde and 35.3 per cent. in its presence; while according to Romijn²³ very exact formaldehyde determinations may be obtained by the KCN method even in the presence of formaldehyde, acetone or benzaldehyde. We thus see that not only has the KCN method the advantage over the H₂O₂ method in that the reaction on which the former method is based is characteristic for aldehyde but that it may even be used for distinguishing and determining formaldehyde in the presence of certain other aldehydes. The fact also that in the H₂O₂

²¹Jour. Amer. Chem. Soc., 600 (1905).

²⁰48 c.c. of the KCN solution was equivalent to 47.56 c.c. N/10 AgNO₃ in expts. 1-3; to 47.5 c.c. N/10 AgNO₃ in expts. 4-12.

²²Zeit. anal. Chem., 23, 22 (1897): "Die Cyankaliummethode gestattet also auch bei Anwesenheit von Acetaldehyd, beziehungsweise Aceton oder Benzaldehyd, eine sehr genaue Bestimmung des Formaldehyds."

method the results will be influenced by any substance which on oxidation, under the conditions of the method, will produce acid products capable of consuming a portion of the alkali present, has led Fresenius and Grünhut²⁴ to recommend that in assaying commercial formaldehyde by the H_2O_2 method the results obtained should be confirmed by the iodometric method, since it is only when both methods give closely agreeing results that the mean of the two may be taken to represent the actual amount of formaldehyde in the solution.

But these are not the only disadvantages of the H_2O_2 method. For according to Smith,²⁵ the "working range" of the H_2O_2 method, on the lower end, ends with solutions containing about 5 per cent HCHO ; so that when we have occasion to carry out comparative experiments requiring a knowledge of the formaldehyde content of the strong solutions and also of solutions which contain considerably less than about 5 per cent. formaldehyde, if we would use the H_2O_2 method we would be obliged to use two different methods in the same work. Therefore, should we not prefer a method (the KCN method) which can be universally applied to formaldehyde solutions, whether these be strong or weak with reference to their formaldehyde content, and which is based on a reaction that is characteristic of aldehyde and also permits of distinguishing and estimating formaldehyde in the presence of certain other aldehydes? Finally, we may add that the KCN method has also the advantage of being based ultimately on the beautiful and exact Volhard thiocyanate method and that the silver nitrate solution required comes nearest to the chosen ultimate standard for volumetric solutions—pure metallic silver.²⁶

In connection with the study of embalming fluids to which has already been made above, it was desired also to obtain an idea as to degree of variation in the strength of the commercial formaldehyde on the American market. Accordingly, samples were obtained from the principal American firms who sell this article. And inasmuch as, for the reasons given above, the KCN method may be considered as more reliable for determining the actual formaldehyde content of such solutions, the determinations were carried out by this method. Seven samples were examined. The results obtained are given in Table IV.

The results given in Table IV show that the majority of the samples examined contained slightly less

formaldehyde than is required by the present U.S.P. ("not less than 37 per cent by weight"), and that only two (Nos. 6 and 7) of the seven samples examined may be said to have come up to its requirement. Similar results have also been obtained by Evans Sons Lescher and Webb,²⁷ who found that the nine samples of commercial formaldehyde solution which they examined ranged from 37 down to 35.4 per cent of absolute formaldehyde by weight. Likewise, Bachman²⁸ reports solution of formaldehyde ranging from 36.6 per cent to 31.8 per cent, instead of the minimum of 37 per cent as required by the present U.S.P. Finally, we may mention in this connection that the formaldehyde requirement of the French Pharma-

TABLE IV.—SHOWING THE HCHO STRENGTH OF VARIOUS SAMPLES OF COMMERCIAL FORMALDEHYDE.

No. of sample.	Amount of formaldehyde solution taken, (c.c.)	Amount of N/10 KCN added ^a , (c.c.)	Amount of N/10 AgNO_3 used, (c.c.)	N/10 KCNs required for one-half of filtrate, (c.c.)	Found equivalent of the formaldehyde solution in terms of N/10 AgNO_3 , (c.c.)	Amount of HCHO expressed in percentage by weight.
1	25	48	20	1.75	31.10	35.47
2	25	48	20	2.05	31.71	36.11
3	25	48	20	2.00	31.64	36.21
4	25	48	20	2.30	32.25	36.53
5	25	48	20	2.35	32.34	36.61
6	25	48	20	2.40	32.44	36.98
7	25	48	20	2.91	33.46	37.97

^aThese solutions of formaldehyde contained the following amounts in grams of the respective samples of the concentrated formaldehyde in 2,000 c.c.: No. 1, 21.0418; No. 2, 21.0958; No. 3, 20.9690; No. 4, 21.1879; No. 5, 21.2005; No. 6, 21.0512; No. 7, 21.1459.

^bThe found value of the KCN solution was as follows: In the case of No. 1, 48 c.c. was equivalent to 47.6 c.c. N/10 AgNO_3 ; in No. 4, to 47.65; and in the remainder, to 47.64 c.c. N/10 AgNO_3 .

copœia (1908) and also of the German Pharmacopœia (1910) is only 35 per cent by weight of absolute formaldehyde. On the other hand, with the exception of only one sample (No. 1), all contained more than 36 per cent by weight of absolute formaldehyde. Therefore, bearing in mind the comparative difficulty of manufacturing and keeping unaltered solutions of formaldehyde of considerably higher concentration than the minimum of the present U.S.P.; and also that by making the U.S.P. requirement more in harmony with these latter pharmacopœias and with what seems to be the actual condition of the American market, it would not mean that the purity requirement will be lowered but only that the solution will be just a little less concentrated; it would seem advisable that the formaldehyde requirement, in the next revision of the U.S.P., be changed from the present minimum of 37 per cent to a minimum of 36 per cent by weight of absolute formaldehyde. Also that the KCN method might with advantage be substituted for the present H_2O_2 method. The KCN method, applied to U.S.P. solution of formaldehyde, may be carried out as follows:

²⁴Zeit. anal. Chem., 44, 15-16 (1905): "Wir sind immer dafür eingetreten, diese beiden Arbeitsweisen bei der Handelsanalyse, besonders bei der des Schiffsanwals, sich gegenseitig zu benutzen und das Mittel der nach beiden Verfahren erhaltenen und hinreichend übereinstimmenden Werte als den wahren Formaldehydgehalt anzunehmen. Diese Forderung, zwei verschiedene Methoden gleichzeitig anzuwenden, bewirkt, dass Fehler nicht zufällig auf die eine oder die andere Methode, sondern auf die beiden Methoden verteilt werden. Beide benutzen viel mehr Reagentien als die eine, und die Formaldehydmenge wird durch andere Verbindungen zeigen können: die eine lässt alle Substanzen oxydieren, die andere alle die jengien, die in alkalischer Lösung durch Jod oxydiert werden oder in anderer Weise Jod verbrauchen, ohne es beim Ansäuern wieder frei zu geben."

²⁵Jour. Amer. Chem. Soc., 25, 1034 (1903).

²⁶Elvove: Amer. Jour. Pharm., 82, 203-211 (1910).

²⁷Evans Sons Lescher and Webb: Analytical Notes, 1907, 1908, p. 22.

²⁸Proc. Minnesota Pharm. Ass., 1907, p. 41; from Bull. No. 63, Hyg. Lab., U. S. Pub. Health & Mar. Hosp. Serv., Wash., p. 295.

Transfer 0.5 c.c. of the sample of Solution of Formaldehyde to a well-stoppered Erlenmeyer flask of about 150 c.c. capacity and determine its weight. Add to it immediately 100 c.c. of a solution of potassium cyanide of a strength close to tenth-normal (6.5 Gm. KCN to 1,000 c.c.), the exact strength of which is known. Mix well, and add to a mixture of 40 c.c. N/10 silver nitrate and 10 c.c. of dilute (10 per cent) nitric acid, in a 200 c.c. measuring flask. Wash the Erlenmeyer flask, adding the washings to the silver solution, and make up the whole to 200 c.c. Shake thoroughly, filter through a dry filter, and titrate the excess of silver in 100 c.c. of the filtrate by means of N/10 ammonium or potassium sulphocyanate, using ferric alum as indicator. The number of c.c. of N/10 sulphocyanate found to require, multiplied by 2, and the product subtracted from 40, will give the equivalent of the uncombined KCN in c.c. of N/10 AgNO₃. Subtracting this from the corresponding equivalent of the total KCN added, multiplying the difference by 0.3, and dividing this product by the weight of the sample taken, the quotient will represent the percentage by weight of absolute formaldehyde in the sample.

THE FLUORESCENT TEST FOR MINERAL AND ROSIN OILS.*

By PERCY H. WALKER and E. W. BOUGHTON.

The Outerbridge method¹ of detecting and estimating mineral and rosin oils in fatty oils, by taking advantage of the fact that the fluorescence, which is a property of some oils, is very much magnified by examining the oil in the light of an inclosed arc, introduces an entirely new method of oil-testing. The author of this method has made the interesting discovery that samples of oil which in sunlight show no fluorescence, when examined by the light of an inclosed arc frequently show a very strong fluorescence. He further states that the examination of a large number of vegetable oils failed to show a trace of fluorescence in any of them, while all samples of heavy mineral and rosin oil, whether debloomed or not, showed strong fluorescence in the proper light. Based upon this observation, he proposes to rapidly detect and determine mineral or rosin oil in fatty oils by matching the fluorescence of the oil under examination with that of a prepared set of standard mixtures.

Outerbridge's statement that fluorescence is very greatly magnified by making the observation in the light of an inclosed arc has been verified in this laboratory. In addition to the lights used by Outerbridge, the uviolet light was tried but found to be in no way equal to the inclosed arc, which is far superior for this purpose to any other source of light.

Fluorescence in an oil does not, however, prove the presence of mineral or rosin oil. Of the 16 samples of pure linseed oil furnished by Committee D-1 of the American Society for Testing Materials, Nos. 3, 5, 6, 10, 12, 13, 14 and 16 showed no fluorescence, Nos. 1, 2 and 11 showed slight fluorescence and Nos. 4, 7, 8, 9 and 15 showed marked fluorescence. A number of samples of other fatty oils of known purity, some of which were cold-pressed from the seed in the Bureau of Chemistry, showed marked fluorescence, in some cases as marked as that of many pure mineral oils.

In order to test the delicacy of the method, a sample of kerosene, which alone showed very strong fluorescence in the light of the arc, was mixed with linseed oil No. 3. This mixture, containing 1 per cent of kerosene, which could easily be detected by its odor, showed a slight fluorescence, but it was not so marked as that shown by linseed oil No. 4 without any addition of other oil. The pure linseed oil No. 3 was then heated to 300° C. and after cooling was found to be strongly fluorescent. The same development of strong fluorescence was observed in a sample of pure olive oil, which, before heating, showed no fluorescence. The examination of a number of turpentine demonstrated that while many of these samples show no fluorescence, some samples of mineral oil also show none, and some samples of undoubted purity show marked fluorescence. It appears, therefore, that while it is interesting to know that the inclosed arc is a very convenient means of strongly magnifying fluorescence, this fluorescence is not proof of the presence of mineral or rosin oil.

Most of the sugar mills in Java have now adapted their plants to the production of white sugars, although they can revert with little alteration to the manufacture of refinery sugars if necessary. The area planted to sugar cane in Java in 1910 amounted to 312,000 acres, against 301,134 acres in 1909 and 289,744 acres in 1908.

Details of production in the silver mines at Cobalt, Canada, show the progress being made in the processes of ore treatment. The quantity of concentrates shipped, which during the first nine months of 1910 amounted to 4,623 tons, rose to 6,915 tons in the same months of 1911. The effect of this increase in the concentration of low-grade ores was to diminish the total tonnage of the shipments, which fell from 23,824 tons to 18,592 tons. Concurrently, the output of bullion has largely increased, the shipments in the first nine months of 1910 having a value of \$247,263, while in 1911 they were worth \$1,116,571. The production of the silver camps in ounces was as follows: Cobalt, 22,272,783; South Lorrain, 626,131; Gowganda, 286,925. The last quarter of the year usually shows the heaviest output, and the probability, therefore, is that 1911 will be considerably in advance of 1910, when the silver yield was 30,651,417 ounces.

*From the Journal of Industrial and Engineering Chemistry.
"A Novel Method of Detecting Mineral Oil and Rosin Oil in Other Oils." By A. E. Outerbridge, Jr. Proceedings, Fourteenth Meeting American Society for Testing Materials.

METALLURGY.

ELECTRIC SMELTING OF ZINC-LEAD ORES AT SARPSBORG, NORWAY, AND AT TROLLHATTAN, SWEDEN.

An English company has been formed for the purpose of acquiring two electric smelting works, one at Sarpsborg, Norway, and one at Trollhattan, Sweden, and also the rights in connection with the construction of another at Tyse, near Odde, in Norway. The process and furnace used at these works are intended for the treatment of complex sulphide ores, and other material such as Broken Hill slime, and the intimately mixed sulphides of lead, zinc, copper and iron that occur in large quantities in various parts of the world, at present of little or no commercial value. Reports have been made by J. C. Moulden for the Sulphide Corporation, and by F. W. Harbord, the latter of whom made thorough tests of the plants at Sarpsborg and Trollhattan. The information in this article and the description of the processes employed are taken from recent reports by J. C. Moulden and F. W. Harbord. The electric furnace employed has carbon electrodes, and the heating and melting of the charge is effected by resistance with a current of low voltage. One electrode is in the bed, and the upper one is removable through the roof. The cross-section of the upper electrode measures approximately $2\frac{1}{4}$ sq. ft. In the first furnaces the charge was introduced through the roof, but more recently the door is placed in the side and the operation is thus made more continuous. The capacity is 3 tons, and 2.8 tons of roasted ore can be treated per 23 hours. The horse-power per furnace is 350, and at Trollhattan the cost of power supplied by the hydro-lectric company is 30s. 3d. per horse-power year. At Mr. Harbord's test on roasted Broken Hill slime, the energy used per ton of ore was 2,078 kilowatt hours. There are 11 furnaces at Trollhattan. The roasted ore, coke or anthracite dust and flux are charged into the furnace, and most of the zinc and some of the lead is volatilized. The greater proportion of this is condensed as a crude spelter with high lead content, but much of it is precipitated as metallic fume and oxide, and has to be returned to the furnace in subsequent charges. The lead that is not volatilized is reduced as liquid metal, and it contains a considerable proportion of the silver. Matte is formed of the iron, copper and sulphur, if present in the ore, and some of the lead, zinc and silver passes into it. Some lead also escapes with the slag. The crude zinc is refined in other electric furnaces and the final zinc product is exceptionally pure, averaging 99.9 per cent, and commanding a premium price in the market. At the time Mr. Harbord made his examination, the furnaces

were built of porous material, apparently similar to that used in old-fashioned zinc-distilling retorts, and some of the lead and silver percolated through and had to be caught underneath; this he calls "leakage" lead. He found also that the lead that exuded in this way during his experiments did not necessarily come from the charges under his observation, and was partly produced during the treatment of other ore at an earlier period. Thus the silver contents of the lead did not at all coincide with that of the ore treated by him. The general results of his trials were by no means ideal, and he makes many suggestions for improvements; but his conclusions are that the process may easily be developed into one of prime importance, especially at the new plant proposed at Tyse, where the estimated cost of a horse-power-year is only 14 shillings. During his experiments at Trollhattan, seven furnaces, during $27\frac{1}{2}$ days, 518 tons of roasted Broken Hill slime, 19 tons of calamine, $22\frac{1}{2}$ tons of zinc-lead powder produced previously, and produced 160.8 tons of crude zinc which by refining yielded 112.4 tons of pure zinc and 24.7 tons of lead, together with 36 tons of powder; in addition to which 41 tons of lead was tapped containing 141 oz. silver per ton and "leak" lead amounting to 17 tons was recovered assaying 27 oz. silver per ton, also 9 tons of skimmings containing zinc, lead and silver. The recovery from these various sources was 64 per cent of the zinc, 74 per cent of the lead and 46 per cent of the silver, not reckoning the metals recovered in the powder form. Mr. Harbord goes into details as to many points where increased extraction and lower costs may be obtained. Mr. Moulden is of opinion that, by means of improvements already indicated, the extraction should eventually be 75 per cent of the zinc, 80 per cent of the lead and 80 per cent of the silver, and on this basis he figures the profit obtainable on slime bought from the Sulphide Corporation and shipped to Sweden. This slime is taken to average 32 per cent zinc, 23 per cent lead and 25 oz. silver, and on the current basis of payment would cost £2. 16s. 8d. per ton at Broken Hill. The freight to Trollhattan would be £2. 5s. 3d., making the total cost per ton at the smelter £5. 8s. 5d. The value of the recovered metals on the percentage figured by Mr. Moulden would be £9. 9s. 11d. The cost of treatment he estimates at £3. 0s. 7d., so that the margin of profit would be £1. 9s. per ton of slime. It is unnecessary to say that the process is in its infancy, and that many improvements are already obvious; also that everything depends on the cost of electric heat. Enough is shown, however, in these reports to make it clear that the process and method of metallurgy deserve close attention.

CARBORUNDUM FOR SPECIAL STEELS.*

In the *Revue de Metallurgie* for September, 1911, is a long paper on the above subject by Dr. L. Baraduc-Muller. It gives the results of his very interesting experiments in the basic open-hearth plant of Ougrée-Marihaye in Belgium.

Following the first laboratory tests, the first trial was made on a 15-ton heat, the final composition aimed at being 0.50 per cent carbon and 0.50 per cent silicon. Briquettes were made from the following mixture, measuring about $8.7 \times 4.3 \times 2$ in. The total weight added was about 400 kg. (882 lbs.).

Carborundum, 94.5 per cent pure.....	141.0 parts
Red oxide of iron, 95 per cent.....	214.0 parts
Carbon, 94 per cent pure.....	14.8 parts
Lime	8.5 parts
Anhydrous silicate of soda.....	6.0 parts

384.3 parts

The bricks, dry but not heated, were charged into the furnace before tapping. They floated in the slag, about two-thirds of their thickness being submerged. They did not break up, notwithstanding the great change in temperature. Their upper surface was soon covered with little jets of burning carbon monoxide and small incandescent globules of metal formed that trickled down and through the slag. After being charged for 22 minutes they had all disappeared. At the end of the reaction the slag was frothy. When casting the steel, the ingots did not show much silicon. The analyses of a test taken before adding the bricks and an average of three tests taken while pouring are given below. The increase of silicon is not 10 per cent of that hoped for.

	1st Test.	2d Test.
Carbon	0.44	0.54
Manganese	0.766	1.19
Sulphur		0.059
Phosphorus		0.064
Silicon	0.018	0.051

Calculation shows that only 8.1 per cent of the silicon charged has entered the steel, after allowing for a normal loss of 30 per cent. The great loss is brought about by the silicide of iron produced trickling in small drops through a basic slag, which also contains considerable oxide of iron. The bricks are also evidently too large, because of the long time it takes for their melting.

A second test was made on a 15-ton heat, the final analysis wished for being 0.2 per cent silicon, 0.25 per cent chromium and 0.45 per cent carbon. Forty-five bricks were taken of the same size as before, but each one was cut into four parts. The total weight was about 251 lbs. (114 kg.). The mixture used was:

Carborundum, 94.5 per cent pure.....	23.7 parts
Sesquioxide of chromium, 98 per cent pure.....	60.6 parts
Carbon, 94 per cent pure.....	7.3 parts
Lime	1.2 parts
Anhydrous silicate of soda.....	3.0 parts

96.8 parts

These briquettes were charged as before, and had

all disappeared in 15 minutes. The slag was slightly foamy. The chief point of interest was to see how much chromium would be introduced. A tapping test gave 0.02 per cent silicon, and the final steel gave:

Carbon	0.450 per cent
Sulphur	0.064 per cent
Phosphorus	0.027 per cent
Manganese	0.630 per cent
Silicon	0.225 per cent
Chromium	0.220 per cent

The result was therefore very satisfactory. Allowing for a normal loss of 30 per cent, calculation shows that 93.7 per cent of the chromium charged has entered the steel. The high silicon in the steel was obtained by the use of ferrosilicon.

Further tests were made with practical considerations in mind. In order to obtain silicon, it was evidently necessary to add the mixture either in the tapping spout or the ladle. This made necessary a speed of reaction such as would be complete before the slag came. The minerals and other materials to be used were crushed very finely, thoroughly mixed, and thin cakes made about $4.3 \times 0.98 \times 0.39$ in. The proper weight of cakes was placed in a conical heap at the bottom of the ladle, in such position that the jet of steel would strike it. The first test under the new conditions was made on a 15-ton heat, the final analysis wished for being 0.6 per cent carbon and 0.25 per cent silicon. About 550 lbs. (250 kg.) of the following mixture was used:

Roll scale.....	100 parts
Carborundum, 94.5 per cent pure.....	53.3 parts
Carbon, 90 per cent pure.....	4.5 parts
Cement	10.0 parts
Silicate of soda.....	3.0 parts

170.8 parts

The analysis of the roll scale gave:

Fe ₂ O ₃	65.10 per cent
FeO	26.20 per cent
Fe	5.30 per cent
Mn.	0.60 per cent
SiO ₂	(Sand) 1.60 per cent
Not determined.....	0.50 per cent

99.30 per cent

The cakes were previously heated in a small reverberatory furnace, and were probably at 600° C. when the steel fell on them. (It was hoped that this would be hot enough.) This was done as a precaution, for all that the steel had to do was to heat them to 1,350° C. The reaction would then commence and, being slightly exothermic, would have no tendency to chill the steel and make a skull in the ladle. Very few cakes floated to the surface of the metal, and the reaction was apparently over in 1 min. 23 sec. The results of a final furnace test and of a ladle test were as follows:

	Furnace Test.	Ladle Test.
	Per cent.	Per cent.
Carbon	0.62	0.65
Sulphur		0.030
Phosphorus		0.023
Manganese		0.650
Silicon	0.018	0.169

Calculation shows that 66 per cent of the silicon

*From The Iron Age.

available has been taken up by the steel, after allowing for a normal loss of 30 per cent. This is considerably better than before, but it is not yet entirely satisfactory.

A second experiment was made with the small cakes, except that they were not heated before using, and after allowing for the normal loss, only 47 per cent of the silicon in the mixture entered the steel. Tests also showed that the silicon was not uniformly distributed in the ladle of steel, the part poured the last being much higher in silicon than the first. This would indicate that not enough time was available for the reaction, and the tests also show that the silicon is produced toward the top of the ladle and not rapidly and completely at the bottom, as it should be.

The next experiment was made on a 15-ton heat which was desired to be 0.45 per cent carbon, 0.25 per cent chromium and 0.2 per cent silicon. The little cakes were made of the following mixture:

Chromite	93.5 parts
Carborundum, 94.5 per cent pure.....	32.6 parts
Carbon, 90 per cent pure.....	6.0 parts
Silica, 95.0 per cent pure.....	10.5 parts
Cement	7.0 parts
Silicate of soda.....	3.0 parts
	153.4 parts

The silica was added to make a slag with the lime, magnesia and alumina of the chromite, that would melt between 1,350° and 1,380° C. The chromite contained 58.8 per cent Cr_2O_3 , 18.8 per cent FeO , 8.7 per cent Fe_2O_3 , 10.2 per cent MgO , 1.16 per cent CaO and 2.46 per cent SiO_2 . About 595 lbs. (270 kg.) of the mixture was put in the bottom of the ladle. The steel was poured hotter than usual, and the reaction was over in 1 min. 40 sec. The analyses of the final tapping test and of an average of three tests taken while pouring the steel into ingots were as follows:

	Furnace Test.	Ladle Test.
Carbon	0.39	0.48
Sulphur	0.058	0.058
Phosphorus	0.019	0.034
Manganese	0.745	0.734
Silicon	0.0094	0.136
Chromium		0.196

The three ladle tests showed a very uniform material. Calculations show that 65.4 per cent of the silicon and 85.3 per cent of the chromium entered the steel, after allowing for the normal loss. The aim of the experiment was to find out if natural oxides could be used with the carborundum.

The experiment tried with the next heat was to see whether chromium alone could be introduced and no silicon. In other words, it was hoped that both the carbon and silicon of the carborundum would reduce chromium. The addition was made in the furnace, as the basic slag could not affect the silicon of the mixture, and the usual amounts of ferrosilicon were added in the furnace and ladle. The 15-ton heat was desired to finish 0.50 per cent carbon, 0.20 per cent silicon and 0.25 per cent chromium. The mixture

was made into bricks about $8.7 \times 4.3 \times 2$ in., which were cut into three parts. They were composed of:

Chromite, with 58.8 per cent Cr.....	100 parts
Carborundum, 94.5 per cent pure.....	20.2 parts
Cement	12 parts
Silicate of soda.....	3.5 parts
	135.7 parts

Ten minutes before tapping, about 397 lbs. (180 kg.) of the briquettes was charged, after taking the last furnace test. The reaction was over in 7 min. 35 sec. The slag was a little frothy and the casting temperature was normal. The analyses were as follows:

	Furnace Test.	Ladle Test.
Carbon	0.51	0.53
Sulphur		0.050
Phosphorus	0.049	0.050
Manganese	0.648	0.609
Silicon	0.028	0.136
Chromium		0.200

The ladle test result is the average obtained on three samples taken at the beginning, middle and end of pouring. They agreed very closely, showing good uniformity. After allowing for normal loss, it is found that 79.3 per cent of the chromium was obtained. The amount of ferrosilicon added was unfortunately forgotten, so no calculations could be made in regard to silicon.

Many pages are then devoted to the question of the comparative cost of the carborundum method, and the use of ferrosilicon and ferrochromium for adding the required chromium and silicon to the steel. Under the conditions existing in Belgium, the addition of silicon alone by this method is not so cheap as the use of ferrosilicon. When chromium and silicon are to be added, however, the saving in cost varies from 23.3 per cent to 45.5 per cent. Laboratory tests show that nickel-chromium steels can be also made by this method, and it would be interesting to see what results would follow the use of mixtures of carborundum with tungsten, molybdenum and vanadium minerals.

As a consequence of the experiments outlined above the method has been adopted by the Ougrée-Marihayé works.

The production of the metalliferous mines and works of the Province of Ontario for the first nine months of the present year was: Gold, 2,276 ounces (\$42,320); silver, 23,185,860 ounces (\$11,593,286); copper, 6,769 tons (\$867,489); nickel, 12,711 tons (\$2,731,575); iron ore, 172,868 tons (\$437,650); pig iron, 296,856 tons (\$4,482,635); white arsenic, 3,016,385 pounds (\$45,535); cobalt and nickel oxides, 277,766 pounds (\$80,372). This is an increase of \$1,799,629 in the production of silver, \$13,591 in gold, and \$163,744 in the output of iron over the first nine months of 1910. The decrease in the production of nickel in the same period was \$258,076; copper, \$54,947; and pig iron, \$556,991.

THE PREPARATION AND PROPERTIES OF METALLIC CERIUM.*

By **ALCAN HIRSCH.**

Recognizing the fact that, in spite of the great volume of research upon the rare earths, but little had been done upon the rare earth metals themselves, the author undertook a thorough investigation of metallic cerium, having in view, first, the preparation of sufficiently large quantities of metal from the purest possible cerium salts, and the study of its physical and chemical properties and its alloys, and, second, the preparation of alloys which might have commercial value from the rare earth residues obtained as by-products of the incandescent gas mantle industry.

The paper contains a bibliography of previous work upon this subject, for which the reader is referred to the original, published in the Transactions of the American Electro-Chemical Society.

In the preparation of cerium and the other rare earth metals, two general methods have been used, viz.: First, the reduction of the oxide or chloride by one of the very electropositive metals; and, second, the electrolysis of a fused salt.

The general plan of the research here reviewed was to make use of the mixed rare earth materials in working out the local conditions for the preparation of the compounds desired for reduction, and then to apply these methods upon pure cerium compounds.

The material for the experiments was mixed rare earth oxalates obtained as by-products of the extraction of cerium and thorium from monozite sand. These oxalates, upon ignition at blast lamp temperature, yielded mixed oxides containing 97.8 per cent rare earth oxides, of which about 49 per cent was cerium dioxide.

Thermal reduction of the oxides by means of various powerful reducing agents gave no satisfactory results, compounds, alloys or lower oxides of the rare earths being formed. Magnesium, calcium, aluminum and black thermite gave alloys or lower oxides. Carbon and silicon gave carbide and silicide, respectively.

For the electrolytic method of preparation, the primary demand was for pure, anhydrous salts capable of fusion without decomposition. Trichloride and double fluorides having been using the past, their preparation and electrolysis upon a large scale was investigated.

CeCl_3 , like the chloride of magnesium, is decomposed into oxide and hydrochloric acid when its aqueous solution is evaporated to dryness and the residue calcined. Chloride containing oxide or oxychloride cannot be electrolyzed, pure CeCl_3 being necessary to this process. After trying many schemes,

pure CeCl_3 which melted to a limpid liquid and could be readily electrolyzed was finally prepared as follows: Double rare earth ammonium nitrates were evaporated very slowly with concentrated HCl until converted to chloride. This chloride was then dehydrated by careful heating in an atmosphere of dry HCl gas, the product, which is very hygroscopic, being sealed up in paraffined bottles. In this way large quantities were prepared.

Anhydrous AlF_3 was prepared with difficulty, with a view to using it as an electrolyte for the preparation of cerium. This substance dissolves cerium oxide giving a conducting solution, but the electrolysis of this jarred material was found to be impracticable for a number of reasons, of which the following are chief: First, the solution of oxide in fluoride has a very high melting point (over $1,000^\circ$); second, graphite vessels alone will stand the corrosive action of the fluoride, but at the high temperature demanded cerium unites with the graphite, forming carbide.

It was from the fused anhydrous chloride that cerium was finally obtained successfully and in large lots. Wrought iron vessels were employed, which served as cathode as well as container, the anode being graphite. Currents of 200 amperes at 15 volts yielded cerium at current efficiency of about 40 per cent, the product, when pure CeCl_3 was used, being a well-melted regulus containing about 98 per cent of Ce and 1.2 per cent Fe.

The only efficient way of purifying cerium from iron was by means of the cerium amalgams, in the promotion of which the iron, etc., came to the surface and was skimmed off. The cerium was finally separated from the mercury by distillation in vacuo.

The physical properties of cerium may be briefly stated as follows:

The atomic weight of cerium is 140.25. Its specific gravity was found to be 6.92 at 25°C .

No definite crystals of cerium were obtained, though photomicrographs of polished and etched surfaces show homogeneity of structure.

The hardness of the metal depends upon whether the surface is rolled or freshly cut, the scleroscope showing a hardness of 25.9 for rolled and 9.5 for freshly cut surfaces.

Cerium is malleable and highly ductile.

Its specific electrical resistance is 71.6 microhms per centimeter cube, while, on the other hand, its heat conductivity is high.

Cerium is ferro-magnetic.

Its melting point, determined by an ingenious method, was found to be 635°C .

The ultimate strength of cerium is equal to 12,900 lbs. per sq. in., the test bar breaking with a snap like cast iron.

Its specific heat was found to be 0.0524.

The heat of combustion of cerium to CO_2 , as determined in a Mahler bomb calorimeter, is about 60,900 cal. per gram.

*A review of the author's work presented as a thesis for the Doctor's Degree at the University of Wisconsin, and read at the Twentieth General Meeting of the American Electrochemical Society in Toronto, Canada, September 21-23, 1911. Abstracted by C. A. Tibbals.

The thermo-electric and optical properties of the metal were studied and some results given.

THE CHEMICAL PROPERTIES OF CERIUM.

Ce decomposes water, very slowly in the cold, more rapidly at the boiling point of water.

Ethyl alcohol, auryl alcohol, chloroform, carbon tetra-chloride, concentrated sulphuric acid and concentrated alkali solutions have no action upon it in the cold.

Ethyl ether and hydrogen peroxide react slowly with Ce at room temperature.

Dilute H_2SO_4 and concentrated and dilute HNO_3 and HCl react moderately with cerium in the cold and vigorously when hot.

Chlorine, bromine and iodine unite directly with the metal, forming cerous salts.

Cerium burns in the air when heated to $160^\circ C.$, unites with hydrogen at $345^\circ C.$, forming hydride, and with nitrogen at $1,000^\circ C.$, forming nitride. The sulphide is also formed by direct union. Ce does not unite with carbon monoxide.

ALLOYS OF CERIUM.

Cerium was alloyed with a large number of elements, and the superficial properties of the alloys noted. In most cases cerium was present to the extent of about 70 per cent.

The pyrophoric properties of the cerium alloys appear to be their most interesting aspect.

Iron, nickel, calcium, magnesium and silicon alloys exhibit this property to the most marked extent, while alloys of platinum, tin, arsenic, selenium, lead, zinc, cadmium, chromium, manganese and tungsten show it to a somewhat less extent.

The silver, gold, copper, antimony, tellurium, aluminum and mercury alloys are not pyrophoric.

The author ascribes the pyrophoric properties of cerium alloys to the following causes: The alloys of certain metals high in cerium are not homogeneous, being made up of cerium and cerium-metal compound. The cerium is soft and malleable and acts as a binder, while the compound is hard and brittle, and when scratched with a file particles are detached. These particles are raised by friction above the ignition temperature of cerium (160°), and sparks result.

A number of the alloys disintegrate upon aging, notably those of gold, silver, copper, tin, lead and aluminum.

They are for the most part brittle and hard, although the alloys of cerium with gold, antimony, arsenic and lead are somewhat soft.

The author acknowledges obligation to Mr. H. S. Miner, chief chemist of the Welsbach Light Company, for rare earth materials furnished by him, and to Professors Burgess and Leuber of the University of Wisconsin.

Curing lemons in the sweat houses by exhaust from gasoline engines instead of by coal-oil stoves is being tried in California.

USES OF BLAST FURNACE GAS: RESULTS IN THE OPEN HEARTH FURNACE—WHEN MIXED WITH COKE OVEN GAS.*

In his dissertation for the degree of doctor of engineering, presented at the University of Breslau, Rudolf Buck discussed at length the "Uses of Blast Furnace Gas."* After a brief historical introduction he takes up the subject under the following headings:

1. The uses of blast furnace gas in well known ways, with special consideration of the latest developments.
2. The blast furnace gas burner.
3. The use of blast furnace gas in the foundry.
 - (a) In pipe foundries.
 - (b) In ordinary foundries.
4. Its use in open hearth furnaces.
5. Its use for the heating of coke ovens.

Especial stress is laid on the suitability of blast furnace gas for use in the foundry, for drying molds of all kinds, heating ladles, etc. Many tests have been made of the gas alone, and when mixed with coke oven gas, and also of hot air from the blast furnace stoves. Coke oven gas, because of its large percentage of hydrogen, is not suitable for foundry use. For instance, in drying a pipe mold the lower part is well dried, but the water produced from the hydrogen when burnt condenses in the upper part and keeps it moist. Because of this, coke oven gas either alone or mixed with blast furnace or producer gas should not be used for foundry work. Hot air from the stoves is not economical, as the blast furnace gas can be burned more profitably directly in the foundry. Waste gases from the stoves and boilers are not suitable, because of the great variation in temperature.

METHODS OF CLEANING THE GASES.

The composition of the blast furnace gas is briefly considered, and then its purification is taken up. Emphasis is laid upon dry cleaning, brought about by sudden changes in the direction of the gases, and by reducing the velocity by increasing the area of the pipes. The so-called dust pockets are for this latter purpose; they also serve to uniformly lower the temperature. It is not impossible that the change in volume caused by the drop in temperature has an important influence on removal of the flue dust. The most suitable construction for these dry purifiers has not yet been decided upon, many different types being experimented with. The main thing is to build them with the greatest possible diameter in order to bring about the greatest possible lowering in velocity.

An example may be taken from practice. The dry cleaner is 65 ft. $7\frac{1}{2}$ in. high and 27 ft. $10\frac{1}{2}$ in. diameter. The incoming gas has an average velocity of

*From The Iron Age.
†Reproduced in Stahl und Eisen, issues of July 20 and 27 and August 3, 1911.

19.7 in. per second, and the highest value is 31.5 per second. The dust was from 2.6 to 3.5 grains per cu. ft. and the temperature 390° F. The gas left with a temperature from 170 to 212° F. and from 0.65 to 0.78 grain of dust per cu. ft. This corresponds to 60° lowering of the amount of dust. It is impossible, however, to use this roughly cleaned gas direct in gas engines. Today gas cleaning is divided into rough and final cleaning. The first is divided into dry and wet rough cleaning. The latter, that follows the rough cleaning, is divided into wet cleaning in fans or centrifugal washers and dry cleaning with filters and fans. The most important thing about wet cleaning is that it is necessary to reduce both the dust and the water as low as possible and also to cool the gas to a minimum. These subjects are dealt with briefly, the author favoring centrifugal washers for removing the very fine dust. He complains, however, of the cost of effecting wet cleaning by whatever system.

ADVANTAGES OF CLEANED GAS.

The advantages of cleaning the gas are given as follows:

1. The troubles brought about by pipes being stopped up with flue dust are either removed entirely or reduced to a minimum.
2. The volume of the gas is greatly reduced, whereby the pipes may be made smaller, thus bringing about lower erecting costs.
3. The steam in the gas is condensed and separated.
4. The gas is more easily lighted and burnt.
5. The heating value of the gas is increased and its efficiency greater for any kind of use.
6. The operation of stoves and boilers is more regular.
7. The cost of cleaning pipes and tubes is lower and the furnace will not have to stop so often.
8. The cleaning of the stoves, bringing about loss of heat, will not be necessary so often and the unfavorable action on the brick work through cooling and cleaning will be less.
9. The life of the whole apparatus will be considerably increased and the cost for material will be less.
10. The heating surfaces in the stoves being cleaner the efficiency will be greater. This means saving of gas.
11. The operation of the gas engines will be more certain.
12. Electrical energy can be produced by means of gas engines and the excess over requirements can be sold to neighboring works or communities.
13. Thorough final cleaning of blast furnace gas makes its use possible in open hearth furnaces, in mixers, for drying operations, in foundries, for heating furnaces in rolling mills, etc.
14. It can be used to heat coke ovens, the dearer

coke oven gas being sold in large part for lighting purposes.

These advantages are not brought about by removing the dust only, but also by cooling and reducing the water to the lowest possible amount. The drying of the gas can hardly be effected except by vigorous cooling, when the maximum amount of steam is condensed and separates as water. Practice has shown that the loss of heat through cooling is not so great as the poor efficiency brought about by wet gas.

The use of blast furnace gas in engines, stoves and boilers is briefly taken up, and then comes an important section on burners. These burners are designed to give as complete combustion of the gas as possible, and flames of various shapes to suit operations in the foundry. This section of the paper is illustrated with drawings of the burners and photographs of the flames. It is followed by a well-illustrated section on the use of the gas in the foundry.

BLAST FURNACE GAS IN THE OPEN HEARTH.

Producer gas can be replaced by coke oven gas, or better still, by a mixture of coke oven and blast furnace gas. The mixture should be in such proportion that the gas is from 179 to 213 B.t.u. The amount of gas necessary for the operation, which depends on the size of the furnace and the stage of the process, is easily regulated by valves in the separate gas pipes. Tests have been made at Friedrich-Wilhelms Hütte in Mulheim-on-the-Ruhr, and for over two and a half years two acid furnaces of 12 to 15 tons have been heated successfully. The process can be carried out with blast furnace gas alone, but it takes so long that the quality of the steel can easily be impaired. With the use of the mixed gases, on the other hand, the operation is not only quite as successful as with producer gas, but is carried on more quickly and to better advantage. A mixture of coke oven and blast furnace gas in about the proportion of 1 to 4 is necessary, so that the coke oven gas is added to the other before coming to the reversing valve.

In Frederick-Wilhelms Hütte the blast furnace gas is brought from the furnaces in an elevated pipe line 23.6 in. in diameter; the coke oven gas comes in an underground pipe line 9.8 in. in diameter. The arrangement of the furnace and the size of ports and checkers are the same as when using producer gas. Details are given of the six test heats, illustrated by diagrams.

In test No. 1 the mixture was in the ratio of 1 to 4 and the average heating value of the mixture was 213 B.t.u. per cu. ft. Its analysis gave 5.4 per cent CO₂, 0.2 per cent O, 18 per cent CO, 28.5 per cent H₂ and 5.1 per cent CH₄. Temperature measurements made on the furnace after the charge was melted varied between 1,774 and 1,900° C. These were taken with a Wanner pyrometer through the middle door. Details are also given, both for this and the other heats, of the temperature and composition of the waste gases

and the temperature of the heating gases, both taken in the checkers. The analysis of the finished steel was 0.224 per cent C, 0.049 per cent S, 0.062 per cent P, 0.52 per cent Mn and 0.18 per cent Si.

In the second test the mixture of gases was in the ratio of 1 to 4.5. The average heating value was 186.8 B.t.u., and the analysis gave 4.2 per cent CO_2 , 0.8 per cent O, 16 per cent CO, 29 per cent H_2 and 3 per cent CH_4 . The analysis of the finished steel was very similar to the one given above, and the temperature of the furnace varied from 1,640 to 1,900° C.

In the third test the mixture was again in the ratio of 1 to 4. The analysis of the mixture gave 4.4 per cent CO_2 , 0.6 per cent O, 15 per cent CO, 36.7 per cent H_2 and 4.2 per cent CH_4 . The average heating value was 199.3 B.t.u. The furnace temperatures varied from 1,692 to 1,900° C, and the analysis of the final steel was 0.28 per cent C, 0.053 per cent S, 0.067 per cent P, 0.49 per cent Mn and 0.18 per cent Si.

The fourth test was not continued long enough to get the finished steel. Blast furnace gas alone was used. Its analysis gave 8.8 per cent CO_2 , 0.2 per cent O, 31.0 per cent CO and 2.1 per cent H_2 . The average heating value was 112.3 B.t.u. The furnace temperature during the period of testing varied from 1,478 to 1,549° C.

The fifth test also did not last very long. The mixture was in the ratio of 1 to 6.5. The average heating value was 162.9 B.t.u., and the analysis gave 5.8 per cent CO_2 , 0.6 per cent O, 19.2 per cent CO, 21.2 per cent H_2 and 2.3 per cent CH_4 . The furnace temperature varied from 1,494 to 1,640° C.

In the last test the mixture was in the ratio of 1 to 3. Its analysis gave 3 per cent CO_2 , 0.2 per cent O, 11.2 per cent CO, 38.2 per cent H_2 and 5.5 per cent CH_4 . The average heating value was 224.7 B.t.u. The temperature of the furnace increased to a maximum of 1,900° C. The analysis of the final steel is not given. The samples of gas for analysis were taken about half way through each test, and carefully analyzed with the Orsat apparatus. They were taken just before entering the reversing valve. The unusually low value for methane, compared with the hydrogen of the mixed gas, can not be explained. The coke oven gas had on the average 1.5 per cent CO_2 , 7 per cent CO, 42 per cent H_2 and 24 to 28 per cent CH_4 . The blast furnace gas has been given already. The results of analysis, however, cannot be questioned.

Blast furnace gas is also used with success to heat an annealing furnace for steel castings. This furnace is so arranged that blast furnace-coke oven mixed gas or producer gas can be used.

The tapping spouts of the open hearth furnaces are also heated with blast furnace gas.

The paper closes with a description of the way in which blast furnace gas may be used in by-product coke oven practice, thus setting free more of the rich coke oven gas for lighting purposes.

THE EFFECT OF ORGANIC AND INORGANIC "ADDITION AGENTS" UPON THE ELECTRO-DEPOSITION OF COPPER FROM ELECTROLYTES CONTAINING ARSENIC.*

By CHING YU WEN AND EDWARD F. KERN.

In the electrolytic process for the refining of copper many of the impurities, notably arsenic and antimony, accumulate in the electrolyte, from which they may be subsequently deposited upon the cathode, causing the cathode copper to be nodular and brittle. Arsenic, as has been well known, may accumulate up to a certain more or less definite concentration before it begins to contaminate the deposited copper. In practice, then, it is of primary importance to keep the electrolyte up to a certain degree of purity, which is usually accomplished by withdrawing portions of the electrolyte at intervals and replacing it with fresh solution, the copper being recovered from the impure solution by crystallization, or by electrolysis, using insoluble anodes. This frequent replacement of electrolyte, and the corresponding recovery of copper from the impure solution, complicates the process of refining and adds to its cost.

The formation of dendritic "trees," particularly upon the edges of the cathodes, is a source of trouble and expense to the refiner, for these "trees" must be removed by hand labor lest they short-circuit the current.

The investigation here reviewed was carried out with the two-fold object:

"First, to prevent the deposition of arsenic and antimony upon the cathode, and, second, to prevent the formation of dendritic 'trees.'"

The problem was to produce solid and smooth deposits from copper electrolytes high in arsenic by the aid of organic and inorganic "addition agents."

The literature shows that little investigation of "addition agents" has been made in the past. The authors review the work done upon the subject, notably that of Kiliani,¹ Barchers,² Hoffman,³ Wickes,⁴ Kern and Jarves⁵, and Förster and Seidel.⁶ Their results may be summarized as follows: The per cent of arsenic deposited from solution is not a function of the potential difference between electrodes⁴: "The junction of an addition agent in an electrolyte is to maintain a reducing menstruum around the cathode, which, in turn, causes the deposit to form denser and smoother."⁵ The deposits formed in electrolytes containing alkali or alkaline earth salts are generally smoother and less crystalline than those from electrolytes which do not contain

*A review of a paper presented at the twentieth general meeting of the American Electrochemical Society in Toronto, Canada, September 1-2, 1911. Abstracted by C. A. Tibbals.

¹Kiliani, *Lege und Hartmannsches Zeitung*, 1885, p. 249.

²Barchers' "Electrolytic Smelting and Refining," p. 206.

³Hoffman, *T. A. I. M. E.*, 34, 312 (1904).

⁴Thesis, School of Mines, Columbia Univ.

⁵School of Mines Quarterly, 30, 119 (1909).

those salts, the explanatory theory being that the good deposit is due to the reducing effect of the alkali or alkaline earth metal ions surrounding the kathode.⁷ The deposits formed at higher temperatures are more coarsely crystalline and show a higher tensile strength than those formed at lower temperatures.⁸

The materials used in the various series of experiments were as follows:

Copper anodes, $4\frac{1}{2}'' \times 2\frac{3}{4}'' \times \frac{3}{8}''$, containing about 1% As and 1% Sb.

Pure sheet copper kathodes.

Electrolytes: A, 15% CoSO_4 , SH_2O , and 10% H_2SO_4 . B, 15% CoSO_4 , SH_2O , 10% H_2SO_4 and 10% As as HAsO_4 .

By mixing A and B in varying proportions, six solutions containing, respectively, 1.5%, 2%, 3%, 4%, 6% and 8% As were prepared.

The apparatus consisted in a battery of eight cells, four in each of two water batteries kept at different temperatures, the space between the electrodes and the current density (40 amperes per sq. ft.) being kept constant in all experiments. Circulation of electrolyte was provided by mechanical stirrers. Most of the runs were of 50 to 100 hours' duration, the potential between electrodes being measured at two-hour intervals.

A large number of experiments are described in detail under the following headings:

I. Electrolytes which contained no "addition agents."

II. Electrolytes which contained inorganic "addition agents."

III. Electrolytes which contained organic "addition agents."

IV. Electrolytes which contained organic and inorganic "addition agents."

In each experiment, time, current density, potential, temperature, composition of electrolyte before and after the run, and composition of anode and kathode metal are stated, photographs of the kathodes shown, and the physical properties of the deposited copper described.

SUMMARY OF EXPERIMENTS.

In the first series, wherein no "addition agents" were employed, four runs were made. Electrolytes containing from 1.5% to 8% of As were used, and the effect of temperatures of from 20° C. to 60° C. were noted. The results point to conclusions as follows:

When the electrolyte contains no "addition agent," large quantities of As and St are deposited with the copper, and large trees form on the kathode, when the percentage of As is from 1.5 to 4. When the electrolyte contains from 6% to 8% of As, the As act as "addition agent," especially at temperatures of from 40° to 60°, the deposits being purer, brighter, more dense and coherent and less brittle than when

lower percentages of As are present. In general, the deposits were better at the higher than at the lower temperatures.

In the second series, wherein inorganic "addition agents" were used, solutions containing from 1.5% to 8% As were electrolyzed at various temperatures and with addition of the following compounds: NaCl (very small, and large quantities), HCl, Na_2SO_4 (little and much), $\text{Al}_2(\text{SO}_4)_3$, and AlCl_3 . Small quantities of sodium chloride, nitrate chlorate and borate were also used as "addition agents" in electrolytes containing 1.5% As, and the resulting solutions run under the same conditions.

In general, all of these salts have a good effect, both as to purity and physical properties, upon the deposited copper, even when large quantities of arsenic are present in the electrolyte.

Of all of them, sodium chloride (present to the extent of 0.01%) is best, with hydrochloric acid, cuprous and cupric chlorides, and sodium sulphate following. Aluminum sulphate has a markedly good effect as "addition agent." In general, the higher the metal of the "addition agent" in the electro-chemical series, the purer, smoother and less brittle the deposit.

The chemical behavior of the Na and the Cl ion in holding back the deposition of arsenic is discussed. Each shows its good effect in that all the sodium salts and all the chlorides tried improved the character of the deposit. The fact that sodium chloride is the best of inorganic "addition agents" is due to its yielding both the Na ion and Cl ion in solution.

In a third series the organic substances gelatin, glue, tannin and peptone were used as "addition agents," electrolytes like those used in the other experiments being employed. The first three produced a remarkably good effect upon the character of the copper deposit, both chemically and physically, while peptone has a bad influence upon the deposit.

Copper deposits formed when the electrolytes contain both organic and inorganic "addition agents" are all good. Those where the combination of gelatin and sodium chloride is used are best, while the combinations glue and sodium chloride, and glue and cuprous or cupric chloride show almost as good results upon the kathode copper. It is especially interesting that, whereas peptone alone has a very bad effect upon the copper deposit, the combination of peptone and sodium chloride makes an excellent "addition agent," the chlorine ion overcoming all the bad effects of the peptone.

In general, the higher the temperature of the electrolyte, the better the quality of the deposited copper, and, of course, the lower the resistance of the electrolyte.

The authors conclude that the combination of sodium chloride (0.01% Cl) and gelatin (0.01% to 0.02%) is the best "addition agent" for copper sulphate electrolytes high in arsenic, for the copper deposited under these circumstances "possesses the greatest purity and highest ductility."

⁷Zeit. f. Electrochem., 5, 508 (1899).

⁸Trans. Am. Electrochem. Soc., 15, 472 (1909).

ELECTRIC FURNACE TREATMENT OF NICKEL ORE, AND THE DEVELOPMENT OF A COMMERCIAL PROCESS.*

By WALTER LEONARD MORRISON.

(A thesis presented to Case School of Applied Science for the degree of Engineer of Mines; read at the General Meeting of the American Electrochemical Society, September, 1911.)

The thesis concerns attempts to treat, for profitable extraction, the nickel ores of the Consolidated Nickel Co. as found at their mine near Webster, N. C.

The nickel deposits occur in peridotite, for a considerable area traversing the formation in veinlets and intimately associated with other silicates of alumina, iron and magnesia. An impure genthite is a form in which the nickel is found to a considerable extent.

	Hard Green Ore.	Soft Green Ore.	2 Samples of Genthite.	
NiO	1.90	8.10	16.60	17.84
SiO ₂ ...	34.00	42.00	49.89	55.38
MgO ...	48.00	36.20	22.35	15.62
Al ₂ O ₃ ...	3.40	1.20	0.00	0.00
Fe ₂ O ₃ ...	1.00	1.00	0.06	0.56
Cr ₂ O ₃ ...	1.70	2.10	..	..
H ₂ O ...	10.00	10.80	12.36	10.77
CoO	..	..	0.00	0.00

The ore deposit has been known for over 50 years, and, although the amount available is very large, its tenor in nickel will average only about 1.75%. Copper and sulphur are absent from the deposit, but as the nickel is found as silicate it is not susceptible to water concentration, while all efforts at chemical extraction or smelting have been commercial failures. As better results might be expected with reduction in the electric furnace, this was tried on a commercial scale, producing a silicide of iron, nickel and chromium which might be useful for improving the quality of steel.

EXPERIMENTAL INVESTIGATIONS.

I. It was first attempted to reduce the nickel by coke in a small gas furnace, at the same time volatilizing the nickel-carbonyl which would distill out of the furnace. No reduction appeared to take place.

II. To reduce and partially fuse the ore in an arc furnace, using coke and an inclined hearth, presented itself as a method of some promise. The product when concentrated on a table with water gave a high nickel concentrate, but the electric furnace could not be made to operate satisfactorily, and was also very inefficient.

III. Several wet processes were tried; sulphuric acid on the roasted ore gave only fair extraction, with cost too high; heating with ammonium chloride instead of coke and then leaching with acid proved no better; heating with salt and leaching gave no better result; sodium sulphate acted the same, while chlorine gas, either dry or wet, did not decompose the silicates.

IV. A laboratory electric furnace of the Borchers type furnished results which agreed well with the larger commercial furnace as to the amount of ore which could be smelted and the metal to be obtained from unit quantity of electricity.

The first laboratory furnace used was too small for successful experiments, although it would run well for an hour or so. The shell was of sheet and angle iron, the interior lined first with asbestos, then burned lime, and the crucible itself cut out of a solid mass of magnesite. This furnace had a capacity of 100 amperes at 50 volts.

The larger laboratory furnace later built also had its shell of iron with angle irons and was 18 inches each way. Inside the iron an asbestos lining came first, and then fire brick. This left a crucible 13 inches deep by 8 inches in diameter. A graphite electrode, 6 inches in diameter, extended up about half-way to the bottom of the crucible, while coke, with pitch as binder, completed the bottom.

With between 350 and 450 amperes at about 50 volts this furnace treated 100 pounds of ore in 6 hours. After starting the arc over a little coke, ore would be fed in around the 2.5 inch movable electrode which extended down through the fire-brick cover of the furnace. It was found that tapping the furnace every 15 or 20 minutes gave the best results, for it obviated freeze-ups and slags sticky with carbides. The furnace becomes inefficient if run longer than necessary to produce the end products, and accumulating much metal in the bottom of the crucible raises the zone of intense activity, with consequent freezing of the metal.

The source of power was a gas engine made by the Springfield Gas Engine Co., single cylinder, 4-cycle, 24-inch stroke, 40 h. p. A 60 kw. D. C. generator, belt-connected, compound wound, supplied 480 amperes at 125 volts, but a high resistance introduced into the shunt field lowered the voltage to between 50 and 60 as suitable for such a furnace.

The high magnesium in the ore caused some difficulty at first, but it was later found that by mixing the hard green ore with the brown variety, which was high in iron and alumina, an excellent slag resulted.

Good working conditions gave slags within the following limits:

Silica	40.	to 50.	per cent
Alumina	30.	to 20.	per cent
Ferric oxide.....	2.	to .5	per cent
Calcium oxide.....	3.	to 10.	per cent
Magnesia	20.	to 15.	per cent
Nickel oxide.....	.4	to .8	per cent
Alkalies	4.6	to 8.7	per cent

There resulted considerable loss by volatilization, both of nickel, alkalies and magnesia; this amounted to from 15 to 20 per cent of the ore.

Comparatively little of the iron enters the slag. Most of it is reduced to metal and is useful in collecting the nickel.

It was noticed that lime and carbon seemed to have a particular effect on the breaking up of the silicates. Calcium carbide probably liberating silicon is considered as possible; on the other hand, too much carbide in the slag forebodes an approaching freeze. This is detected by the odor of acetylene about the cold slag.

The product of the smelting was always a silicide

*Abstracted by H. B. Pulsifer.

of iron and nickel, with some aluminum, calcium, chromium and traces of magnesium.

	Metal from—		
	Brown Ore.	Green Ore.	Mixtures.
Nickel	11.00	21.80	13.70
Iron	56.80	51.20	53.90
Silicon	30.10	18.80	27.40

To obtain metal with less silicon always resulted in a loss of more than 20 per cent of the nickel. This alloy is insoluble in hydrochloric, nitric, sulphuric, nitro-sulphuric acid and in aqua regia. It dissolves readily in nitric to which a few drops of hydrofluoric acid have been added.

V. The commercial smelter as built at the mine consisted of two generators with the steam engine in the power room, a boiler room, the furnace room with the electric furnace, a concentrating room with tables and ore bins.

Four 200 h. p. boilers supplied steam to the Hamilton-Corliss type engine which drove the generators with belts from a line shaft. The generators were each of the duplex compound wound type, capacity 170 kw., 375 r. p. m., 501 volts and 3,500 amperes.

The electric furnace was of the type like one used by Dr. Heroult at Sault Ste. Marie; a 1.5 inch cast iron plate set on porcelain insulators formed the base, iron rods extended up from this plate into a graphite-glucose mass which graded into a coke-glucose mixture for the bottom of the crucible. Five-eighths inch iron plate, strengthened with angle iron, formed the shell of the furnace. This was insulated from the cast iron base. It contained the fire-brick walls of the crucible and supported the cover of fire bricks clamped together. A lining of coke proved best to resist the chemical action about the walls of the crucible.

The electrode extended down through the cover sections, being regulated with a hand wheel from the floor. Built-up electrodes proved useless, and solid graphite, end-spliced, gave very good results.

Much trouble resulted from no facility for drying the ore. It made bad work in the furnace, hindered the operatives, and decreased the efficiency by from 6 to 8 per cent.

RECORD OF FURNACE RUN OF DECEMBER 9, 1909.

Charge No. 1.	Charge No. 2.	
4 tons brown ore.	3 tons brown ore.	
1.02 tons limestone.	0.6 ton limestone.	
0.48 ton coke.	0.48 ton coke.	
Moisture in ore, per cent.		42.50
Total dry ore treated, pounds.		8050
Net time of smelting, hours.		45.75
Limestone used, pounds.		3600
Coke used, pounds.		1940
Power used, average kilowatts.		170
Kilowatt hours used.		7755
Kilowatt years used.		0.896
Ore treated per kilowatt year.		9000
Ore treated per kilowatt day.		24.7
Nickel silicide obtained.		1242
Per cent of nickel in alloy.		11.90
Silicide obtained per kilowatt year, pounds.		1380
Silicide obtained per kilowatt day, pounds.		3.8

OPERATING COST PER 24 HOURS.

1 superintendent.	\$ 10.00
1 assistant.	2.50
2 oilers at \$1.80.	3.60
2 firemen at \$2.40.	4.80
2 assistant firemen at \$1.80.	3.60

2 head furnacemen at \$2.50.	5.00
4 assistant furnacemen at \$1.80.	7.20
2 miners at \$1.50.	3.00
1 blacksmith.	2.00
1 time keeper.	2.00
Coal, 18 tons, at \$4.50.	81.00
Machine oil.	4.00
Electrodes.	9.10
Flux and coke.	3.00
	\$140.80

Other runs were often better than this one given for December, and, as this average power cost is \$270 per kw. year for partial load only, it is figured that the cost for power should be only \$206 per kw. year under full load. It is also stated that the cost of hydro-electric power is only 14% of what this steam-generated power costs.

From the experience gained at this plant the author designs an electric crucible furnace which should work much better and more efficiently. Dryers are also proposed. Plans are given and some details as to cost of a 100-ton plant. The total estimated cost of such a plant is given as \$280,540. At a daily expense of \$885, 15 tons of silicide should be produced. This makes the cost per ton \$59.00.

The general conclusion is that a newly designed, larger, and thoroughly efficient plant will be required to make the exploitation of this mine a commercial success.

According to a report of the Ontario Provincial Bureau of Mines, the output of natural gas for the Province for 1909 was valued at \$1,188,179, while in 1910 the value of the gas produced rose to \$1,491,239, representing 7,263,427 thousand feet, worth an average of 20.5 cts. per 1,000 cu. ft. So far the western peninsula of Ontario has been the only section of the Province to yield a sufficient amount of gas to make the drilling of gas wells profitable. The older wells have been increasing the output, and a new field in the county of Elgin is under development. In the gas-producing region, which lies on the north and east shores of Lake Erie, the productive area has been steadily expanding. New districts are continually being developed, and it now appears likely that the whole of the Ontario side of Lake Erie will eventually prove productive gas territory. The three great producing fields are in Welland County, in Haldimand and Norfolk Counties, and in Essex and Kent Counties. The Essex-Kent fields had a considerable preponderance in yield, producing last year 3,841,234 thousand cubic feet. The Haldimand-Norfolk wells gave more than 1,000,000 thousand cubic feet less, and the Welland field about 2,000,000 thousand cubic feet less. The three fields have 828 wells, 982 miles of pipe, and employ 186 workmen, who were paid \$118,785. The greater yield of the Kent-Essex wells is apparent when it is considered that the Kent-Essex has but 47 wells, while Welland has 337 wells and Haldimand-Norfolk 444 wells.

CASE HARDENING OF STEEL.

A new process for the case hardening of steel and the results obtained from its use were described in a paper read at the October meeting of the Iron and Steel Institute at London, England. A recent issue of *The Iron Age* contains the following matter from the above paper:

The type of furnace which experience has proved to be best adapted to the case-hardening of small and medium-sized objects with the mixed case-hardening agent (granular carbon and carburizing gas) consists of cylindrical muffles arranged vertically and heated by producer gas. It is a double-muffle furnace provided with regenerators, of the kind constructed by C. M. Stein, of Paris, for the Sampierdarena Engineering Works, of G. Ansaldo, Armstrong & Co.

The parts of the apparatus particularly applied to the case-hardening process are represented in the drawing, which shows a section through the center of the vertical muffles of the furnace with the accessory apparatus completely mounted. Inside each of the muffles of refractory material is placed a cylindrical retort of mild steel, A, the external diameter of which is about 0.4 to 0.8 in. less than the internal diameter of the muffle. Mannesmann seamless tubes serve very well for making the retorts, as they are easily obtainable in stock sizes up to 14 in. diameter. For retorts of larger diameter, as in the case of the furnace here represented, tubes welded by the autogenous process are used.

The retort is supported on a bottom flange attached to a frame fixed to the brickwork of the furnace. The arrangement which serves to hold the retort, whether it is being supported by the bottom flange or by the upper flange, H, intended to receive the cover, is such as to enable one retort to be replaced by another in a few minutes. To the bottom flange is attached a cast-iron funnel, B, closed at the bottom by a non-return valve Q.

The steel nozzle C, admitting the carburizing gas, passes through an opening in the side of the funnel, provided with a good-sized boss. The gas, which, as has been previously stated, may consist of carbon monoxide or dioxide, in a pure state or mixed with other gases, is introduced through the screwed tube D, and through the nozzle C. This latter is connected to the hollow cast-steel dish E, intended to support the pieces to be case-hardened by means of the disk F, consisting of a steel plate covered with refractory material.

The carbon monoxide, after passing through the hole bored axially in the nozzle C, is delivered into a small distributing device placed inside the dish E, from which it passes to the case-hardening chamber A, through a number of holes provided in the disk F. During case-hardening the dish E is supported upon a series of projections on the same lower flange to which the funnel B is attached. This ring has thus to sustain

the entire weight of the dish E, the disk F, and any articles to be case-hardened which may be resting directly on the disk.

In the case of spur wheels, the maximum diameter of which might be 4 to 6 in. less than the internal diameter of the retort, the first thing to be done is to unscrew the tube D; then by means of the plunger L, actuated by the hydraulic cylinder I, the whole of the parts C, E and F are lifted together, the blunt conical end of the plunger engaging with these as it

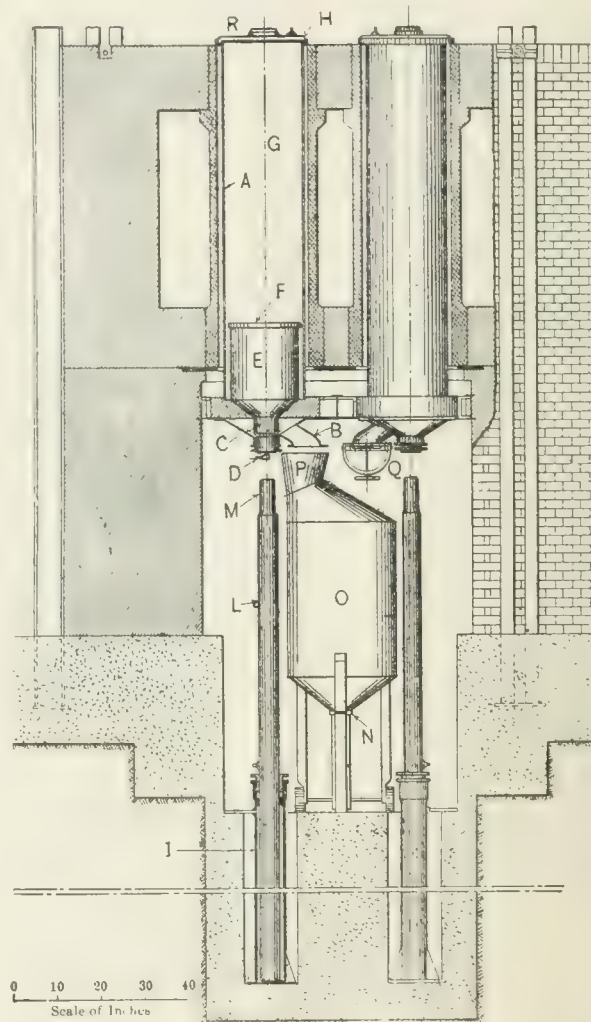


Fig. 1—Section of Vertical Case Hardening Furnace.

rises. The plunger is cooled internally with water circulation, to avoid heating it in case it should accidentally remain raised inside the retort for too long a period. When the piston has reached the end of its stroke, the upper surface of the disk being now only about 12 in. below the cover of the retort, the cover is removed, and the wheels to be case-hardened are piled horizontally one on top of the other upon the disk. As the wheels are inserted one by one, the apparatus carrying the weighted portions gradually sinks, and the charge is complete when the apparatus has reached the lowest position indicated in the figure. In that position the last wheel charged would be at least 12 in. below the upper edge of the retort.

Generally, the articles to be case-hardened are charged hot at a temperature of about 1,500 to 1,650° F., this being a matter of prime importance, as it affects most favorably both the quality of the product and the economy and regularity of the operation. For the preliminary heating of the pieces a small coal-fired muffle will serve, since the uniformity of temperature which can be obtained in such a furnace is quite sufficient for the purpose. If, however, the case-hardening furnace is not being worked to its fullest capacity, one of the vertical muffles may be used for the preliminary heating of the articles, while the case-hardening of the charge is proceeding in the other. In that case the best method of working the furnace is to use the muffles alternately, the one for case-hardening and the other for pre-heating the pieces.

As soon as the articles to be case-hardened have been inserted as described, the cover is placed on the retort and the bottom discharge pipe N, leading from the iron box O, is passed through the central opening in the cover. This box is full of the granular carbon still hot (generally at about 1,650° F.), which was discharged from the retort after use during the case-hardening operation last concluded. The box is easily swung above the muffles, being suspended by a tackle from a swinging arm. The outer walls of the box, notwithstanding that its contents of granular carbon are at a temperature of about 1,800° F., do not become heated above 400 to 500° F. during the time required for the loading and unloading of the retort, even when working under the most unfavorable conditions.

The box O having been adjusted as indicated, the butterfly valve which closes the lower end of the opening N is gradually opened, and in a few seconds the whole of the free space in the retort surrounding the pieces is filled with glowing granular carbon. The author has previously pointed out that the hot granular carbon employed by him for case-hardening with his mixed medium constitutes a mass which, particularly at high temperatures, is endowed with a mobility comparable to that of a liquid. On this account it is able to penetrate simply by its own weight into all crevices among the articles to be case-hardened, and between these and the walls of the retort. The filling up may be further assisted by means of iron rods introduced through holes in the cover and manipulated by the operator who stands on the furnace.

When the granular carbon has been filled in up to a level of $\frac{3}{4}$ or 1 in. below the upper edge of the retort the butterfly valve of the tube N is closed, the box O is lifted, the central opening of the cover is closed, the plunger L is lowered and the tube D is screwed into the nozzle C, through which the carbon monoxide or dioxide, pure or mixed with air, is admitted gradually to the distributor E and thence into the retort. In order to regulate the current of carbon monoxide an ordinary gas meter of the dry type is used. The case-hardening then proceeds without further trouble, the only thing that is necessary to be watched and regulated being the current of gas and the temperature.

For the observation of the temperature, an operation which, with the type of furnace described and when adjusted properly, does not often require to be made, an ordinary thermo-electric pyrometer is used by passing it through any one of the holes in the cover from time to time and inserting it into various parts of the mass of granular carbon. The temperature should be maintained quite constantly at a fixed point between 1,650 and 2,000° F.

The gas issuing from the retort through the pipe fixed in the cover can be collected in a gasometer, and after suitable regeneration it will serve for further heats. The necessary length of the heating period can be determined with the greatest precision, and is dependent on the result which it is required to obtain. When the heating has proceeded long enough the current of carbon monoxide is shut off, the tube D is removed and the discharge of the retort is undertaken. Where results of a very special nature are not aimed at, the discharging operation begins by withdrawing the whole of the granular carbon at once, and removing it through the non-return valve Q in the box O, now placed underneath.

Where, however, exceptionally graduated case-hardening is required the specific action of the carbon monoxide is isolated during the last phase of the case-hardening from the direct action of the granular carbon. A quantity of the granulated carbon is drawn out from the retort. Only those parts of the case-hardened pieces are uncovered in which a well-graduated case-hardening is required. Sufficient solid carbon is retained in the retort to ensure the desired chemical equilibrium of the gas. The remainder of the carbon is removed from the retort after the second phase of the case-hardening with isolated carbon monoxide has proceeded long enough to attain the desired results.

In almost all cases occurring in the practical production of case-hardened pieces for machine construction, in which the carburized zone is seldom required to exceed 0.08 in. in thickness, the period of case-hardening never exceeds 2 hours, and may even be reduced to 1 hour, the time being reckoned from the moment of completing the charging operation to that at which discharging begins. It therefore includes the time necessary for bringing the temperature up to working point, which required not more than 10 min., owing to the fact that both the solid carbon and the pieces are already hot when charged.

The foregoing particulars show that in practice, under ordinary working conditions, the time for a complete heat is less than 2½ hours, and since the weight of a charge, which retorts of the size shown in the accompanying sketch can conveniently hold, varies from a minimum of 220 lbs. in exceptionally unfavorable cases to a maximum of over 1,100 lbs., it will be seen that the productive capacity of each retort ranges from a minimum of 1 ton to a maximum of 5 tons of case-hardened steel pieces per 24 hours.

The furnace with vertical muffles has distinct advantages in the speed of working for a given charge, simplicity of operating, and in the uniformity of the treatment of all the pieces forming the charge in the muffle. The clearance spaces left in the retort are reduced to a minimum.

described. The material used was an ordinary mild steel of the following composition: Carbon, 0.12 per cent; silicon, 0.06 per cent; manganese, 0.47 per cent; sulphur, 0.02 per cent; phosphorus, 0.03 per cent.

(A) CASE HARDENING FOR 10 HOURS AT 2,000° F. WITH THE

The numerical data contained in the three tables relating to the first series are graphically represented in Fig. 2.

MIXED AGENT.

No. of the range of single layers analyzed, in.....	Distance from the surface of the central portion of the layer analyzed, in.....	Amount of carbon in the single layers.		
		First determination, per cent.....	Second determination, per cent.....	Mean, per cent.....
1	0.02	1.16	1.18	1.17
5	0.10	0.80	0.82	0.81
10	0.195	0.33	0.35	0.34

Both in the first series of experiments, as well as in the succeeding series, similar pieces of steel were subjected simultaneously to each of the various treatments and under identical conditions, one piece of which was used after each operation for the purpose of making a quantitative determination of the carbon in the successive layers.

(B) AFTER REHEATING THE SAME CARBURIZED ZONE FOR 5 HOURS AT 2,000° F. IN ISOLATED CARBON MONOXIDE.

No. of the range of single layers analyzed, in.....	Distance from the surface of the central portion of the layer analyzed, in.....	Amount of carbon in single layers.		
		First determination, per cent.....	Second determination, per cent.....	Mean, per cent.....
1	0.02	0.84	0.86	0.85
5	0.10	0.76	0.80	0.78
10	0.195	0.46	0.46	0.46
15	0.29	0.23	0.27	0.25

(C) AFTER REHEATING SAME ZONE FOR ANOTHER 5 HOURS (ALTOGETHER 10 HOURS OF REHEATING) AT 2,000° F. IN ISOLATED CARBON MONOXIDE.

No. of the range of single layers analyzed, in.....	Distance from the surface of the central portion of the layer analyzed, in.....	Amount of carbon in single layers.		
		First determination, per cent.....	Second determination, per cent.....	Mean, per cent.....
1	0.02	0.69	0.71	0.70
5	0.10	0.66	0.68	0.67
10	0.195	0.52	0.54	0.53
15	0.29	0.38	0.40	0.39

In the accompanying tables are collated the results of estimations of the carbon in successive layers of a thickness of 0.02 in., obtained by case-hardening with the mixed agent, followed by exposure to the action of carbon monoxide isolated in the manner previously

(A) CASE HARDENING FOR 10 HOURS AT 2,000° F. WITH THE MIXED AGENT.

No. of the range of single layers analyzed, in.....	Distance from the surface of the central portion of the layer analyzed, in.....	Amount of carbon in the single layers.		
		First determination, per cent.....	Second determination, per cent.....	Mean, per cent.....
1	0.02	1.18	1.14	1.16
5	0.10	0.79	0.82	0.81
10	0.195	0.49	0.51	0.50

(B) AFTER REHEATING THE SAME CARBURIZED ZONE FOR 5 HOURS AT 2,000° F. IN ISOLATED CARBON MONOXIDE.

No. of the range of single layers analyzed, in.....	Distance from the surface of the central portion of the layer analyzed, in.....	Amount of carbon in the single layers.		
		First determination, per cent.....	Second determination, per cent.....	Mean, per cent.....
1	0.02	0.82	0.81	0.80
5	0.10	0.76	0.78	0.77
10	0.195	0.58	0.58	0.58
15	0.29	0.45	0.44	0.45

(C) AFTER REHEATING THE SAME ZONES FOR A FURTHER 5 HOURS (TOTAL REHEATING PERIOD 10 HOURS) AT 2,000° F. IN ISOLATED CARBON MONOXIDE.

No. of the range of single layers analyzed, in.....	Distance from the surface of the central portion of the layer analyzed, in.....	Amount of carbon in the single layers.		
		First determination, per cent.....	Second determination, per cent.....	Mean, per cent.....
1	0.02	0.87	0.85	0.86
5	0.10	0.85	0.88	0.87
10	0.195	0.75	0.75	0.75
15	0.29	0.60	0.58	0.59

In the second series the material used was a chromium nickel steel of the composition here given. This is a steel of the kind ordinarily used for Krupp armor-plates. The author proposes later to present particulars relating to case-hardening to considerable depths as carried out upon this steel by the process described in the paper. The composition is: Carbon, 0.33 per cent; silicon, 0.06 per cent; manganese, 1.15 per cent;

sulphur, 0.02 per cent; phosphorus, 0.015 per cent; chromium, 1.50 per cent; nickel, 3.17 per cent.

The numerical data contained in the three last tables, relating to the second series, are graphically represented in Fig. 3.

The principal causes of brittleness and peeling, which always occur in a greater or less degree in steels

VANADIUM IN NEW MEXICO.

In 1884, F. M. Endlich, manager of the Lake Valley mines, which lie in the southwestern portion of Sierra county, New Mexico, and were famous producers of silver ore to the value of over \$14,000,000, discovered in the lead ore of these properties, small yellow crystals which upon examination proved to consist of lead, arsenic, vanadium and oxygen. In honor of the discoverer, states Brigham Leatherbee in *The Mining Magazine*, this combination was named Endlichite, and has been so classed by Dana and subsequent mineralogists. During 1896 or 1897, William F. Hall discovered the same variety of vanadium ore in the lead and silver mines he was operating about twenty miles north of Lake Valley, and which have since become the property of the Vanadium Queen Mining Co. As there was no commercial demand for vanadium at that time, Mr. Hall preserved only the finest crystals, which he sent as specimens to a large mineral company in Philadelphia, by whom they were distributed to the various public and private mineral collections throughout Europe and America.

During the winter of 1909, vanadium was discovered on the properties of the Southwestern Lead & Coal Co. on the eastern slope of the Sierra de los Caballos, across the Rio Grande and 40 miles east of the other deposits. This property was then taken over by the Vanadium Mines Co., which is now operating it. The property of the Vanadium Queen Mining Company is situated in the Las Animas district, which has heretofore been considered a gold district, as but little else has been worked here, although some silver, lead and manganese have been mined. The total gold production from both quartz lodes and placer workings is recorded at over \$9,000,000. The vanadium ore occurs in a well defined contact vein, between the shale and lime, which shows numerous outcrops for over three miles; a number of prospecting shafts, open-cuts and tunnels indicate that there is a continuous ore body for this entire distance. The vanadium is found in different combinations, endlichite and vanadinite predominating, and the assays run from 3.25 to 99 per cent in metallic vanadium. So far the management has devoted its entire attention to prospecting along the vein and to opening up the most promising portions, with the intention of having the ore well blocked out before commencing actual operations. In the meantime, constant experiments upon the treatment and reduction of ores have been under way in various metallurgical laboratories, and it is presumed that shortly the first unit of an experimental plant will be installed upon these properties.

The Vanadium Mines Co. controls two fissure veins of vanadium-bearing ore, one of which has been prospected by a 50 ft. shaft, while the other has been more extensively developed and now has the main shaft down below 200 ft. with various levels. The ore is hauled from the mine to the old concentrating mill of

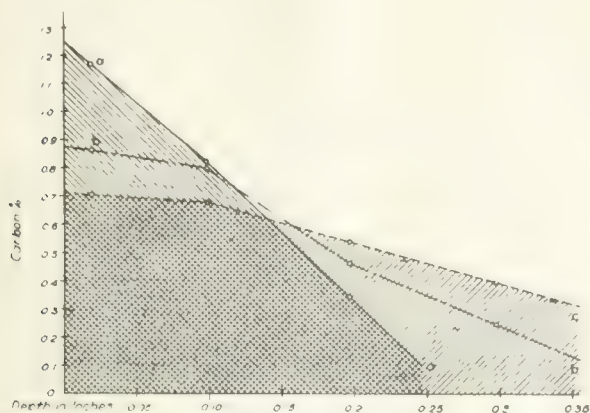


Fig. 2—Diagram Showing Effect of Case Hardening.

case hardened by the ordinary process, are excessive concentration of carbon in the superficial zones of the case-hardened pieces, and the rapid diminution of the carbon in passing to the deeper layers. The ability to alter the form of the curves of case-hardening, as

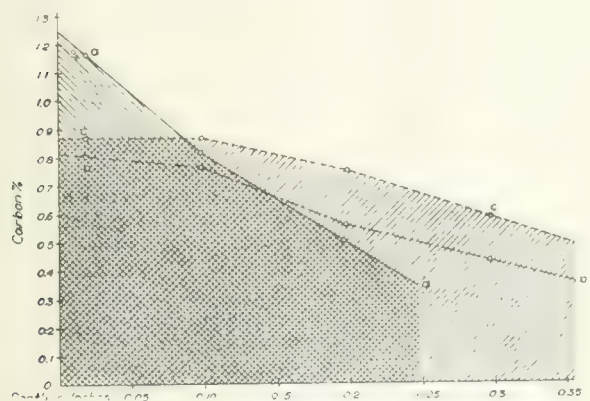


Fig. 3—Diagram Showing Effect of Case Hardening.

shown in the diagrams, suggests that case-hardening can also be successfully applied to those special steels on the exterior of which the ordinary process of case-hardening yields carburized zones, which are practically useless on account of their excessive brittleness.

An important factor in the development of gas heating in the Sheffield, England, trades has been the cheapness in the local supply of that commodity. For industrial purposes the manufacturers can, in fact, command the cheapest gas in the country. For engines the price is only 22 cts. per 1,000 cu. ft., and for other purposes 30 cts., with a reduction to 26 cts. if the annual consumption exceeds 500,000 cu. ft. Any consumption exceeding the latter total is charged at 22 cts.

the defunct coal company, which mill has been enlarged by a set of Cornish rolls and some new vanners and slime tables. This concentration is being performed as experimental work, the foundations having been laid for a new and permanent plant to be equipped with the latest patterns of the best machinery, as soon as the type of equipment best suited for the ore has been selected. After concentration, the concentrate, which assays about 8.632 per cent vanadium and 59 per cent lead, is transported by mule teams to Cutter, where a large oxide plant has been erected on the Santa Fe railway. The product from this plant is shipped to Rankin, Pa., where it is made into ferro-vanadium alloy.

Prospecting is being done elsewhere in the county for vanadium and there are many rumors afloat regarding the discovery of valuable deposits.

AN EXPERIMENT WITH VERTICAL RETORTS FOR COAL GAS MANUFACTURE.*

A bench of vertical retorts has been installed at the La Grange Station of the Western United Gas and Electric Co. of Aurora, Ill.

Mr. J. S. De Hart describes the same as follows:

The bench consists of 4 retorts, 25 ft. long, oval in section, 9 ins. by 21 ins. at the top, 20 ins. by 29 ins. at the bottom, heated by a producer with twin grates and twin charging holes, all supported on steel platform, 10 ft. above grade. The upper mouthpieces are fitted with separate feeding drums, each operated at a speed to keep the retorts full of coal. The lower mouthpieces of each retort are fitted with a coke extractor and revolving drum. The speed of the coke extractor controls the carbonization in each retort. The coal feeding drums make a $\frac{3}{4}$ revolution a minute, the coke extractors make a $\frac{3}{4}$ revolution an hour and the drums discharge 4 times an hour.

The bench is located at one end of the retort house and is run in connection with horizontal benches, or alone, as the gas supply will allow. Retort house governors control the pressure for each system. Coal is passed through a small crusher and is raised by an elevator to a 24-hour storage bin above the retorts. The coke is taken away from the bench in wheelbarrows. A small vertical engine runs the coal feeding drums and extractors.

The heating of the bench is very simple. All the producer gas goes directly to the top of the combustion chambers, where it is distributed by 4 dampers. The furnaces have angle iron grate bars and step grates for the supply of water; but, in addition to the above, steam is used and measured by disks and regulating valve. The furnaces are cleaned once each 48 hours and are practically free from clinkers. Secondary air passes through a recuperator in the back wall of the bench, across the bottom, and is distributed to the top

and middle of the combustion chambers by 4 dampers. When the proper distribution is obtained the main stack damper controls the heats. All the firebrick material, with the exception of 3 retorts, was made in this country. The 3 retorts are machine made and imported from England.

The bench was completed about September 1, 1909, and was operated until February 1, 1910, the greater part of the time in conjunction with the old benches. The bench was not protected from the weather until late in the winter and the coal available was taken directly from the cars in a more or less unsatisfactory condition at times. In November it was possible to shut down the horizontal benches for 10 days, during which time the following average results were obtained from Youghiogeny coal:

Coal used, pounds	20,151.00
Coke made, pounds	14,841.00
Gas made, cubic feet	108,637.00
Generator fuel, pounds	3,598.00
Yield gas per pound coal	5.39
Coke yield per cent coal	73.6
Generator fuel per pounds coal	17.89
Generator fuel per cent coke made	24.30
Candle power on Sugg Argand, D.	14.47
B. T. U.'s gross	609.00
Candle feet	77.91

The coal feeding drums were found to be unsatisfactory and cast iron hoppers with self-sealing doors were substituted.

These proved very simple to operate and gave no trouble.

The high volatile coal used developed a tendency to bridge at the top of the retort.

Experiments indicated a yield of 7 lbs. of ammonia and $14\frac{1}{2}$ gals. of tar per ton of coal carbonized.

According to the London Times a British oxygen company has decided to erect an oxygen factory in Sheffield, where it is considered the growing demand for oxygen in connection with metal cutting warrants this extension. When the Sheffield plant is completed the company, which in 1908 had only three factories (London, Birmingham and Manchester), will possess eight factories in the United Kingdom, all situated in centers where the demand for oxygen is important—at London (Westminster and Greenwich), Birmingham, Cardiff, Manchester, Sheffield, Newcastle and Glasgow. These extensions are entirely due to the demand for cheap oxygen in connection with oxy-acetylene welding and oxygen metal cutting. In view of the heavy freight charges on gas cylinders conveyed by rail, it is obvious that local sources of supply must tend to reduce the price of oxygen to the consumer. All the company's plants are of the modern liquid-air type, producing oxygen of a high degree of purity, and when the Sheffield plant is in operation their total output will be about 300,000 cu. ft. a day. This supply is largely in excess of the present demand, but it is anticipated that this quantity may ultimately be required if the use of oxygen for metal cutting extends.

*From "Light-Heat-Power."

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NOTES AND COMMENTS.

Potash in the United States.

The U. S. Geological Survey has recently issued a short report on the subject of potash deposits in this country. The material contained in this report has been read with considerable interest by American manufacturers, as all of them are very anxious to secure a supply other than that at present available only in Germany.

The Geological Survey has no very definite information to give as to deposits in this country. Their field work thus far has consisted in tabulating data already available as to previous investigations and possibilities of deposits. They mention five general sources which will probably receive investigation before their work is completed. Of these five the only one which seems to be of any particular importance is saline deposits which may be found in the arid lands of the West, and it is this source which has received the bulk of their time thus far. They report that their early investigations indicated that the assumption of

the existence of beds of potash soils in the Laramie basin in southern Wyoming is logical, but the data at hand which are fairly complete prove to be chiefly of negative character, so far as revealing any unusual amounts of potash is concerned.

Basing their next investigations on the early geological work of Gilbert and Russell who describe in detail the lakes, Bonneville and Lahontan, they next started drilling in Nevada. These operations commenced early in October and the well thus far has reached a depth of a little over 300 feet. Thus far evidence of saline deposits has not been discovered, but it is expected that they will continue drilling until a depth of at least 1,000 feet is reached. It will thus be noted that the investigations of the Geological Survey have thus far presented no conclusive evidence of there being workable potash deposits in this country. The reports that such beds have been found in considerable quantities seems to have been very highly colored by the desire of American manufacturers to secure such deposits.

Facilities for Industrial Research.

There was published in the November number of the Journal of Industrial and Engineering Chemistry an editorial on this subject which contains so much valuable material that we venture to present some of the statements made to our readers.

"Much has been said and written of late about the man qualified by training and experience to conduct industrial research but what would seem to be a matter of nearly or even greater importance is the question of the facilities available for the study and solution of these industrial problems. Industrial research is the great basic principle upon which we develop our manufacturing and engineering industries. The results of this research must not only precede the establishment of manufacturing enterprises but such study must continue to accompany, supervise and direct any manufacturing operations which assume to be intelligently managed."

Then after commenting on the fact that many plants will operate for a time with makeshift methods and unscientific management but that they are finally overtaken in their errors and forced out by their more progressive rivals, the author proceeds to say:

"There are times when the whole future of a business depends upon selecting the right process, method or machine to prepare a piece of work. Some managers, however, will be contented under such a stress to render a snap decision. Others will make a few cut and try experiments, others will spend much money to find out what the other fellow is doing or using or send a man to Europe to pick up a few ideas."

It is seen by a close survey of the chemical industries of this country that the firms who have been successful have not been firms which depend on this method of solving their problems. There is practically

no large manufacturing company in this country that has been uniformly successful and which does not maintain a department for industrial research. The fact that much equipment must be provided by each of these firms and that much of the work done by one is duplicated in that done by another, brings out very clearly the saving in time and energy which would come if we had some well equipped central research body which could investigate at least all of the fundamental problems and put the reports before manufacturers in such a way that they would be available for all. The author of the article to which we have referred makes this comment:

"A great scholar once said to me that he did not see why it takes so much expensive apparatus and equipment to do chemical research work now when some of the most brilliant work in former years have been done with cigar boxes and a few glass bottles and other wholly inexpensive appliances."

The comparison is well made between the early mining methods when it was possible for any one familiar with the geology of a country to go out and after sufficient investigation locate a profitable mining plain, but when the field has been well covered it takes much time, labor and money to discover a mining plain which is profitable and which is yet open to be taken. The same condition prevails in chemical industries. In the early days, it was possible to use very crude methods and to use very crude apparatus to make discoveries of considerable importance, but at present when the field has been thoroughly prospected it takes an outlay of time, money and brains to evolve anything new or to introduce any improvements in old processes.

"The laboratory requirements for industrial study are no more exacting, expensive or difficult to develop than those of mechanical, electric or metallurgical engineering laboratories and those of the other engineering subjects. It seems to me that we chemists have not developed our facilities while other engineers have kept pace with the advancement of their respective industries."

The writer then gives his experience in visiting various engineering laboratories in Europe and contrasts the equipment found in the other branches of engineering with that found devoted to the instruction of chemical engineering. He could also have extended this observation to the schools in this country, with the same result. There are many schools having first class equipment for instructional work in other engineering subjects while there are practically none with anything worthy of the name of laboratory equipment for chemical engineering. If the profession of chemical engineering is to advance in this country, we must have increased facilities for carrying on experimental work in industrial chemical problems. As the author of the editorial in question has well said:

"Laboratories dissolve problems and establish facts not from the standpoint of fundamental theories and

principles involved which presumably have been established by the test-tube and beaker method but to solve problems from the standpoint of their proposed application."

Six kinds of seaweed are used in the construction of Japanese isinglass, or agar-agar, the method of manufacture being described as follows in a recent U. S. Consular report: The seaweed is first crushed, each kind separately, to remove shells or other adhering matter, and then washed clean with water. The washed seaweed is placed on a mat and dried until its color becomes white by the action of the sun, frost, and dew. This operation takes place during September and October, and when bleached the weight of the seaweed is decreased nearly one-half. After bleaching, the six kinds of seaweed—in the proportion of Izu, 4; Egokusa, 4; Misaki, 3; Hirakusa, 3; Nanbu, 4; and Onikusa, 2—are all put together in a boiler and cooked for about 14 hours, until they have become soluble. The liquid is then strained through a sack and a box with a bamboo sieve on one side, from which it runs into a container. From the container the liquid is ladled into trays about $3\frac{1}{2}$ feet long and 3 inches deep. After remaining in the trays about 12 hours these are placed on a low stand, and the isinglass is cut into strips 3 inches wide and 14 inches long, with a knife and ruler. These strips are then put into a long closed wooden box (the ends of which are 3 inches square, one end being open and one filled in with a wire sieve) and pushed through the sieve end in the form of long fine strips. The isinglass is then placed on a low stand, which is covered with a clean mat, and dried in the sun during the day and frozen during the night for two or three weeks during January and February, being watered at midnight. The quality of the isinglass depends upon the weather during this time, the best being made when it is clear and cold, the poorest when it is warm and rainy. When the isinglass is bleached sufficiently, it is compressed and packed in Japanese matting tied with a straw rope.

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